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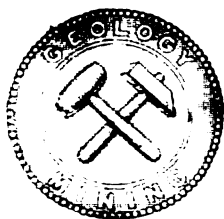
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JANUARY, 1917

Bulletin of the American Institute of Mining Engineers

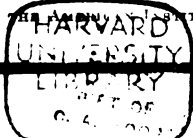
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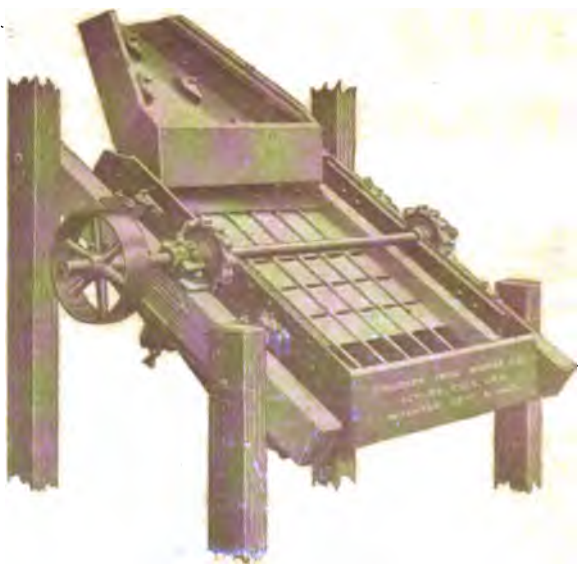
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The Impact Screen

has demonstrated what an immense amount of material can be passed through a relatively small screen area. The "impact," perpendicular to the screen surface, accomplishes its extremely high efficiency by the simultaneous attainment of two most desirable conditions. Firstly, it keeps the meshes open in a surprisingly perfect manner, and secondly, the vibration causes the coarse to work up through the bed of material passing over the screen, and the fine to settle down against the screen cloth, where it has an opportunity to pass through.



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Pamphlet 9-B Describes it Fully

Colorado Iron Works Co.

Established 1860

Denver, Colo.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 121

JANUARY

1917

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY SROUGHTON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$12 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

NEW YORK MEETING, FEBRUARY 19 TO 22, 1917

The New York Meeting this year will be held from Monday, February 19 to Thursday, February 22. The Committee on Arrangements, the personnel of which is given elsewhere in this *Bulletin*, is making plans to duplicate the conspicuous success of last year's meeting in New York. The papers to be read at this meeting, it is believed, furnish material for interesting and profitable technical sessions and symposiums. The social features are being carefully planned since it is believed that these form an important part of such meetings; several functions are to be given with the object of fostering the get-together, know-each-other-better spirit. Ladies are invited to attend this meeting and participate in the various affairs planned by the Ladies' Committee and by the Committee on Arrangements. This preliminary notice is given in order that members may make their plans now to be in New York at the time of the meeting. Full details will be sent to the membership through the mails as soon as the program has been definitely fixed.

NOMINATIONS FOR OFFICERS AND DIRECTORS

The Committee on Nominations begs to submit the following names as its nominees for the respective offices indicated:

For President, SIDNEY J. JENNINGS, New York.

For Vice-Presidents, C. W. GOODALE, Butte, Mont.; MARK L. REQUA, San Francisco.*

For Directors, J. E. JOHNSON, JR., New York; ALLEN H. ROGERS, Boston; HOWARD N. EAVENSON, West Virginia; ROBERT M. RAYMOND, New York; WILLET G. MILLER, Canada.

The nomination for President has attracted the attention of a large number of members, as is evidenced by the receipt of many hundreds of letters and post cards. Most of these favored either Sidney J. Jennings or Philip N. Moore, each of whom received many cordial endorsements. Such a wide interest on the part of the members is a very wholesome sign, but where there is so much activity it is natural to expect a certain amount of criticism. Some of the letters indicate that the authors feel that the nomination of President should be left entirely to the Committee on Nominations without any effort by the members at large to influence them. Others feel that the nominations are largely controlled by a small group and that the members at large can have little influence. As a result of the experience of the last few months, a majority of the Committee on Nominations begs leave to suggest that the method of making nominations be changed so that two candidates, at least, for the various offices be nominated, and suggests that the Directors appoint a committee to investigate this question thoroughly with a view to making definite recommendations. In accordance with its opinion that the best interests of the Institute would be served by a change in the Constitution, allowing the nomination of two candidates, and in view of the fact that this Committee has no authority to make more than a single nomination for each office, considering the high standing of both Mr. Jennings and Mr. Moore and the large number of members favoring each of these gentlemen, this Committee respectfully suggests the desirability of the immediate nomination of Philip N. Moore by a petition of 25 members in accordance with the present Constitution. This Committee regrets its inability to make two nominations.

Respectfully submitted,

S. W. MUDD, *Chairman.*

New York, November 29, 1916.

MR. BRADLEY STOUGHTON, *Secretary,*

American Institute of Mining Engineers,

29 West 39th Street, New York City.

Dear Sir:—

In accordance with Article VII of the Constitution, we, the undersigned, herewith nominate for the office of President of the American Institute of Mining Engineers at the approaching election, Mr. Philip N. Moore, of St. Louis.

Edmund B. Kirby,

J. Parke Channing,

Geo. C. Stone,

J. A. Van Mater,

E. Gybbon Spilsbury,

E. E. Olcott,

Hugh Rose,

C. P. Perin

Frank W. DeWolf,

H. H. Stoek,

E. A. Holbrook,

C. M. Young,

H. A. Buehler,

W. E. McCourt,

Eugene McAuliffe,

Herman Garlichs

* In place of Walter H. Aldridge, declined.

H. E. Judd,
S. M. Marshall,
W. S. Harper,
H. Kenyon Burch,
Robert H. Richards,
Henry M. Howe,
H. M. Chance,

Chas. Y. Clayton,
H. A. Wheeler,
A. L. McRae,
V. H. McNutt,
C. R. Forbes,
H. T. Mann,
E. V. Matlack.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, NOV. 24, 1916

The sum of \$250 was appropriated to the Chicago Section.

The President was authorized to appoint five members of the Board of Directors, including himself as Chairman, to discuss with other technical societies the proper organization for handling all matters of joint interest to the profession of engineering as distinct from purely technical activities, and to submit its findings in the form of recommendations to the Board of Directors.

Charles F. Rand was appointed a Trustee of the United Engineering Society to succeed himself.

John H. Janeway was elected a member of the Library Board, vice Karl Eilers, resigned.

Alexander C. Humphreys was elected a member of the Library Board to succeed himself.

A resolution was passed regretting that the language of the message sent on June 23, 1916, to the Chairman of the International Engineering Congress, by authority of the Board of Directors, was considered unwarrantably mandatory in tone and assuring the San Francisco Section that there was no intent on its part to arbitrarily censor, or cause to be censored, Mr. Shockley's paper, and the San Francisco Section was referred to the letter of President Ricketts to C. W. Merrill, dated Nov. 25, 1916, which was approved.

Upon the recommendation of the Committee on Papers and Publications it was resolved that an Index to Volumes 36 to 55 be published by the Institute and sold to the members.

It was resolved that the Secretary of the Institute be instructed to confer with the secretaries of the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers, American Chemical Society, and the Mining and Metallurgical Society of America in an endeavor to interest these societies in a joint participation with the Institute in the dedication of the new buildings of the Bureau of Mines at Pittsburgh, Pa., during some part of September, 1917.

MEETING OF AUTOMOBILE ENGINEERS

The Society of Automobile Engineers is planning to hold its Annual Meeting on Jan. 11, 1917. The Society extends a cordial invitation to the members of the Institute to attend a technical session on the afternoon of that day, in the auditorium of the Engineering Societies Building, and to take part in the proceedings.

Papers are being prepared on the subjects of high-speed engines, dynamics of the automobile and electrical equipment. It is also expected that important papers will be presented on aerial navigation over water, and the use of motor boats in war. Captain V. E. Clark of the Signal Corps is to give a paper on the requirements of aeroplane design particularly as regards the experience of the Army in using aeroplanes in Mexico

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period Nov. 10, 1916 to Dec. 10, 1916:

Chester Atkinson, La Fundicion, Peru.
 F. J. Brule, Toronto, Ont., Canada.
 Sterling H. Bunnell, New York, N. Y.
 G. H. Cunningham, Hobart, Tasmania.
 Courtenay DeKalb, Tucson, Ariz.
 E. H. Dickenson, New York, N. Y.
 E. B. Durham, Mount Kisco, N. Y.
 L. B. Eames, Pueblo, Colo.
 N. H. Emmons, 2d, Copperhill, Tenn.
 R. E. Hoffman, Hannibal, Mo.
 B. B. Hood, Chrome, N. J.
 H. E. Jackman, Rochester, N. Y.
 F. H. Jackson, Joplin, Mo.
 Knight S. Jordan, Provo, Utah.
 William Kelly, Vulcan, Mich.
 W. H. Lanagan, San Francisco, Cal.
 H. H. Lauer, Northampton, Pa.

Paul Llewellyn, East Chicago, Ind.
 Harry March, London, England.
 E. P. Mathewson, Toronto, Ont., Canada.
 E. W. Moore, Budapest, Rumania.
 W. J. Pentland, Guadalajara, Mex.
 Heinrich Ries, Ithaca, N. Y.
 W. E. Saunders, Philadelphia, Pa.
 J. G. Scrugham, Reno, Nev.
 W. H. Seagrave, Kennecott, Alaska.
 H. B. Small, Magangue, Colombia.
 W. H. Staver, Washington, D. C.
 R. K. Stockwell, Santiago, Chile.
 Warren D. Thompson, Boston, Mass.
 Thor Warner, Kingman, Ariz.
 S. T. Wellman, Cleveland, Ohio.
 F. H. Willcox, Pittsburgh, Pa.

Arthur K. Adams has been appointed geologist to the Andes Copper Co.

William D. Blackmer, of Los Angeles, has been appointed manager for the Northern Ore Co., Edwards, N. Y.

Fred J. Brule has been appointed chief engineer for the British-America Nickel Corporation.

Stanley C. Bullock, late of Chile, who returned to England to serve in the army, has received promotion to the rank of Captain in the Royal Engineers.

W. A. Carlyle is resident director in Ontario for the British-America Nickel Corporation.

Thomas F. Cole has resigned the presidency and directorate of the Greene-Cananea Copper Co., and the Board of the Inspiration Consolidated Copper Co. Mr. Cole has held the presidency of the Greene-Cananea Co. since its organization, eight years ago.

W. D. Thornton succeeds Mr. Cole as president of the Greene-Cananea Copper Co.

H. H. Colley has been promoted to the superintendency of the Old Dominion mill and smelter at Globe, Ariz.

Henry F. Collins has resigned the executive management of the Huelva Copper and Sulphur Mines to accept the position of consulting engineer for the same company.

Frank R. Corwin has resigned as assistant superintendent of the Consolidated Arizona Smelting Co., to accept a position as assistant superintendent of the International Smelting Co., at Miami, Arizona.

Justice Grugan resigned on Dec. 1 his post as manager for the Northern Ore Co., Edwards, N.Y. He will establish offices as consulting engineer at Room 621, 30 Church Street, New York.

Arthur Jarman has resigned as assistant superintendent of the Grand Junction mine, Waihi, N. Z.

H. D. Kinney has become associated with the Mines Operating Co., at Butte, Mont.

Benjamin B. Lawrence has been appointed a life trustee of Columbia University.

Adolph Lewisohn has been elected President of the Tennessee Copper Co., succeeding Utley Wedge.

Charles E. Mills, general manager of the Inspiration Consolidated Copper Co., has been appointed managing director of the Cananea Consolidated, Sonora, Mexico.

J. D. Sperr is now engaged as consulting and managing engineer for the United Verde Consolidated Copper Co., and as engineer and geologist with J. P. Wilson & Co., Exploration Engineers.

Arthur W. Stevens is on the staff of the Dutch Sweeney mines, Quartz, Tuolumne Co., Cal.

M. B. Stewart has accepted a position as mining engineer for the Lumoghi Coal Co., at Collinsville, Ill.

W. W. Wishon has been appointed consulting engineer for the Big Casino Min. Co., near Searchlight, Nev.

POSITIONS VACANT

No. 175. At a mine in Missouri: A general designer on timber work for concentrator power house and general mill work; two draftsmen on all kinds of mine and mill work; two tracers, good letterer and map tracer; one transit man for surveys who can plot his work accurately.

No. 177. The University of Illinois maintains 14 Engineering Experiment Station Research Fellowships, and one is under the patronage of the Illinois Gas Association. Each carries \$500 per annum. Appointments must be accepted for two consecutive collegiate years. Degree of Master of Science will be conferred if all requirements have been met. Applications must reach the Director of the Station at Urbana, Ill., not later than Feb. 1, 1917.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Member, technical graduate, married. Age, 36. Experience in copper, lead and zinc mining. Ten years' executive experience as superintendent. Thorough experience in churn drilling and general development work. Desires position as superintendent or examining engineer. No. 338.

Member, Colorado School of Mines' graduate, married. Two years' Mexican experience in gold and silver; three years' mining and milling experience of natural abrasives in the United States. Speaks Spanish. Interview in New York or Boston. No. 339.

LOCAL SECTION NEWS

BOSTON SECTION

W. E. C. EUSTIS, *Chairman*,

R. L. AGASSIZ, *Vice-Chairman*,

E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology,
Boston, Mass.

ALBERT SAUVEUR,

H. L. SMYTH.

The 37th meeting of the Boston Section was held at the Engineers' Club, on Monday evening, Oct. 2, 1916.

Fourteen members and guests attended the dinner. The Society was fortunate in having present not only the announced speaker, J. G. Carleton of Marblehead, Mass., but also a brother member from afar, Thomas Warner of California.

Mr. Carleton, who has spent the past few years in South America, gave an informal talk on the mining of the precious metals on the South American continent. He drew a comparison of the geographical distribution of the precious metals in North and South America and pointed out the similarity in the geological occurrence of the eastern mining regions, mentioning particularly Venezuela and Brazil. He also directed attention to the fact that the great placers of the northwestern part of North America had their counterpart in the gold-bearing region of Columbia. The marked contrast between the southern end of North America, so rich in metals, and the southern part of South America, so poor in mineral deposits, was clearly brought out.

It is difficult to avoid the conclusion that the South American Cordillera is less intensely mineralized than the corresponding mountain

chains in North America. There are comparatively few metamorphic deposits, and very few high temperature veins like those of Goldfield and Cripple Creek. Of the latter type, mention was made of Guanaco, Vaquillas, Sierra Overa, and Andacollo, as very disappointing examples of deposits found in rhyolites and allied rocks.

The region that is undoubtedly the most promising is that of the Cordillera of Bolivia, with its rich mineralization of tin, silver and gold. Mr. Carleton gave an account of the Huanchaca mine, with which he was familiar, and Potosi, which mine is stated to have yielded 50,000,000 kg. of silver from 1553 to 1910. Huanchaca produced \$50,000,000 between 1873 and 1888. Bolivia yields, at present, about 80,000 to 150,000 kg. of silver per annum, much of which comes from the silver tin mines. In depth, tin replaces silver. The greatest producers of tin are Unsia and Chocallo. Potosi is increasing its output of tin, while its silver output is falling rapidly. The veins seem to be changing into pyritic tin-bearing deposits.

The talk was illustrated by sketches and was most interesting and instructive.

The second speaker, Mr. Warner, described a recent trip which he made into the Mackenzie River country of the great Northwest. Starting from Edmonton, Alberta, the trip covered 1900 miles and occupied four months. Dog sleds afforded the principal means of transportation and accounted for 1420 miles of the journey. Athabaska Lake, which is open four months in the year, required six days to cross; the return over the lake was made just as the ice was breaking up.

The most important mineral deposits in the district visited by Mr. Warner appear to be those of nickeliferous pyrrhotite occurring in gabbro and norite intrusions in the Laurentian formations. Extensive areas of tar sands suggest the occurrence of oil-bearing deposits.

Mr. Warner's classification of the various Indian tribes of that country follows:

The Beavers.—The most advanced; copiers of the white man. Hunters.

The Credes.—Will not hunt, but trap and live on the meat thus obtained.

The Chippewa.—Industrious. Specialize in black fox and silver black fox trapping.

The Dog Ribs or Esquimo.—The lowest form of Indian.

In one respect they all resemble the Indians of this country, and that is in their love for whiskey. For one who is traveling light, as we must in that country, "pain killer," according to Mr. Warner, in quantities of one-quarter teaspoon to one cup of water, makes a satisfactory, acceptable, and possibly less injurious substitute, at least so far as the Indians are concerned.

Mr. Warner, in closing, answered numerous questions.

The 38th Meeting of the Boston Section was held at the Engineers' Club on Monday evening, Nov. 6, 1916.

Twelve members and guests were present for the dinner. Augustus Locke of Brookline, Mass., presented a paper entitled "Surface Induration in Outcrops of Copper Deposits."

Mr. Locke, in conjunction with Messrs. Perry and Bateman, and under the direction of Professor Graton of Harvard, has been engaged for several years in the study of the secondary enrichment of copper deposits. In connection with this work the croppings have naturally been

given close attention, and this paper embodies some of the results and conclusions of that study.

Mr. Locke first called attention to the fact that the words gossan, eisen hut, chapeau en fer, colorados, etc., indicate an early knowledge of outcrops. The discoveries at Morenci; Butte and Bingham showed that valuable deposits might underlie practically barren croppings, and suggested the advisability of studying these occurrences.

He next enumerated some of the questions for which they had tried to find satisfactory answers, as, for instance, the meaning of traces of elements in the croppings and the possibility of determining the texture of the primary ore from an examination of the croppings. Next were mentioned some of the results so far obtained.

The question of the "flinty croppings," so common in some outcrops and practically abundant at Ely, was taken up. The curious fact that the croppings are oftentimes much harder than the primary ore was explained as due to one or more of the following: Desiccation; limonite cementation; precipitation of silica in the form of chalcedony.

Induration due to increasing the limonite and silica in the outcrop is given the name "flinty cropping." This is seldom over 5 ft. thick and is always found confined to the surface. Explanation of this phenomenon was discussed and suggestions sought. Brecciation promotes the process, possibly by affording channels through which dissolved silica may readily pass. The author seemed inclined to give capillary attraction the credit for bringing dissolved silica to the region of the surface. An important and unfortunate result of the phenomenon is that it often renders uncertain or impossible the prediction of the nature of the primary rock below, *i.e.*, porphyry or limestone, for instance.

After an interesting discussion the meeting adjourned.

EDWARD E. BUGBEE, *Secretary*.

COLUMBIA SECTION

W. H. LINNEY, *Chairman*,

OSCAR LACHMUND, *Vice-Chairman*.

LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154,
Spokane, Wash.

STANLY A. EASTON,

S. SHEDD.

The sixth annual meeting of the Columbia Section was held in the Stone Room, Spokane Hotel, Spokane, Wash., on Nov. 25, 1916.

With Stanly A. Easton in the chair, about 25 members and guests sat down to a six-course dinner. After the dinner, Mr. Easton addressed the meeting as retiring chairman. He cited the activities of the Columbia Section during the year past, called the attention of the members to the present close relation of the Local Sections and the parent organization, gave some detailed information regarding the Arizona meeting of the Institute, dwelt at some length on the value of professional ethics to an engineer who wishes to attain high standing among his peers and with the general public, warned his hearers that the present period of inflated prices will prove the "acid test" in mining, stated that in his opinion the present European war will be determined largely by the availability of the metals, and finally referred to an article by Dr. Eliot, formerly presi-

dent of Harvard University, which appeared in a recent issue of the *Atlantic Monthly* and in which the author referred to "miners and other adventurers." This statement, Mr. Easton felt, was based upon ignorance of the men engaged in the industry and to be deplored because of the high standing of the author.

The secretary-treasurer followed with his report, which showed that the section has held two meetings during the past year and has participated in several conventions and excursions held under other auspices. It also showed that the membership has increased during the year from 129 to 158. The secretary read letters from several members who were not able to attend and also one from Mr. E. Jacobs, secretary Western Branch C.M.I., who, contrary to his usual custom, was absent.

Election of officers resulted as shown above.

The newly elected chairman, W. H. Linney, emphasized the good work which the Columbia Section has done, and said that he hoped that the section would keep its high standing and gain something in membership and value to the community during the coming year.

The chairman then introduced Prof. C. G. Warfel, Assistant Professor of Mining at the State College, Pullman, Wash., who read an interesting paper on "Electric Reduction of the Iron Ores of Eastern Washington." The material for the paper was gathered by request of the Spokane Chamber of Commerce and was preliminary throughout, the figures submitted being tentative, though thought to be not far from right. The paper covered iron analyses, construction costs, markets, depreciation and amortization, and other details. Dr. Warfel assumed the cost of materials delivered in Spokane to be about as follows: Iron ore, \$2.50 per ton; coke, \$5 to \$7 per ton; limestone, \$2 per ton; labor, \$2 per day. With these costs, he said, it would be necessary to procure electric current at a cost of about \$2 per horsepower year (\$16 per kilowatt year) to meet competition. Animated discussion followed the reading of the paper.

R. S. Ord, manager of the Corbin Coal & Coke Co., believed the use of the lower grades of coke would not be possible because manufacturers do not favor the low-grade or "black butts" coke on account of the narrow margin of profit and the better demand for high-grade coke at reasonable margins of profit. He also believed electric-current costs could be cut below present rated market prices.

W. Earl Greenough, former manager of the Marsh mines in the Cœur d'Alenes district, said that competitive rates indicate a reduction on rates quoted, i.e., \$25 to \$30 per horsepower year.

Prof. F. A. Thomson, of the Mining Department, Washington State College, Pullman, believed the question would be solved by adequacy of ore supply and power costs.

S. L. Boyer made reference to power costs at points of generation and at points more or less remote therefrom.

D. C. Livingston, Professor of Geology, University of Idaho, cited power costs in the irrigated areas of southern Idaho where power is sold during non-use periods at a very low price, approximating \$16 per horsepower year, or thereabouts.

The subject of flotation was then brought up for discussion. D. F. Haley, consulting engineer for the Interstate-Callahan Mining Co., gave some interesting figures on differences in grinding at certain milling plants. S. A. Easton cited the present Latouche operation in Alaska where all-sliming is securing a satisfactory result, and spoke of the char-

acter of the Alhambra ore in the Cœur d'Alenes, which is very finely disseminated throughout the gangue and of which the silver-lead contents will probably be recovered by an all-sliming process yet to be worked out, in which flotation will play an important part. Richard Marsh talked about selective and preferential terms, and Professor Thomson reported an interesting laboratory test on an ore carrying the copper, lead and zinc sulphides in which these minerals were successfully separated by the aid of flotation, and which he believes could be made of commercial value.

Coöperation between the State and National Surveys in geological survey work was brought up by some of the members and the question was left open for discussion at a later date.

More frequent meetings of local members are to be tried out.

The following named members were selected as a committee on papers: J. C. Haas, S. A. Easton, L. K. Armstrong.

The meeting then adjourned.

L. K. ARMSTRONG, *Secretary*.

MONTANA SECTION

J. L. BRUCE, *Chairman*,

W. C. SIDERFIN, *Vice-Chairman*,

WALTER E. GABY, *Secretary-Treasurer*, 319 N. Jackson, Butte,
Montana

N. B. BRALY,

W. T. BURNS.

The semi-annual meeting of the Montana Section was held at the Silver Bow Club, Butte, Montana, on Friday, Nov. 10, 1916, at 8.30 p.m. This was a postponement of the regular October meeting in order that the members might enjoy the presence of Dr. L. D. Ricketts, president of the Institute.

An informal dinner was served earlier in the evening to 90 members and guests. During the dinner a telegram was sent to E. P. Mathewson, expressing deep regret that he, the founder of the Montana Section, could not be present. On cards that were distributed at the dinner, was printed:

"We have with us to-night Dr. L. D. Ricketts, who, besides being President of the Institute, is the King of Arizona.

"And we haven't with us to-night Mr. E. P. Mathewson, the Instigator of the Montana Section, so let's remember him."

Vice-Chairman W. C. Siderfin introduced Dr. Ricketts as the principal speaker of the evening, and the latter addressed the members at some length on the subject of committee work as applied to technical research. He explained his subject by pointing out historical developments in mining and metallurgy, showing how the real results had been accomplished mainly by coöperative work in the study of intricate problems. As related to Butte, he mentioned in particular the development of our cheaper methods of mining on a large scale, citing the Ohio system of slicing and relating how this and other methods had been first suggested by the crude practices of earlier days, only later to be perfected and applied on a large scale by corps of able engineers. He also gave an interesting account of some features in the development of flotation practice, and cited some instances of the modern treatment of mixed ores by ingenious combina-

tions of methods, pointing out the great possibilities for research in this special field. He finished his remarks by emphasizing the value of committee work on such problems as now confront us in all branches of mining, and suggested that the local sections of the Institute select men to study them, believing that coöperative work among such men is most likely to yield important results.

B. B. Thayer, past-president of the Institute, then told of his trip to South America with Dr. Ricketts, Reno Sales, and others, giving a very interesting account of conditions in that part of the world. He spoke of the tremendous future before that country, and urged that the Spanish language be thoroughly taught in all our mining and other technical schools. He also reviewed the growth of the Andes Exploration Co., and spoke of Potrerillos and other properties in South America.

Dr. F. G. Cottrell was introduced and talked on the scope of work in the Bureau of Mines. In connection with metallurgy, he prophesied that future advances in that science would be largely concerned with gases, stating it to be the fact that at Anaconda the weight of gaseous products going to waste through the stack was 20 times that of the slag.

H. S. Ware, assistant superintendent of the Anaconda Smelter, followed with an interesting account of some of the points in connection with the Institute meeting in Arizona. A map of Arizona and New Mexico had been prepared, and a collection of the descriptive booklets of the different plants visited afforded the members an opportunity to examine details of these works. He wished to record the appreciation of the Montana members who went on the trip for the cordial reception given them by the Toltec Club of El Paso, with the Mexican dinner that was prepared for them, and for the automobile trips to points of interest about the city. Compliments were also extended to the mining companies and their staffs at Santa Rita and Hurley, New Mexico, and at Douglas, Bisbee, Warren, Globe and Miami, Arizona, for the excellent entertainment provided by them, and to the Arizona members in general for the delightful time enjoyed during the entire trip.

Discussion by Dr. Ricketts brought out some of the novel features of the plant at Inspiration, particularly the automatic hoist. Referring to smelter practice in the Southwest, he mentioned the superiority of the Peirce-Smith converters in point of dust losses, and spoke of the manufacture of sulphuric acid from converter gas in regions of sulphide ores, for use in leaching operations in oxide regions where no sulphur is at hand.

C. W. Goodale was called on to talk, and mentioned in particular the pleasures of the Roosevelt Dam trip and the sights along the Apache Trail. Mr. Goodale visited Jerome and Ajo, not covered by the itinerary of the Arizona trip, and told something of these places.

C. R. Kuzell, of Anaconda, spoke on smelter methods as observed by him in Arizona, and referred to the stay at Grand Canyon.

Frederick Laist, general manager at Anaconda, discussed features of dust losses as handled in his plant. Dumping from the cars is done in tunnels connected with the stack so that the dust is kept going in one direction to be collected. Using this and some points in flotation practice for illustration, he called attention to the exceedingly small quantities sometimes dealt with by the modern metallurgist, factors that would have been disregarded not so long ago, and to the very minute quantities of some elements that had to be excluded in certain processes.

F. E. Marcy spoke of the importance to the small operator of separating classes of tailings, considering it an act of courtesy to the fellow who will have to come along later and treat them.

C. L. Berrien, assistant general superintendent of the Anaconda mines, was asked to describe mining methods of the Southwest as applied to the Butte district, and made clear that the only method feasible here is the cut-and-fill, or rill system, because the surface over most of the camp must be held up. He said cost systems had been introduced for the original square-set method for comparison with the new methods as applied at Butte, and that 15 to 58 c. per ton were being saved by the new methods. He said that 15 per cent. of the square-set stopes had been shifted to the cut-and-fill system in two months, and that the real saving had not yet been reached.

R. H. Sales was requested to speak on South America and gave an interesting picture of the country in general and of the character of the people. In some places, he said, 95 per cent. of the population is composed of natives.

At 11.15 p.m. the meeting adjourned somewhat precipitously because 30 or more of the members and guests from Anaconda had to make a break for their train.

WALTER E. GABY, *Secretary*.

NEW YORK SECTION

DAVID H. BROWNE, *Chairman*,

PERCY E. BARBOUR, *Vice-Chairman*,

A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.,

C. A. BOHN, *Treasurer*,

JOHN V. N. DORR,

LEWIS W. FRANCIS.

The first meeting of the New York Section for the year 1916-17 was held at the Machinery Club on Monday, Nov. 27, 1916. The meeting was preceded by a dinner. The subject for the evening was the work of the American Engineers in Belgian Relief. As the topic was one in which women as well as men are interested, the wives of members were invited to attend. This is the first time that ladies have been present at any of our section meetings and the innovation proved successful, about 35 ladies being on hand. 75 members of the section were present.

The chief speaker of the evening was Edgar Rickard, Assistant Director in America of the Commission for Relief in Belgium, who told us of the organization and scope of the work.

Dr. Arrowsmith, one of the delegates of the Commission, showed lantern slides illustrating the method of getting the food into Belgium and distributing it throughout that country and northern France.

We were also favored by the presence of the Director of the Commission in America, Mr. Honnold, who made a few remarks.

Dr. Ricketts, the President of the Institute, attended the meeting. He spoke briefly in appreciation of the speeches made by our guests and of the work they are doing, and moved a vote of thanks by rising.

A. D. BEERS, *Secretary*.

AFFILIATED STUDENT SOCIETY NOTES

The Student Branch at the University of Illinois met for the first time on October 10, at the home of Professor H. H. Stoeck, Head of the Department of Mining Engineering. The following officers were elected:

President, C. W. Campbell,
1st Vice-President, L. H. Whitney,
2nd Vice-President, L. E. Yindrock,
Secretary-Treasurer, N. M. Morris.

On the evening of October 27, the Society gave a smoker, at which Dr. F. M. Van Tuyl, of the Department of Geology, told of his experiences while on the University expedition, led by Professor Savage and himself, to the Hudson Bay region, where they made a study of the Ordovician, Silurian and Devonian formations.

The Society is arranging to have a series of talks on mining and metallurgical subjects during the year.

N. M. MORRIS, *Secretary-Treasurer*.

The Mining and Metallurgy Society of the Carnegie Institute of Technology recently elected the following officers:

President, Kenneth M. Smullen,
Vice-President, F. W. Smith,
Treasurer, R. O. Williams,
Secretary, E. Lundeen.

The Society has one meeting each week, for which one of the members prepares a paper. The program through November was as follows:

Oct. 27.—R. O. Williams, "Experiences in Coal Mining."

Nov. 3.—C. O. Kingsbury, J. C. Reisinger, "Experiences at Bisbee."

The next three weeks were devoted to a study of copper, as shown by the three typical districts: Lake Superior, Butte, Mont. and Bingham Canyon, Utah. The processes in these districts are different. This subject was taken up in the following order:

Nov. 10.—F. D. Anderson, "The History of Copper as Shown in the Three Typical Districts."

Nov. 17.—F. W. Smith, "The Methods of Mining Copper in the Three Typical Districts."

Nov. 24.—H. E. Hornickel, "The Metallurgy of Copper in the Three Typical Districts."

We hope to have monthly meetings also, to which all graduates of the department who are in the vicinity of the Institute may be invited. We are planning to ask non-members of the society to speak at these meetings. Professor McIntosh has already favored us with a paper on "The Recovery of Ferro-Alloys from Slags."

E. LUNDEEN, *Secretary*.

The Associated Miners of the University of Idaho have elected the following officers for the current year:

President, C. Y. Garber,
Vice-President, W. K. Horning,
Secretary, A. J. Lyon,
Treasurer, C. J. McCormick.

The third meeting of the year was held on Monday evening, November 6. Mr. Jerome J. Day, President of the Tamarack and Custer Mining Co., and President of the Northport Smelter, addressed the students. His subject was the training of a mining engineer. He told the students that they should work 20 hr. out of the 24, if necessary, as the requirements of the mining world are exacting; and that, while they must leave the University well trained, they must recognize that on graduating they are really just commencing life. He encouraged them by naming a long list of Idaho graduates who rank high in the profession, and said that the outlook never was brighter than now for the mining engineer. Mr. Day said that he had yet to learn of the failure of an Idaho graduate.

A. J. LYON, *Secretary.*

The Student Branch at the Ohio State University has elected officers for the current year:

President, George Putnam,
Vice-President, R. C. Lawrence,
Treasurer, R. K. Bowden,
Secretary, A. W. Seabright,
Director, C. E. Moser.

On November 1, the Student Branch gave a Six O'clock Dinner, at which the following addresses were made:

The Society in the Past—Assistant Professor Smith.

Some of the Old Fellows—Professor Sommermeier.

Bromine Recovery at Pomeroy—Professor Bownocker.

Flotation Concentration—Professor McCaughey.

The Human Engineering Congress—Professor Demorest.

On November 16, the Society was addressed by Dr. A. M. Bleile of the Department of Physiology, on the subject of Camp Sanitation.

A. W. SEABRIGHT, *Secretary.*

Officers of the University of Washington Mining Society for the school year 1916-17 are:

President, Jess C. Johnson,
Vice-President, E. Lee Tucker,
Secretary, Richard Luther,
Treasurer, Walter F. Brown,
Cor. Secretary, Henry G. Boulton.

Early in the year we amended the constitution, making all members of the Mines College eligible to become members of the Society, rather than only Juniors and Seniors as heretofore. We hope in this way to increase the value of the Society.

One very satisfactory smoker has been given this year, and various prominent mining men have addressed the Society. Among the speakers have been Mr. Bart Thane, Superintendent of the Alaska Gastineau Mines, Thane, Alaska, and Mr. Seagrave, Superintendent of the Kennecott Mines at Kennecott, Alaska. Mr. H. H. Smith of Tacoma is expected to speak in the near future.

The Mining Society, in conjunction with the Societies of the Civil, Electrical and Chemical Engineers, has planned to give an all-engineers' dance.

HENRY G. BOULTON, *Cor. Secretary.*

The **Mining Association of the University of California** has elected the following officers:

President, M. H. Knowles,
Vice-President, C. R. Knox,
Secretary, H. T. Helgesson,
Treasurer, A. C. Kroeger.

The Association has at the present time a membership of 86, and is in a flourishing condition.

During this term the Association has been favored with addresses, as follows:

Opportunities for the Young Engineer—Mark L. Requa.

Mining Resources of the West—F. McN. Hamilton.

Qualities of the Successful Engineer—T. A. Rickard.

Business in Mining—C. T. Hutchinson.

Mining at Chuquicamata—T. A. Barrington.

Mining at Ely, Nevada—J. Bouse.

The curriculum of the College of Mining has been enlarged by the addition of a course in Petroleum Engineering. This is a separate and distinct course and leads to the degree of B. S. in the College of Mining. A Petroleum Club has been formed, and has a membership of 75. This club holds bi-weekly meetings and it is planned to have suitable people lecture on subjects relating to all phases of the Petroleum Industry. The first lecture was given by Dr. J. C. Merriman on "The Origin of Oil in California."

The **Mining Society of the University of Kentucky** is having a successful half year under the following officers:

President, W. C. Eyl,
Vice-President, H. Kempton,
Secretary-Treasurer, O. G. Schwant.

The Society holds an election at the beginning of each semester.

A Society pin was chosen last Spring and is now being worn by the members.

The Society is planning to publish a paper, but has not yet decided upon its form.

The manner of initiating Freshmen has been chosen. Each new member is required to wear a standard miner's cap about the campus for two weeks. This also helps to advertise the mining department.

Including the professors of the department, the society has 27 members.

OTTO SCHWANT, *Secretary-Treasurer*.

The **School of Mines Society, University of Minnesota**, held a special meeting on November 2, at which Dr. W. H. Emmons, head of the department of Geology, spoke on the "Possibilities of the Mining Profession."

On November 21, Henry M. Payne, C. E., consulting engineer for the Canadian Klondike Mining Co., gave a talk on "Placer Mining of the Frozen Gravels of the Arctic," illustrated with lantern slides.

At a meeting on November 28, Mr. Hugh Roberts, consulting geologist for the Longyear Exploration Co., spoke on "Surface Prospecting." He laid special stress on the economic losses resulting from the present rather shiftless methods.

Mr. E. M. Lambert, professor of mathematics, and member of the School of Mines faculty, will speak to the Society on the subject of the "Development of the Modern Steam Engines," on December 21.

H. K. ARMSTRONG, *Secretary*.

The Students' Mining Society, Massachusetts Institute of Technology, has elected the following officers for the present year:

President, George Roper, Jr.,
Vice-President and Treasurer, Philip N. Rowe,
Secretary, Robert W. Rogers.

The Scientific Society of the Colorado School of Mines has elected the following officers:

President, C. M. Knepper,
Vice-President, H. E. Munn,
Secretary, L. Ehnborn.

The first meeting of the school year was held on December 8. The meeting was called to order by President Knepper at 7.30. Attention was called to the importance of the cement industry to the mining engineer from the standpoint of mining the raw materials, and the use of the finished material in construction of buildings and underground hoist and pump rooms and the lining of shafts and tunnels. Professor R. S. Hawley gave an interesting lecture, illustrated by colored slides, on "Use of Portland Cement in Construction and Mining."

The meeting adjourned at 8.30. Sixty members were present, and all felt that a profitable and interesting evening had been spent on the discussion of concrete from the mining engineers' standpoint.

L. EHNBÖM, *Secretary*.

The Mining Engineering and Geological Society of the State College of Washington wishes to announce that Dr. Henry Mace Payne, late of Russia, was an honored guest at a luncheon given by the Society, which was prepared by the Home Economics Department. Dr. Payne related many of his experiences of his sojourn in Russia and talked on "The Wide World as a Field for the American Engineer."

The Society held a joint meeting with the Twentieth Century Club, at which Dr. Payne gave an illustrated lecture on the thawing of frozen gravels.

The annual feed of the Society will be held Dec. 8, when we expect to initiate ten freshmen into the secrets of the organization.

Every two weeks the society meets in room 107, College Hall. A short business meeting is held and two papers are read. These papers are prepared by the students and faculty. After the papers, a discussion is held and then the papers are preserved by the secretary for future reference. The meetings are alive and always well attended.

AUBREY C. MILLER, *Secretary*.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

From the Library of John B. Pearse

The library of John B. Pearse, Vice-President of the Institute in 1877-78, and one of the first manufacturers of steel for rails in this country, is being sold at auction. The scientific and technical books were sold on November 21-23, and the Librarian purchased a large number of volumes for the United Engineering Society library.

Among the notable acquisitions is an edition of Pliny, of Venice, 1489; one of 1618 in German with extensive woodcut illustrations; a manuscript copy of "The Case of Peter Hasenclever," an iron manufacturer in 1764-1769 in New York State; books by Kerl, Wedding, Fairbairn, Cotta, Holley, Maurer, Plattner, and other well-known early metallurgists; a very large collection of books and pamphlets on amber; a history of metals issued in 1671, by John Webster; a partial set of the *Oesterreichische Zeitschrift für Berg und Hüttenwesen*; a very early edition of Paracelsus; and many early books on machinery. In all, over 2000 books and pamphlets were purchased.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy and Chemistry

- ASSAY OF TIN AND ANTIMONY. Ed. 2. By L. Parry. London, n. d.
 CHEMICAL STUDY OF ILLINOIS COALS. Bull. No. 3, Coal Mining Investigations. Urbana, 1916.
 MELTING ALUMINUM CHIPS. Bull. No. 106, U. S. Bureau of Mines. Washington, 1916.
 THE NITRATION OF TOLUENE. Tech. Paper No. 146, U. S. Bureau of Mines. Washington, 1916.
 USE OF MUD-LADEN FLUID IN OIL AND GAS WELLS. Bull. No. 134, U. S. Bureau of Mines. Washington, 1916.

Geology and Mineral Resources

- COAL RESOURCES OF DISTRICT VI-ILLINOIS. Bull. No. 15, Coal Mining Investigations. Urbana, 1916.
 FLORIDA GEOLOGICAL SURVEY. 8th Annual Report. Tallahassee, 1916.
 HANDBUCH DER REGIONALEN GEOLOGIE. Edited by G. Steinmann and O. Wilckens. Pts. 1-18. Heidelberg, 1910-15.
 INVESTIGATION OF THE COALS OF CANADA. (Extra Volume.) Ottawa, 1915.
 IOWA GEOLOGICAL SURVEY. Annual Report. Vol. XXV, 1914. Des Moines, 1916.
 MINES HANDBOOK AND ENLARGEMENT OF THE COPPER HANDBOOK. Vol. XII. By W. H. Weed. New York, 1916.
 NEW ZINC MINING DISTRICT NEAR EDWARDS, N. Y. By David H. Newland. (Reprinted from Economic Geology, Oct.-Nov., 1916.) (Gift of author.)
 PRELIMINARY OIL REPORT ON SOUTHERN ILLINOIS. (Extract from Bull. No. 35, Ill. State Geological Survey.) Urbana, 1916.
 TECHNICAL REPORT UPON A MINING SURVEY MADE ON LANDS OF THE MOTEMBO PLANTATION, CORRALILLO, PROVINCE OF SANTA CLARA, IN CONNECTION WITH REPORTS RECEIVED BY THE GOVERNMENT OF THE REPUBLIC CONCERNING THE FINDING OF EXTENSIVE DEPOSITS OF POTASH. (Gift of Bradley Stoughton.)
 UNION OF SOUTH AFRICA. Mines Department. Annual Report, 1910-1912, pts. I-II; 1913, pt. III-IV.
 ———. Annual Report of the Government Mining Engineer. 1914. Pretoria, 1915.
 ———. Department of Mines and Industries. Annual Report of the Government Mining Engineer. 1915. Pretoria, 1916.

General

- BIBLIOGRAPHY—SPONTANEOUS COMBUSTION AND WEATHERING OF COAL. (Gift of Edwin B. Ricketts.)
 NEW YORK TIMES INDEX. Vol. IV, July-September, 1916. New York, 1916.
 U. S. PATENT OFFICE. Manual of Classification of subjects of invention. Revised to January 1, 1916. Washington, 1916.
 CONNELLSVILLE COAL TARIFF ASSOCIATION. Excerpts from testimony of J. P. Muller, before Interstate Commerce Commission, July 12-13, 1916.
 ———. In re bituminous coal rates to central freight association territory. Series X, XI. Before the Interstate Commerce Commission, July 11, 1916. (Gift of Richard Peters, Jr.)
 COLLECTION OF REPORTS OF PRESIDENTS' CONFERENCE COMMITTEE ON FEDERAL VALUATION OF THE RAILROADS IN THE UNITED STATES. (Gift of Clemens Herschel.)
 CATALOG STUDIES. Vols. 1, 4, 6, 7, 10, 12, 15, 23, 24, 27, 28, 34, 37-40, 42, 43, 48 & Index. (Gift of Catalogue Equipment & Supply Co.)
 BLUE PRINT BOOK. 56 prints. (Gift of Catalogue Equipment & Supply Co.)

Gift of Engineering and Mining Journal

- ANACONDA COPPER MINING COMPANY. Safety First. Suggestions to employees. Butte, 1914.
- CLEVELAND CLIFFS IRON COMPANY. Rules and Regulations for Superintendents and Foremen, for the protection of life and property. 1912.
- SALT IN CHESHIRE. By A. F. Calvert. London-New York, 1915.
- COPPER QUEEN CONSOLIDATED MINING COMPANY. General Rules for the prevention of accidents.
- ENGINEER. Yearbook of the students, faculty and alumni of the Michigan College of Mines. Vol. 1, 1915-16.
- NEVADA CONSOLIDATED COPPER Co. Safety First. October, 1914.
- NOUMÉA. Chambre de Commerce. Compte rendu. Année, 1913. 1914.
- Panama-Pacific International Exposition Company. Legacy of the Exposition. San Francisco, 1916.
- NEW ZEALAND. Government Statistician. Statistics of the Dominion of New Zealand. 1913. Vol. III—Production, Finance Accumulation, postal and telegraph. Wellington, 1914.
- Sveriges Officiella Statistik. Bergshandtering för år 1914. Stockholm, 1915.
- . Handel berättelse för år 1914. Stockholm, 1916.
- . Industri berättelse för år, 1914. Stockholm, 1916.

Trade Catalogs

- AMERICAN GRONDAL Co. New York, N. Y. Grondal Briquetting. Concentrates made by Grondal separators. Magnetic Separators.
- BACHARACH INDUSTRIAL INSTRUMENT Co. Pittsburgh, Pa. Catalog B. Hydro volume and pressure recorders.
- BERGER MFG. Co. Canton, Ohio. Bull. No. 12, Metal Lumber.
- BUILDERS IRON FOUNDRY. Providence, R. I. Bull. No 86. Venturi Gas and Air Meter for Station and Industrial Service.
- CAMERON CENTRIFUGAL PUMP WORKS. New York, N. Y. Bull. No. 7150. Centrifugal Pumps. Sept., 1916.
- COLORADO IRON WORKS COMPANY. Denver, Colo. Pamphlet No. 31. Grinding machinery.
- DENVER FIRE CLAY Co. Denver, Colo. Catalog D. Case Metallurgical Furnaces. 1916.
- EMPIRE CONCENTRATOR COMPANY. Denver, Colo. Empire Concentrating Table.
- INGERSOLL-RAND COMPANY. New York, N. Y. Form No. 92075. Fabricas y Talleres. Sept., 1915.
- JEFFREY MANUFACTURING COMPANY. Columbus, Ohio. Bull. No. 191. Jeffrey "Arcwall" coal cutters.
- BULL. No. 192. Care and operation of Jeffrey 35B-Shortwall mining machine.
- SULLIVAN MACHINERY COMPANY, Chicago, Ill.
- BULL. No. 58D (Second ed.) Sullivan Tandem Corliss Air Compressor Class "WC," Sept., 1910.
- BULL. No. 58H (Third ed.) Sullivan Small Air Compressors. Oct., 1914.
- BULL. No. 58L. Air compressor accessories. Oct., 1913.
- BULL. No. 58M. Cross Compound Power-driven Air Compressors "WJ-2," "WN-2," "WS-2." May, 1913.
- BULL. No. 58-S. Angle Compound Power-driven Air Compressors Belt-driven class "WJ-3," Direct Connected, Class "Wn-3." Dec., 1913.
- BULL. No. 75-A. Single Stage. Power-driven Air Compressors Classes "WG-3," "WK," "WK-2." Sept., 1916. New York, N. Y.
- WALLACE SUPPLY Co. Hand Power Bending Tools.
- WESTINGHOUSE ELECTRIC & MANUFACTURING COMPANY, Pittsburgh, Pa. Catalog 3002-A. Sections 7, 9, 3073, 3097, 3103, 3163, 3185, 3226, on Motor-driven Machinery, and Section 3042—Portable Electric Tools.
- Circular 1502. Westinghouse Distributing Transformers. April, 1909.
1535. Railroad Electrification. January, 1914.
1516. Baldwin Westinghouse Electric Locomotives. May, 1912.
1546. Train Operation. October, 1914.
1550. Westinghouse 1500-volt Direct-current Sub-station Equipment. Oct., 1914.
- SMALL MOTORS. Nos. 21, 22, 23.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Nov. 10, 1916 to Dec. 10, 1916.

- BARNSDALL, THEODORE N., Treas. & Mgr., Highland Gas Co.,
24 West Corydon St., Bradford, Pa.
- BEASLEY, FRED E., Asst. Supt., Copper Refinery, Consolidated Mining &
Smelting Co., Trail, B. C., Canada.
- BREWSTER, THOMAS T., Vice Pres., Gen. Mgr., Mt. Olive & Staunton Coal Co.,
1012 Federal Reserve Bldg., St. Louis, Mo.
- BRINDLE, ARTHUR ST. C., Mining Recorder's Office, Trout Lake, B. C., Canada.
- BULLOCK, HARRY E., Pres., Kentucky-Jewel Coal Co., Lothair, Ky.
- BUTLER, PERCY BENNETT, Mine Operator, P. O. Box 335, Joplin, Mo.
- BUTLER, ROBERT S., Supt., Three Johns Mining Co., Joplin, Mo.
- CABOT, GODFREY L., Carbon Black Maker, Gas Engr.,
940 Old South Building, Boston, Mass.
- CARPENTER, EVERETT, Chief Geol., Empire Gas & Fuel Co., Bartlesville, Okla.
- CLARK, CHARLES W., Genl. Mgr., United Verde Copper Co., Jerome, Ariz.
- CONOVER, CAIRY C., Asst. Supt., National Zinc Co., Springfield, Ill.
- DALY, MARCEL, 1605 21st Ave., Seattle, Wash.
- DAVIES, JOHN V., Cons. Engr., Vice Pres., Jacobs & Davies Inc.,
30 Church St., New York, N. Y.
- DUNN, DAVID L., Min. Engr., Mgr., Spurgeon-Tiger Mining Co., P. O. Box 391,
Joplin, Mo.
- EMRICH, JAY L., Chief Chem., Durango Plant, American Smelting & Refining Co.,
800 Seventh Ave., Durango, Colo.
- FITCH, HOWARD A., Pres., Kansas City Structural Steel Co.,
1012 Baltimore Ave., Kansas City, Mo.
- FOSTER, ROLLINS S., Safety Engr., Bonne Terre, Mo.
- FOUTZ, G. V., Chem., Aguilera & Co., Minas de Ponupo, La Maya, Oriente, Cuba.
- FRITCH, GILES MARTIN, Met., American Brass Co., Kenosha, Wis.
- GARMS, WALTER IRVING, Min. Engr., Genl. Mill Foreman, Ray Consolidated
Copper Co., Hayden, Ariz.
- GAYLORD, EARL G., Geol., Kern Trading & Oil Co., Flood Bldg., San Francisco, Cal.
- HULL, DANIEL R., Met. Chem., American Brass Co., Kenosha, Wis.
- JOBES, CHARLES T., Cons. Min. Engr., 3236 The Paseo, Kansas City, Mo.
- KITHIL, KARL L., Vice Pres. & Genl. Mgr., Schlesinger Radium Co.,
P. O. Box 1316, Denver, Colo.
- KOLENSKY, ALEXANDER V., Min. Engr., Nevsky 1, Petrograd, Russia.
- LONGYEAR, ROBERT D., Geol., E. J. Longyear Co., 710 Security Bldg.,
Minneapolis, Minn.
- MCDERMOTT, EDWARD D., Min. Engr., Russian Mining Corp., Ltd.,
Zyrianofsk, Roudnik, Tomsk Government, W. Siberia.
- McKEE, WILL E., Supt., Machy., Calumet & Arizona Mining Co., Warren, Ariz.
- NOWLIN, ROBERT A., Min. Engr., Crozer Land Asso., Elkhorn, West Va.
- PATTERSON, GEORGE S., Min. Engr., Northern Ore Co., Edwards, N. Y.
- PITT, DALE L., Min. Engr., Tacoma Smelting Co., Tacoma, Wash.
- RINEHART, CHARLES R., New York Representative and Engr., Lehigh Car, Wheel,
& Axel Wks., and Fuller Engrg. Co., 50 Church St., New York, N. Y.
- RUSS, LEON F., Geol., Cia. Mexicana de Petroleo "El Aguila," S. A.,
Apartado 150, Tampico, Tamp., Mexico.
- SCHNEIDER, ANTON, Supt., Amalgamated Phosphate Co., Brewster, Polk Co., Fla.
- SHERIDAN, GUY E., Chief Chem., Timber Butte Milling Co., 617 Diamond St.,
Butte, Mont.
- SHOFFSTALL, ARTHUR S., Research Engr., The International Nickel Co.,
Bayonne, N. J.
- SNYDER, T. E., Genl. Mgr., J. S. Wentz Co., 906 Markle Bank Bldg., Hazelton, Pa.
- STONE, LAURENCE H., Supt., Richard Mine, Thomas Iron Co., Wharton, N. J.
- STURGES, THOMAS B., Vice Pres. & Genl. Mgr., Pennsylvania Drilling Co.,
30 Carson St., Pittsburgh, Pa.
- SULLIVAN, CLARKE, Mill Supt., Eden Mining Co., Bluefields, Nicaragua,
Central America.

VAHRENKAMP, FREDERICK H., Min. Engr., Pres., Genl. Mgr., Boston Development Co., 12 Boston Bldg., Salt Lake City, Utah.
 WATERHOUSE, LESLIE V., Mill Supt., Mt. Lyell Mining & Railway Co., Queenstown, Tasmania.
 WILKE, WILLIAM, Cons. Engr. 86 Norwood Ave., Buffalo, N. Y.
 WOODWARD, HERBERT WATSON, Engr., Iron Cap Copper Co., Copper Hill, Ariz.

Associate Members

GUSTIN, FRANK J., Attorney-at-Law, Secy., Fissures Exploration Co., 1301 Walker Bank Bldg., Salt Lake City, Utah.
 RILEY, WALTER J. Pres., O. F. Jordan Co., East Chicago, Ind.
 SCAMMELL, MATTHEW J., Genl. Supt., Maryland Plant, Bethlehem Steel Co., Sparrows Point, Md.
 STRAUS, ROGER W., Asst. to Chairman, Board of Directors, American Smelting & Refining Co., 120 Broadway, New York, N.Y.

Junior Member

KNIGHT, OSCAR A., Asst. in Metallurgy & Metallography, Watertown Arsenal, Watertown, Mass.

Total Membership, Dec. 10, 1916. 5,894

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

Application Lacking Endorsement

Application for membership has been received from Mr. Drewitt, whose record is given below. This application lacks the necessary number of endorsers, but since this candidate lives at some distance from the headquarters of the Institute, the record is published here in order that any members who are acquainted with him may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing the candidate.

George Edward Drewitt, Mt. Morgan, Queensland, Aust.

Proposed by

Born . . . 1895-1900, Higher Grade School, Gillingham, England.
 1903-09, Electrical and Mechanical Engrg., Gillingham Tech. College (evening).
 1909-10, Min. Engrg. course, South Kensington, London, England. 1900-09, Engr. and draftsman, Medway Iron Works, Rochester, England. 1910, Engr., Maschinen-patrick, Lubeck, Germany. 1911-13, Asst., West African Mines. 1913, Reorganizing cement works and coal mines and reporting on properties, The Victorian Portland Cement Co., Richmond, Vict., Aust. 1912-13, Reporting, Pacific Phosphate Co., Ocean Island, Gilbert Group, Central Pacific. 1913-14, Reporting on mines and mills and Supt. erection of mills.

Present position: Engr. Dept., Mt. Morgan Gold Mines Co., Ltd.

MEMBERS

The following persons have been proposed during the period Nov. 10, 1916, to Dec. 10, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A

sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Edward Crawford Arnold, Flat River, Mo.

Proposed by H. A. Guess, A. E. Ring, H. G. Washburn.

Born 1885, Graham, Texas. 1902, High School, Graham, Texas. 1902-06, A. & M. College, College Station, Texas; degree, M. E. 1909-10, University of Texas, Austin, Texas. 1906-07, Instructor, Manual Training, High School of Marlin, Texas. 1907-8, Levelman, Transitman and Resident Engineer, Wichita Falls & Southern Ry., Wichita Falls, Tex. 1908-09, Shift Boss, Cyanide Plant, Inde Gold Mining Co., Inde, Durango, Mexico. 1911-13, Shift Boss of Mill, Filter and Cyanide Plant, Assayer, Inde Mining Co., Inde, Durango, Mexico. 1913-15, Asst. Engineer, Federal Lead Co.; Asst. in Experimental Plant, Federal Lead Co. 1915-16, Chief Engineer, Federal Lead Co., Flat River, Mo.

Present Position: Chief Engineer, Federal Lead Co., Flat River, Mo.

John Williams Austin, Superior, Ariz.

Proposed by Walter H. Aldridge, I. A. Ettlinger, W. A. Browning.

Born, 1880, Saginaw, Mich. 1898, Graduated from High School, Saginaw, Mich. 1907-10, Attended Michigan College of Mines at Houghton, Mich. 1910, B. S. and E. M. Degrees. 1907-1910, Miner and Millman, Nevada-Utah Mining Co., Bingham, Utah. Miner, Assayer for Utah Consolidated, Bingham, Utah. 1911, Mill Shift Boss, Bingham New Haven Co., Bingham, Utah. 1912, Assayer, Old Dominion, Globe. 1913, Chief Engineer, Superior & Boston Copper Co., Globe.

Present position: Chief Chemist, Magma Copper Co.

Herman Edward Beyer, New York, N. Y.

Proposed by W. L. Saunders, L. D. Albin, Geo. A. Howells.

Born 1879, Long Island City. 1902, Mechanical Eng., Cornell Univ. 1902-06, In shops Southwark Foundry & Mach. Co. 1906-10, Mgr. Pittsburgh office, Southwark Foundry & Mach. Co. 1910-13, Hardie Tyler Co., Birmingham, Ala. Unetti Howel Marble Co., Birmingham. 1913-16, Condenser Engineer, Cressoa Norris Co., Phila., Pa. 1916, Condenser Engineer, Ingersoll Rand Co., New York, N. Y. C.

Present position: Condenser Engineer, Ingersoll Rand Co.

George Henry Booth, Miami, Ariz.

Proposed by George R. Lehman, C. E. Arnold, L. D. Ricketts.

Born 1875, Poughkeepsie, N. Y. 1898, Mass. Institute of Technology, B. S. 1916, Mechanical Engineer, Inspiration Consolidated Copper Co. 1912-15, Chief Draftsman of Inspiration Consolidated Copper Co. 1912, Standard Portland Cement Co. 1908-12, Miami Copper Co. 1906-08, Moctezuma Copper Co. 1903-06, Detroit Copper Mining Co. 1900-03, Newport News Ship Building & Dry Dock Co. 1898-1900, New York & New Jersey Telephone Co.

Present position: Mechanical Engineer, Ins. Cons. C. Co.

Charles Alvin Brooks, Lead, So. Dak.

Proposed by B. C. Yates, A. J. M. Ross, W. J. Sharwood.

Born 1888, Carbonate, S. D. 1908, Lead High School, Lead, S. D. 1908-11, Helper, Mine Survey, Homestake Mining Co. 1911 to date, Mine Surveyor, Homestake Mining Co.

Emil John Bruderlin, Guayacan, Chile, South America.

Proposed by E. H. Laws, H. J. Stehli, S. J. Gormly.

Born 1887, Denver, Colo. 1906-10, Colo. School of Mines, M. F. 1902-06, Manual Training High School, Denver, Colo. 1910-16, Ohio & Colo. Smelting & Refining Co., Salida, Colo. Ore Trading Co., Guayacan, Chile.

Present position: Plant Engineer.

Evans W. Buskett, Joplin, Mo.

Proposed by E. F. L'Engle, E. T. Lednum, G. B. Corless.

Born 1874, St. Louis, Mo. 1895, B. S. in Chem. and Met., Missouri School of Mines. 1907, Met. Engr., Missouri School of Mines. 1908, St. Louis S. & R. Co., as Met. Engr. April, 1910, Mining, Joplin, Mo. June, 1911, Assayer, Bi-Metallic Mine, Philipsburg, Mont. 1912, Ruhl & Shanklin, Joplin, Mo., Surveying. 1913-14, Assayer, American Zinc, Lead & Smelting Co., Cartersville, Mo.

Present position—1914 to date: Customs Assay Office, Joplin, Mo.

Alexander W. Carroll, Perth Amboy, N. J.

Proposed by S. Showronski, A. Clayton Clark, Edward Keller.

Born 1867, Baltimore, Md. Grad. of Baltimore Technical School. 1888-92, Pennsylvania Steel Co., Asst. to Master Mechanic. 1892-97, Asst. Master Mechanic, Homestead Steel Works, Carnegie Steel Co. 1897-1900, Master Mechanic of American Steel & Wire, Beaver Falls, Pa., 1900-03, Chief Eng., Delamar-Copper Ref. Co., Chrome, N. J. 1903-10, Gen. Supt. Chrome Steel Works, Chrome, N. J. 1910-15, Mechanical and Electrical Engineer, Barclay Parsons & Klapp, N. Y. C.

Present position: Gen. foreman, Raritan Copper Works.

Samuel Northrup Castle, Davenport Neck, New Rochelle, N. Y.

Proposed by H. L. Smyth, Charles H. White, George S. Raymer.

Born 1880, Honolulu. 1890-96, Halle a/d Salle, Germany. 1897-98, Hamilton College. 1898-1901, Harvard College, A. B. 1901-02, Harvard University Graduate. 1902-04, Cornell Univ. Graduate School. Work at Harvard specially history and philosophy, general science and economics, the latter with particular reference to engineering phases and the extra year in special work. Cornell, 2½ years, covering Mechanical, Electrical and Hydraulic work. Special attention to fluids and gases. 1904-06, Allis-Chalmers, Milwaukee Shops, Manufacturing. 1906-08, Allis-Chalmers Construction Engineer, N. Y. District. 1908-09, Europe, Investigation Tour-Shop practice, hydro-electric, electro-metallurgical developments. 1909, General Electric, Commercial Engineer.

Present position: Commercial Engineer, General Electric Co.

James Chapman, Johannesburg, South Africa.

Proposed by W. J. Chalmers, T. S. Chalmers, Bruno V. Nordburg.

Born 1880, Epsom, Surrey, England. 1896-1902, Gen. Education in Private Schools in England. Technical in Woolwich, Polytechnic in Mechanical Engineering and Allied Subjects. 1896-99, pupil in shops of Messrs. Fraser & Chalmers Ltd. at Erith, Kent, Eng. 1899-1908, Draftsman, later Chief Draftsman, in Messrs. Fraser & Chalmers Works, Erith. 1908-16, Engineer in London and Johannesburg, S. Africa, at the office of Fraser & Chalmers Ltd. I have to do with the engineering of all mining business at Fraser & Chalmers of London with works at Erith, Kent, Eng., so far as S. Africa is concerned.

Present position: Engineer in London and Johannesburg, office of Fraser & Chalmers, Ltd.

Peter Clarence Chetney, Bisbee, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. W. Rait.

Born 1899, Necedah, Wis. 1908, Armour's Tech. School, Chicago, Ill. Necedah High School, Necedah, Wis. 1912, P. & R. G. R. R. Co., Pueblo, Colo. (Bridge Insp.). 1913, Welton Land Co., Pueblo, Colo. (Rock and Transitman), Colo. Fuel & Iron Co. (Rock & Transitman); Pueblo Irrigation Dist. (Transit and Levelman). 1915, Calumet & Arizona Mining Co., Eng. Dept.

Present position: Calumet & Arizona Mining Co.

Clarence L. Colbert, Midvale, Utah.

Proposed by Curtis Pigott, L. D. Anderson, C. A. Lemke.

Born 1878, Whitewater, Wis. 1895, Public School, Whitewater, Wis. 1897-1900, University of Wisconsin. 1895-97, Iona Bay Gold Mining Co., Telluride, Colo., Shift boss and amalgamator. 1900-06, Bingham Consolidated Smelting Co., Bingham, Utah, Converter and Blast Furnace Foreman. 1906-07, Utah Smelting Co., Ogden, Utah, in charge of plant. 1908-16, United States Smelting Co., Midvale, Utah, Asst. Supt.

Present position: Asst. Supt., United States Smelting Co., Midvale, Utah.

Walter Lorrain Cook, Chicago, Ill.

Proposed by Alton L. Dickerman, James G. Carleton, E. E. Olcott.

Born 1879, Burlington, Iowa. 1896-98, Manual Training School, Chicago.

1899-1903, Mass. Inst. of Technology, Boston, Mining Course, Degree S. B. 1903-04, Velardena Mining & Smelting Co., Mexico, Surveying. 1905, Mining Examination, Canada and Nevada. 1906-7, Supt. Promontorio Mine, Moctezuma, Sonora, Mexico, Travers Durkee Coppers. 1908, Leasing-Trail Mines, Cripple Creek, Colo. 1909-10, Leasing, Bassick Mine, Colo. 1911, Supt. Santiago & Anexas Mine, Tlalpahahua, Mexico. 1912-13, Mining Examinations in Mexico for S. Pearson & Sons and Exploration Co. of London and Mexico and other companies. 1914-15, Supt. Cia. Huanchaca de Bolivia, Pulacayo, Bolivia. 1915, Examining Copper Mines in Chile for Cia. Minera de Andacolla. 1916, Special work for Standard Fuse Corp. and American Ammunition Co..

Present Position: With Standard Fuse Co.

James Burton Cooper, Los Angeles, Cal.

Proposed by David Cole, B. Britton Gottsberger, Julius Bergman.

Born 1878, Chesterfield, Eng. 1883-94, Public School in Chesterfield, followed by Grammar School. 1894-96, Bowker's prep. school in Chesterfield. 1898-1901, Missouri Pacific Ry. Co., Fort Scott, Kans., as machinist. 1901-04, machinist or foreman for number of companies, approximately as follows: A. T. & S. F. Ry. at Argentine, Kans.; Mexican Central Ry., Chihuahua & Aguas Calientes; Ben Johnson, Supt. of Motive Power, Big Springs, Tex.; Haynes-Apperson Auto Co., Kokomo, Ind.; No. Pac. Ry. Co., Osawatomee, Kan.; Los Pintas Mining Co., Altar District, Sonora. 1905-06, Gen. Foreman Mech. Dept., Kans. City, Mex. & Orient Ry. Co., Fairview, Okla., under J. A. Foley, Supt. 1906-08, Selling, erecting and designing machinery and mine and mill plants in Mexico, for E. Dalby and Co. 1909-16, Sales Engineer for Allis-Chalmers Mfg. Co.

Present position: Sales Engineer, Mining Mach. Dept., Allis-Chalmers Mfg. Co.

Edward Wright Cowperthwaite, Warren, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. M. Rait.

Born 1889, Coal Creek, Colo. 1913, Colorado School of Mines, E. M. 1914, Mining of Elkton Mine, Cripple Creek, Colo.; Mining and Shift Boss, Calumet and Ariz. Mining Co., Warren, Ariz.

Present Position: Mining and Shift Boss, Calumet & Ariz. Min. Co., Warren, Ariz.

Frank Lewis Culin, Jr., Bisbee, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, E. E. Whiteley.

Born 1892, Seattle, Wash. 1907-10, Univ. of Ariz., Preparatory School. 1910-12, University of Ariz. 1913-15, University of Ariz., B. S. in Mining. 1915-16, University of Ariz., M. S. Sc. 1912-13, Copper Queen Consolidated Mining Co., Assay Office. 1914, New State Mining & Reduction Co. 1915 (Summer), Chisos Mining Co. 1916, Ariz. State Bureau of Mines (June and July), Copper Queen Consolidated Mining Co. (as Mucker). (Sept. 5) Calumet & Ariz. Mining Co. Engineering Office.

Present position: Calumet & Arizona Engineering Office.

Carleton Wilsey Cushman, Weedon, Province of Quebec, Can.

Proposed by Leland D. Adams, P. S. Morse, E. N. Englehardt.

Born 1886, Sacramento, Cal. 1900-04, Oakland High School. 1906-09, University of Cal. 1906, Refinery, Pittsburg-Silver Peak M. & M. Co. 1909-12, Butters Salvador Mines, Ltd., and Butters Divisadero Mines Ltd., Mine Sampler, Mill Shift Boss, and Cyanide Supt. 1912-14, U. S. S. R. & M. Co., Rainbow Mine, Mill Shift Boss, Mine Sampler and general underground work, asst. on exploration work. 1915, U. S. S. R. & M. Co., Mammoth Mine, Shift Boss.

Present position: Weedon Mine, Weedon, P. Q., Canada.

Lyndall P. Davidson, Anaconda, Mont.

Proposed by Louis V. Bender, Frederick Laist, Enoch A. Barnard.

Born 1894, Bozeman, Mont. 1915, Mont. State College, B. S. 1915-16, State Board of Health, Analyst. 1916, Anaconda Copper Mining Co., Assistant in Testing Department.

Present position: Assistant in Testing Department.

Arthur C. Dennis, 410 Clinton Building, Tulsa, Okla.

Proposed by Harry F. Wright, Jerry B. Newby, W. E. Dodge.

Born 1887, Des Moines, Iowa. 1906-07, Case School of Applied Science. 1908-11, University of Minnesota, B. A., 1912. 1914-15, University of Chicago, Graduate

Student in Geology. 1904-05, Assistant chemist, River Furnace, Cleveland, O. 1907-08, Chemist, Cuyuna Laboratory, Deerwood, Minn. 1912, Geologic Aid, U. S. G. S. 1913-16, Geologist with Gypsy Oil Co., Tulsa, Okla.

James John Denny, Cobalt, Ontario.

Proposed by R. B. Watson, Arthur A. Cole, James Johnston.

Born 1884, Sussex, England. 1897, Two years at Prof. Glover's Academy, Kingston, Ontario. Three years at High School of Mining, Kingston, Ontario (Chemistry and Mineralogy). 1903, Queens Medical College, Kingston, Ont. 1903-05, Asst. in Chemistry and Mineralogy at School of Mining, Kingston, Ont. 1905-06, Geological Survey, Rossland, B. C. 1906-08, Asst. to Mill Supt., Coniogos Mine, Cobalt, Ont. 1908-11, Chemist, Nipissing Mines, Cobalt, Ont. 1911-16, Metallurgist, Nipissing Mines, Cobalt, Ont.

Present position: Metallurgist.

George C. Dewey, Selby, Cal.

Proposed by E. A. Englehardt, E. D. Braden, P. S. Morse.

Born 1891, Washington, D. C. 1897-1910, Public Schools of Washington, D. C., through High School. 1910-14, Univ. of Penna., B. S. in Ch. E. 1910 (Summer), Tabulator 13th Census. 1911 (Summer), Asst. Assayer Bureau of the Mint, Treasury Dept. 1913 (Summer), Asst. Chem., Bureau of Chemistry, Dept. of Agriculture. 1914-15, Research Laboratory of the National Canner's Association, Washington, D. C. 1915, Chemist and Assayer, Selby Smelting & Lead Co., Selby, Cal.

Present position: Selby S. & L. Co., Selby, Cal.

Henry E. Doelle, Greenwood, B. C.

Proposed by Oscar Lachmund, F. S. Norcross, Jr., F. A. Thomson.

Born 1889, Arcadia, Wis. 1903-07, High School, Arcadia, Wis. 1908, Wis. Univ., Madison, Wis. 1909-12, State College of Wash., B. S. Mining. 1912, Millman, British Columbia Copper Co. Ltd., Napoleon, Wash. 1913, Asst. Engr., Stewart Mining Co., Kellogg, Idaho. Investigator B. & M. Red. Dept. Anaconda Copper Mining Co. 1914, Surveyor, British Columbia Copper Co., Ltd., Princeton, B. C. 1915, Engineer, Bishop Creek Milling Co., Bishop, Cal., Supt. Lone Star Mine, B. C. Copper Co., B. C.

Present position: Mine Supt., B. C. Copper Co., Ltd.

Kenneth H. Donaldson, Clifton, Ariz.

Proposed by Norman Carmichael, Arthur Crowfoot, John Kiddie.

Born 1888, Greenwich, Conn. 1900-07, Horace Mann Elementary and High Schools. 1907-12, Columbia School of Mines, N. Y. C. 1912-14, Sampler and Investigator at Boston & Montana Reduction Dept. of the Anaconda Copper Mining Co. 1914-15, Shift Boss, Wedge Roasters, Arkansas Valley Smelter, Leadville, Colo. 1915-16, Investigator and Asst. Testing Engineer and General Foreman Concentrator No. 4 of the Arizona Copper Co., Ltd., at Clifton, Ariz.

Present position: General Foreman Concentrator No. 4.

George A. Dornin, Baltimore, Md.

Proposed by Emil Gathmann, Joseph W. Richards, Howard Eckfeldt.

Born 1875, Norfolk, Va. 1892-99, Mech. Engrg., Lehigh Univ. 1899-1900, Inspector, U. S. Navy. 1900-04, Chief Inspector, Bethlehem Steel Co. 1905, Williamson Bros. Co., Philadelphia, Pa. 1906, Sayles Bleacheries, Saylesville, R. I.

Present position—1914 to date: Sales Engr., Gathmann Engr. Co.

John H. Doughty, Forty Fort, Pa.

Proposed by R. V. Norris, J. M. Humphrey, Paul Sterling.

Born 1876, Mattewan, N. Y. 1899, Mechanical Engineer, Cornell Univ. 1899, Fishkill Landing Machine Works—Draftsman. 1900, Exeter Machine Works—Draftsman. 1901, Lehigh & Wilkes-Barre Coal Co. 1902, Lehigh and Wilkes-Barre Coal Co., Asst. Outside Supt. 1905, Lehigh & Wilkes-Barre Coal Co., Mechanical Engineer.

Present position: Mechanical Engineer.

Frank Daniel Druding, Flat River, Mo.

Proposed by Ambrose E. Ring, H. G. Washburn, H. A. Guess.

Born 1883, New York City. 1898, Public Schools, N. Y. C. 1898-1902, College

of City of New York. 1902-06, Columbia School of Mines—Graduated with degree of Mining Engineer. 1906-08, Utah Copper Co., Bingham, Utah., Engineer on surface mining. 1908-09, Shift Boss and Foreman, No. 3 Mill, Federal Lead Co., Flat River, Mo. 1909-12, Mill Superintendent, St. Louis Smelting & Refining Co., St. Francois, Mo. 1912-16, In charge of Slimes Dept., Nevada Consolidated Copper Co. McGill, Nev. 1916, Superintendent No. 4 Mill, Federal Lead Co., Flat River, Mo.

Present position: Superintendent of No. 4 mill, Federal Lead Co., Flat River, Mo.

William Barnard Easton, Chicago, Ill.

Proposed by B. Britton Gottsberger, David Cole, Julius Bergman.

Born, 1863, Poughkeepsie, N. Y. 1883, Worchester Polytechnic Institute, Class of, 83. 1884-91, Draftsman with Martin P. Boss, San Francisco. 1895, Fraser & Chalmers, Chicago. 1902, Head Draftsman until 1907, then Gen. Mgr. 1903, Chief Engineer, Allis Chalmers. 1905-16, Chalmers & Williams, Chicago, as Chief Engineer and at present 2nd Vice President of that company.

Present position: 2nd Vice President of Chalmers & Williams.

John T. Ellsworth, 123 Elm Street, Anaconda, Mont.

Proposed by Louis V. Bender, Harry S. Ware, R. B. Caples.

Born 1887, Barre, Mass. 1900-04, Peterboro High School, Peterboro, N. H., 1904-08, Massachusetts Institute of Technology, Boston, Mass., Mining Engineering Course. 1908-09, Chemist, Boston Consolidated Gas Co., Boston, Mass. Nov., 1909-Nov., 1910, Timekeeper, buyer, etc. Hercules Mining Co., Joplin, Mo. Nov., 1910-June, 1913, Engineer for Joplin District, American Zinc, Lead & Smelting Co. June, 1913-Oct., 1914, Mine and Mill Supt., What Cheer Mining Co., Joplin, Mo. Oct., 1914-March, 1915, Furnace Tests, National Zinc Co., Argentine, Kansas.

Present position: Foreman Electrolytic Zinc Plant, Anaconda Copper Mining Co., Anaconda, Mont.

Edward William Engelmann, Hayden, Ariz.

Proposed by Louis S. Cates, David D. Moffat, Wm. T. McDonald.

Born 1887, Cape Girardeau, Mo. 1903-07, State Normal School, at Cape Girardeau, Mo. 1907-11, Missouri School of Mines, Rolla, Mo., B. S. in Mining Engr. 1908, Engr. Dept. St. L. & S. F. R. R., Chaffee, Mo. 1909 (Summer), Milling at Nevada Cons. Copper Co., McGill, Nev. 1910 (Summer), Mining and aerial tram work for Smuggler Union Mining Co., Telluride, Colo. 1911-13, Milling, Experimental and Metallurgical Dept., Utah Copper Co., Garfield, Utah. 1913-14, Supt. of Mill, Butte, Darymusee Mining Co., Deer Lodge, Mont.; also Experimental and Flotation Dept. at Butte & Superior Copper Co., Butte, Mont.

Present position: Flotation Engr. for Ray Consolidated Copper Co., Hayden, Ariz.

T. R. Finucane, Cobalt, Ontario, Can.

Proposed by R. B. Watson, Arthur A. Cole, James Johnston.

Born 1882, Rochester, N. Y. 1903, Cornell University. 1911-17, McKinley Darragh Savage Mines, Cobalt.

Present position: Manager, McKinley Darragh Savage Mines, Cobalt, Ont., Can.

David Lawton Forrester, Morenci, Ariz.

Proposed by Norman Carmicheal, Arthur Crowfoot, John Kiddie.

Born 1883, Racine, Wis. 1889-1900, Grade School and High School, Washington, D. C. 1902-03, University of Wisconsin, Madison, Wis. 1907-11, Missouri School of Mines, B. S., E. M. 1903-07, Prospecting: Mohave Co., Ariz. (6 mos.); Yucca Cyanide Mining Stamp Milling & Milling Co., Cedar, Ariz. (2 yrs.); Mining, Copper Queen Copper Co., Bisbee, Ariz. (1 yr.), Economic Mill, Victor, Colo.; Portland Gold Mining Co., Victor, Colo. 1907-11, Missouri School of Mines. 1911-13, Engineer Federal Lead Co., Flat River, Mo. 1913-14, Railroad Contracting, Construction, Wendling, Ore. 1914-15, Concrete Foreman, Montague O'Reilly, Portland, Ore. 1915-16, Ariz. Copper Co., Ltd., Morenci, Ariz., Flotation and Mill Testing.

Present position: Testing Engineer.

Jesse Butler Fox, Warren, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. M. Rait.

Born 1891, Louisville, Ky. 1905-06, High School, Globe, Ariz. 1906-08, Academy, Throop Polytechnic Institute, Pasadena, Cal. 1908-16, Mechanical Drawing and Metal Mining Course, International Correspondence Schools. 1908-11,

Chairman and Rodman for Sultan & Wayne, H. C. Hopkins, Van Wagenen & Hewes, Mining Engineers, also R. A. Smith, City Engr., and for Ariz. Eastern R. R. both Construction and Maintenance of Way, Globe, Ariz. 1911-12, Rodman and Instrumentman, Dept. of the Interior, Protecting Lands and Property, Imperial Valley, Cal.; J. A. Ockerson, Engineer in Charge, Yuma, Ariz. 1912-13, Rodman and Instrumentman, Ariz. Eastern R. R. Maintenance of Way, Globe, Ariz. 1913-14, Transitman's helper, Calumet and Ariz. Mining Co., Warren, Ariz. 1914-16, Transitman, Calumet & Ariz. Mining Co., Warren, Ariz.

Present position: Transitman, Calumet & Ariz. Mining Co.

Edward Elway Free, 1105 Madison Ave., Baltimore, Md.

Proposed by H. A. Megraw, Richard H. Vail, W. R. Ingalls.

Born 1883, Dagus Mines, Pa. 1901-02, Bellefonte Academy, Gen. 1902-06, Cornell University, Mainly chemical and engineering, Degree of A. B. 1914-16, Johns Hopkins University, Graduate work, no degree as yet, candidate for Ph. D. 1906-07, Chemist, Univ. of Arizona. 1907-12, U. S. Dept. of Agriculture, mainly geological work in connection with potash deposits. 1912-14, Member of firm of Gould, Free and Ash, San Francisco; geological and chemical work on potash. 1914-16, Consulting work and work at Johns Hopkins, mainly technology and geology of potash. Above work on colloids.

Present position: Consulting engineer in private practice.

Frank Garratt, Latrobe, Pa.

Proposed by R. J. Wysor, Fred Crabtree, J. S. Unger.

Born 1886, Birmingham, England. 1905-10, Carnegie Inst. of Technology, B. S. in Met. Engrg. Until 1915, Chief Chemist, Firth-Sterling Steel Co., McKeesport, Pa. Also metallurgical research work for above concern.

Present position—1915 to date: Supt., Latrobe Electric Steel Co.

John C. Gaul, Butte, Mont.

Proposed by Chauncy L. Berrien, C. W. Goodale, Reno H. Sales.

Born 1886, Calumet, Mich. 1900-04, Calumet High School. 1905, University of Notre Dame. 1907, Mich. College of Mines. 1908-12, Colo. School of Mines, Degree of E. M. 1906, Hancock Consolidated Mining Co., Engineering Dept. 1912, Britannia Mining & Smelting Co., Miner. 1913, Hancock Consolidated Mining Co., Miner. 1914, Anaconda Copper Mining Co., Leonard Mine, Miner, Shift Boss. 1915, Anaconda Copper Mining Co., Berkeley Mine, Asst. Foreman.

Present position: Foreman, Berkeley Mine.

Oscar Watson Gilman, Lowell, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, E. E. Whiteley.

Born 1883, Nobnosti, Mo. 1901, First experience underground with Copper Queen, Bisbee, Ariz. 1902, Mucker and Miner, Calumet & Ariz. 1903, Miner, Timberman, Pumpman, and Shift Boss. 1905-06, Miner and Timberman. 1907, Miner, O. D. Keestone, Globe, Ariz. 1905, Leopold Copper Co., N. Mexico, Shift Foreman. 1907-16, Calumet & Ariz., timberman, shift boss, shift foreman and general foreman.

Present position: General Foreman.

Charles Marshall Gravatt, Port Royal, Va.

Proposed by J. S. Grasty, John B. Guernsey, Thomas L. Watson.

Born 1887, Erie, Pa. 1906-08, Locust Dale Academy. 1902-12, Univ. of Virginia. 1913-15, Univ. of Virginia, M. E. 1912-13, Miner, Portland and El Paso Mines, Cripple Creek, Colo. 1913, Assayer, Hazel M. & M. Co., Van Horn, Tex. 1914, Field work, Virginia Geol. Survey. 1915, Mapping, Virginia Geol. Survey. 1916, Engr., Holladay Mine, Spottsylvania Co., Va.

Present position: Prospecting under direction of Dr. J. S. Grasty.

George Gary Griswold, Jr., Columbia Gardens, Butte, Mont.

Proposed by W. N. Rossberg, G. E. Sheridan, J. O. Proctor.

Born 1890, Denver, Colo. 1904-08, High School, Salida, Colo. 1908-09, Leasing-Lunnig, Nev.; Surface Boss, Cerro Gordo Mine, Keeler, Cal. 1909-10, Colo. School of Mines, Golden, Colo. 1910-11, Miner, Monarch Mining Co., Leadville, Colo. 1911-14, Colo. School of Mines, E. M. 1914-15, Timekeeper, Timber Butte Milling Co., Butte, Mont. 1915-16, Mining Engineer.

Present position: With Timber Butte Milling Co.

Alfred McL. Hamilton, Sasco, Ariz.

Proposed by Kuno Doerr, J. Kruttschnitt, Jr., P. A. Mosman.

Born 1879, Montreal, Canada. 1905, McGill University, B. Sc. 1905, Chemist, Anaconda. 1906, Asst. Supt., Aguascalientes, Mexico. 1907, Asst. Supt., Velardena, Mexico. 1908, Supt., Velardena, A. S. R. Co. 1916, Supt., Sasco, Ariz., A. S. R. Co. Present position: Supt., Sasco Smelter, A. S. R. Co.

O. R. Hamilton, Lansing, Mich.

Proposed by William Kelly, F. W. McNair, O. C. Davidson.

Born 1882, Newark, Ohio. 1899, Newark High School. 1906, Mich. College of Mines, E. M. 1906, Engineer with Cerro de Pasco Mining Co., Cerro de Pasco, Peru. 1907, Sociedad de Minas de Cobre de Cutter Cove, Magallanes. 1908, Sociedad de Minas de Cobre de San Bartolo, Antofagasta, Chile. 1909, Engineer, Stambaugh Trop Iron Co., Mich. 1910-11, Pickands Mather Iron Co., Mich. 1911-12, Spring Valley Iron Co., Palatka, Mich. 1912, Fee agent at Iron River, Mich. 1913-16, Mining Engineer with Michigan Geological Survey and Michigan Board of State Tax Commissioners.

Present position: Mining Engineer, State of Michigan.

Roy Carlton Higley, Detroit, Mich.

Proposed by Francis Nicholson, Roy H. Allen, Franklin W. Smith, Basil Prescott.

Born 1885, Lewistown, Mont. 1902-03, Vashon College, Wash. 1908-09, Valparaiso Univ., Valparaiso, Ind. 1909-10, Grad., scientific course, also practical mechanical education. 1904, Mill Shift Boss, Mina Grande M. & M. Co., Durango, Mex. 1905, Mine Foreman, Mina Grande M. & M. Co., Dgo., Mex. 1906, Mine Foreman, Santo Nino, Dgo., Mex. 1907, Mine Supt., Santo Nino, Dgo., Mex. 1910, Mine Supt., El Conde M. & M. Co., Dgo., Mex. 1911, Mine Supt., Mex. West Coast Mines Co. 1912, Mine Foreman, Presena Mine, Parral, Mex. 1913, Mine Foreman, Presena, Alfarena, Morena Mines, Parral, Chih., Mex. 1914, Mine Supt., Presena Group of Mines, Alvarado Min. & Mill. Co., Parral, Chih., Mex. 1915-16, Supt. of Mines, Alvarado Min. & Mill. Co.

Present position: Supt. of Mines.

James Madison Hill, Washington, D. C.

Proposed by F. L. Ransome, Davis White, D. F. Hewett.

Born 1884, Chicago, Ill. 1902-06, University of Chicago, Degree of S. B. 1907-16, With U. S. Geological Survey.

Present position: Associate Geologist, U. S. G. S.

Jan van Houten, Raton, New Mexico.

Proposed by Charles A. Chase, Carl Scholz, John V. N. Dorr.

Born 1867, Groningen, Holland. Common and High Schools, Holland, until 1886. Maxwell Land & Grant Co., Raton Coal & Coke Co., St. Louis Rocky Mt. & Pacific Co. Present position: President, St. Louis Rocky Mt. & Pacific Co.

William Otis Hotchkiss, Madison, Wis.

Proposed by F. L. Ransome, Arthur C. Spencer, B. S. Butler.

Born 1878, Eau Claire, Wis. 1903, B. S. in Gen. Eng., Univ. of Wis. 1908, C. F. Univ. of Wis. 1916, Ph. D., Univ. of Wis. 1901, Recorder with Topographer, U. S. G. S. 1902, Geologist and Engineer, Donora Mining Co. (Union Steel). 1903, Leith Exploration Party, Ontario; Chief of party, Oscar Rohn, Baraboo Iron District. 1904, Instructor, Geol. Dept., Univ. of Wis. 1906, Economic Geol. for Wis. Geol. and Natural History Survey, also Chief of Highway Division. 1908, State Geologist of Wis. 1911, Ex. officio member of State Highway commission.

Present position: State Geologist.

Myol Lamont Jacobs, 711 Prospect Ave., Bethlehem, Pa.

Proposed by Frank D. Carney, W. L. Cumings, E. O. C. Acker.

Born 1885, Warren, Pa. 1899-1902, York, Pa., High School; Warren, Pa., High School; Marietta, Ohio, High School (Graduate 1902). 1906, Mercersburg Academy (Graduate). 1906-10, Lehigh University, Graduate, E. M. Degree. 1903-05, Burma Oil Co., Pipe Line and Misc. Const. Work. 1910-12, Jacobs & Davies, Tunnel Work; Astoria Tunnel. 1912-13, Mexican Light & Power Co., Tunnel and Dam Work. 1913-16, N. Y. Municipal Ry. Corp., Elevated R. R. Construction. 1916, T. H. Clement & Co., Engineers. 1916, Bethlehem Steel Co.

Present position: Supt. of Quarries, Bethlehem Steel Co.

H. G. Humes, Park City, Utah.

Proposed by Geo. S. Krueger, O. N. Friendly, James Humes.

Born 1882, Youngstown, Ohio. 1901-03, Butte School of Mines. 1903-04, Mass. Institute of Technology. 1904-08, Columbia University School of Mines. 1908, Assistant Engineer of Austin Manhattan Consolidated Mng. Co. 1909, Superintendent, Western Utah Copper Co.; Captain Duncan MacVichie, General Manager, Salt Lake City, Utah. 1910, Purchasing Agent, Ohio Copper Co.; Mr. Colin McIntosh, General Manager, Salt Lake City, Utah. 1912, Superintendent, King Solomon Copper Mng. Co., Ltd., Cowichan Station, Vancouver Island, B. C. Present position: Gen. Foreman, Silver King Coalition Mines Co., Salt Lake City, Utah.

Cunnard E. Johnson, Morenci, Ariz.

Proposed by Norman Carmichael, Arthur Crowfoot, John Kiddie.

Born 1886, Kalmar, Sweden. 1892-1902, Grammar Schools at Sand Coulee, Stackett, and Great Falls, Mont. 1912-16, Missouri School of Mines and Metallurgy, Rolla, Mo., B. S. in Metallurgy. 1902-12, Anaconda Copper Mining Co. B. and M. Red't Dept., Great Falls, Mont., Asst. Assayer, Asst. Chemist. 1913 (Summer), Canadian Copper Co., Copper Cliff, Ontario, Laborer, Reverb. Dept. Chemist. 1914 (Summer), Anaconda Copper Mining Co. B. & M. Red't Dept. Great Falls, Mont., blast furnace helper. 1915 (Summer), Anaconda Copper Mining Co., B. and M. Red't Dept. Great Falls, Mont., Asst. Chemist.

Present position: Chemist, Dept. of Concentration, Ariz. Copper Co., Morenci, Ariz.

John Dow Johnson, Treadwell, Alaska.

Proposed by P. R. Bradley, L. Wernecke, R. G. Wayland.

Born 1888, Valparaiso, Ind. 1910-13, University of Washington, Seattle, Wash. 1908-11, Chief Clerk, Bridge Engineer, Oregon Short Line & S. P. R. R., Pocatello, Idaho. 1911-13, Student, with vacation field work in Bridge Office. June to Sept., 1913, Surveyor's Asst., Alaska Treadwell Gold Mining Co., Treadwell, Alaska. Sept., 1913, to Sept., 1914, Instrumentman. Sept., 1914, Mine surveyor.

Present position: Mine Surveyor, Treadwell, Alaska.

Robert R. Landon, Cebu, P. I.

Proposed by J. Clayton Nichols, A. H. Jones, Fred. H. Penn.

Born 1873, Milton, Vermont. 1896, Ames, Iowa, B. M. E. 1908-16, President and General Manager, Iloilo Electric Co., Iloilo, P. I. At present, President and General Manager Visayan Electric Co., Cebu, Philippine Islands.

Present position: Mine owner and operator.

Sidney Sherman Lang, Houghton, Mich.

Proposed by J. Parke Channing, C. H. Benedict, E. W. Walker.

Born 1881, Houghton, Mich. 1904, E. M. and B. S., Michigan College of Mines. 1904-07, Burro Mt. Copper Co. (New Mexico). 1907-08, Peregrina Mining Co. (Mexico). 1908-09, Guanyuato Development Co. (Mexico). 1909-10, Joseph Qualey & Co. (Mexico). 1910-11, Dr. F. S. Pearsons (Alaska). 1911-13, General Development Co. (Mexico). 1913-16, Naumheag Copper Co. (Michigan).

Present position: Superintendent, Naumheag Copper Co.

Charles Lehmann, Société des Mines de Cuivre de Naltagua, El Monte (Chili).

Proposed by Mark R. Lamb, Thomas T. Read, Wm. H. Shearman.

Born 1890, Paris. 1900-07, Dulwick College, London. 1907-11, University College, London University. 1911-12, Royal School of Mines, London (B. Sc. Engineering), London University. 1912-16, Société des Mines de Cuivre de Naltagua.

Present position: Manager.

Albert Leonarz, Lansford, Pa.

Proposed by Edwin Ludlow, R. E. Hobart, J. B. Warriner.

Born 1876, Tholey, Germany. 1882-90, Public School at Tholey, Germany. 1890-92, Brlders Academy for District of Kaiserslautern, Germany. 1893-94, Technicum, Academy at Mittweida, Germany. 1895-96, Polytechnicum Academy at Darmstadt, Germany. 1896-97, Technicum Academy at Ilmenau, Germany—Degree: Electrical Engineer. 1897-99, Electricitats Aktiengesellschaft, (Schuckert

& Co.), Nurnberg, Germany. 1899-1901, Alois Zettler, Munich. 1902-06, A. E. G. (Allgemeine Electricitats gesellschaft) Branch at Stuttgart, Ger. 1907-09, Berlin Electric Co., New York. 1909-10, Mexican Coal & Coke Co., Las Esperanzas, Mex. 1910-11, Otto Gas Engine Works, Mexico City. 1912-16, Lehigh Coal & Navigation Co., Lansford, Pa.

Present position: Supt. of preparation.

Norman David Lindsley, Box 872, Morenci, Ariz.

Proposed by Frank Ayer, Rienzi Macfarlane, P. B. Scotland.

Born 1884, Seattle. 1906, Graduated from Broadway High School, Seattle, Washington. 1911, Graduated from Washington State College, with Degree of B. S. in Mining Engineering. 1908, "Mucker" Granby Consolidated Mining & Smelting & Power Co. Phoenix, B. C. 1909, Hand Miner, Alaska Reliance Gold Mining Co., Juneau, Alaska. 1910, Draftsman, U. S. Forest Service, Sand Point, Idaho. 1911, Installed Irrigation System for K. B. Weulfingen, Lakeside, Washington. 1912, Transitman for Kellogg Lumber Co., Entiat, Washington. 1913, Assayer, Engineer, Dividend-Lakeview Consolidated Gold Mining Co., Ltd., Osyoos, B. C. (P. O. Oroville, Washington). 1914, Howard Flat Irrigation District, Chelan, Washington. 1915, Assistant to Chief Sampler, Arizona Copper Co., Ltd., Morenci, Ariz.

Present position: Chief Sampler of Leasing Dept., Arizona Copper Co., Morenci, Ariz. Formerly member of Affiliated Student Society, Washington State College.

Roy Nichols McBride, Miami, Ariz.

Proposed by B. Britton Gottsberger, R. B. Yerxa, F. W. Solomon.

Born 1889, Antrim, Ohio. 1906, Common Schools, Antrim, Ohio. 1906-09, Muskingum College, New Concord, Ohio. 1909-11, New Mexico School of Mines, Socorro, N. Mex. 1911-12, Missouri School of Mines and Metallurgy, Rolla, Mo. 1913-14, Missouri School of Mines and Metallurgy, Rolla, Mo. Received degree of B. S. in Mine Engineering. 1912-13, Millman and Underground Work at Miami, Ariz. 1913, 6 mos., Diamond Drill Operator, International Diamond Drilling Co., Princeton, B. C. Can., 1914-15, 9 mos. with Sullivan Machinery Co., Claremont, N. H., as demonstrator salesman. 1915, Several months with Inspiration Con. Copper Co., as Flotation Experimental Engineer. 1915-16, Smelter Representative for Miami Copper Co., at International Smelter, Miami, Ariz.

Present position: Smelter Representative.

Harry E. McDonnell, South Chicago, Ill.

Proposed by H. A. Brassert, Walther Mathesius, H. P. Howland.

Born 1881, Chicago, Ill. 1904, Graduated University of Michigan, B. S. in Mechanical Engineering. 1904-06, Estimating Dept., Newport News Shipbuilding & Dry Dock Co. 1906-07, Steam Eng. Dept., Ill. Steel Co. So. Wks. 1907-09, Blast Furnace Dept., So. Wks. 1909-15, Supt., Blast Furnace, Mines Wks., Supt., Blast Furnace, Milwaukee Wks. 1915-16, Asst. Supt., 5-10 Blast Furnaces, So. Wks.

Present Position: Asst. Supt. 5-10 Blast Furnaces, So. Wks., Ill. Steel Co.

Charles Alexander Mechesney, Pittsburgh, Pa.

Proposed by Samuel A. Taylor, W. E. Fohl, Howard Evanson.

Born 1875, New Derry, Pa. 1898, Penna. State College, B. S. 1898-1900, H. C. Frick Coal & Coke Co. 1901-15, Continental Coal & Coke Co. 1905-16, Pgh. Rwy. Co., Phila. Gas. Co.

Present position: Engineer for Phila. Gas. Co.

Robert James Morgan, c/o Spassky Copper Mine, Siberia.

Proposed by Edward T. McCarty, Herbert C. Woolmer, J. M. Wardell.

Born 1886, Dunedin, New Zealand. 1904-06, Mining and Metallurgical Course at Waihi School of Mines. 1906, Passed New Zealand Government Examination as Battery Superintendent. 1907, Licensed in the Dominion of New Zealand as an Assayer. 1907-10, Auckland University School of Mines, Metallurgical Course. 1904-07, Employed by the Waihi Gold Mining Co., Ltd., in their Assay, Refining and Cyaniding Dept. 1910-11, Employed as Assistant to Dr. J. M. Bell on a Geological Expedition to New Caledonia. 1911-12, Employed by the Electrolytic Refining & Smelting Co. of Australia at Port Kembla. Foreman of Tank House. 1912-16, Employed by the Spassky Copper Mine Ltd. as Assistant Metallurgical Engineer at their Smelter at Spassky, Akmolinsk, Siberia.

Present position: Assistant Engineer at the Spassky Co., New Concentrator at Sara Soo, Siberia.

Harvey Seely Mudd, Los Angeles, Cal.

Proposed by S. W. Mudd, F. A. Keith, Robt. E. McConnell.

Born 1888, Leadville, Ariz. 1906-08, Stanford University. 1908-12, School of Mines; Columbia, E. M. Degree. 1912-13, Engineer, Shattuck-Ariz. Copper Co. 1913-16, with Seeley W. Mudd.

Present position: With Seeley W. Mudd.

Fred August Nathan, Bisbee, Ariz.

Proposed by Philip D. Wilson, D. M. Rait, Ira B. Joralemon.

Born 1883, Reno, Nev. 1900, Graduated from Reno High School. 1904, B. A. Degree, University of Nevada. 1908, Mining Dept., Calumet & Ariz. Mining Co. 1911, Accounting Dept., Calumet & Ariz. Mining Co. 1913, Asst. Purchasing Agent, Calumet & Ariz. Mining Co.

Present position: Asst. Purchasing Agent.

George E. Nicholson, 3512 Locust Street, Kansas City, Mo.

Proposed by W. R. Ingalls, Ludwig Vogelstein, F. Y. Robertson.

Born 1861, Kings County, N. Y.

I have been identified with the mining and smelting of zinc ores since 1878, mainly in the Middle West; likewise in the construction of lead and zinc smelting furnaces for a good many years. Have built a number of zinc smelting plants for others, and built and operated for ourselves, one plant in Missouri, three in Kansas, and four in Oklahoma, and have just completed a large plant at Kusa, Oklahoma, where we have approximately 10,000 retorts in operation, all of which are under our active management.

Samuel Willard Ogden, East Chicago, Ind.

Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.

Born 1856, Rushford, N. Y. Common schools. Thirty-four years with the Grasselli Chemical Co.

Present position: Genl. Mgr., Grasselli Chemical Co., Grasselli, Ind.

LeRoy Dennison Osborne, Lang, Cal.

Proposed by Philip Wiseman, S. W. Mudd, Frank A. Keith.

Born 1884, Denver, Colo. 1891-98, Grammar School. 1898-1902, High School. 1902-04, University of Colorado. 1904-06, Various underground and milling experience, Camp Bird & Barstow Mines, Ouray, Colo. 1906-08, Underground, later Foreman and then Supt., Ely Witch Copper Co., Ely, Nev. 1909, C. W. Terry, Sonora, Cal. 1910-12, Chemist, Sterling Borax Co., Lang, Colo. 1912-13, Underground Surveyor, Braden Copper Co. 1913-15, Field Eng., Guggenheim Ex. Co., Chile, S. A.

Present position: Asst. Supt., Sterling Borax Co.

Hugh Park, Cobalt, Ont., Can.

Proposed by R. B. Watson, Arthur A. Cole, James Johnston.

Born 1879, Muskegon, Mich. 1897-98, University of Mich. 1903-06, Stanford Univ. 1897-1906, Various: Utah, California, British Columbia. 1907-08, Supt., Nipissing Mining Co. 1908-16, Mgr. Nipissing Mining Co.

Present position: Mgr. of the Nipissing Mining Co.

Charles James Price, 1430 Polk Street, Topeka, Kansas.

Proposed by Wm. D. Gordon, W. S. Harrison, H. H. Utley.

Born 1857, Tumbidge, England. Common School education only, Monticello, N. Y., and Atchison, Kansas. From 1878 to 1896 with the Homestake Mining Co., South Dakota, as Miner, Shift Boss, General Foreman and Superintendent. 1896-1908 with Wehrner, Beit & Co. on the Witwatersrand in South Africa as Manager, General Manager and Consulting Engineer.

Present position: Consulting for the Bank Commission of Kansas, also Consulting Engineer.

Henry R. Putnam, 39 West 67th Street, New York City.

Proposed by Sidney J. Jennings, F. Y. Robertson, F. F. Colcord.

Born 1886, East Orange, N. J. 1904-09, Geological Option of Mining Course at Massachusetts Institute of Technology. 1909-16, In employ of the United States

Smelting, Refining & Mining Co. practically continuously having been employed principally in operation and exploration.

Present position: Attached to Office of Manager of Metal Sales, U. S. S. Co., N. Y. C.

James Childs Rea, Pittsburgh, Pa.

Proposed by Taylor Allderice, K. Taylor, William H. Rea.

Born 1882, Pittsburgh, Pa. 1904, B. S. Degree, Princeton. 1905-06, Asst. Supt. of Sheet and Plate Mills, Colonial Steel Co., Pgh. 1906-08, Asst. Supt. of Rolling Mills, Oliver Iron & Steel Co., Pgh. 1908-15, Asst. Supt. and Supt. in various departments of the Oliver Iron & Steel Co. 1915-16, Manager, Oliver Iron & Steel Co.

Present position: Manager, Oliver Iron & Steel Co.

David J. Roberts, El Paso, Texas.

Proposed by Kuno Doerr, J. J. Ormsbee, P. A. Mosman.

Born 1868, Waushara Co., Wis. Common Schools, Colorado, and Commercial Dept. of Denver University. 1891-1902, Engaged in merchandising with my father. 1893-95, Railroad work in Mexico, Mexican Central R. R. 1896-1916, El Paso Smelting Works, Clerical & Asst. Mgr. 1911-16, Ore Purchasing Agent, El Paso Smelting Works, also American Smelting & Refining Co.

Present position: Ore Purchasing Agent.

Walter Brewster Robinson, Huntington, Ore.

Proposed by Robt. M. Betts, Paul W. Gaebelein, Felix E. Wormser.

Born 1879, Buffalo, N. Y. 1904, Asbestos Mines, Quebec. 1905-06, Assayer, North Pole Mine. 1907-10, Development on Copper Prospects, Snake River. 1911-15, Bookkeeper, Surveyor, Cornucopia Mines Co.

Present position: Supt. Snake River Mines, Huntington, Ore.

Fred Sandtner, Lowell, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. N. Rait.

Born 1871, Austria. Self Educated. 1881-85, Miner in coal mines in Ohio, Pennsylvania and Colorado. 1885-1902, Timberman in Colo. Metal Mines. 1903-04, Shift Boss, Oversight Mine, Cananea Cons. Copper Co., Cananea, Sonora. 1905-06, Foreman, same Mine. 1907-08, Foreman of Leases, Florence Mine, Goldfield, Nev. 1909-10, Foreman and Supt., Santa Rosalia Mine, Sonora, Mex. 1911-1916, Junction Mine, Calumet & Ariz. Mining Co. as Timberman, Shift boss and Mine Foreman.

Present position: Mine Foreman.

John Theron Scott, Hammond, Ind.

Proposed by G. P. Hulst, Fred P. Clark, R. Ruetschi.

Born 1893, Terre Haute, Ind. 1910, Finished High School, Terre Haute, Ind., having taken mathematics course. 1914, Grad., Rose Polytechnic Institute, Terre Haute, Ind., B. S. in Chem. Engrg. 1914-16, Indiana Laboratories Co., Hammond, Ind.

Present position: Chem., Indiana Laboratories Co.

Charles Albert Smith, Bisbee, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. M. Rait.

Born 1888, Malden, Mass. 1906, Diploma, Medford, Mass., High School. 1910, A. B. Degree, Bowdoin College. 1913, B. S. in Mining, Mass. Inst. of Technology. 1913-14, Miami Copper Co., General underground work, mucking, mining and timbering. 1914-15, San Nicholas Copper Co., prospecting, assessment and shallow development work. 1915-16, Calumet & Ariz. Mining Co., underground engineering and stope work.

Present position: Stope Engineer.

Warren S. Smith, New York, N. Y.

Proposed by Rowland F. Hill, Robt. K. Painter, Chas. P. Berkey.

Born 1891, Seattle, Wash. 1904-08, Seattle High School (Liberal Arts). 1908-12, University of Washington, B. S. in Geology and Mining. 1912-13, Columbia Univ., A. M.; Major subject, Geology. 1913-15, Columbia Univ., Ph. D.; Major Subject, Geology. 1915, Geologist, assisting Dr. Berkey on Board of Water Supply work for the City of New York. 1915-16, Geologist, General Chemical Co., Bauxite,

Ark. 1916, Geological work in Ontario, Canada. At present doing exploratory work at Mokomon, Ont., on Sulphite Ores.

Present position: Geologist, General Chemical Co.

Harry M. Spoor, Minas Tofo, La Higuera, Coquimbo, Chile.

Proposed by C. A. Buck, W. L. Cumings, Hubert Merryweather.

Born 1878, San Francisco, Cal. To 1897, Los Angeles, Cal., Public and High Schools. 1912-13-14, Shovel-runner and Gen. Foreman for Maney Bros. & Co., Daniek Con. Co., and various contractors in California and Nevada on railroad work. 1909-10-11, Steam-shovel runner for Mexican Light & Power Co., Necaxa, Mexico. Eight months in 1909 Mining and Timbering for Superior-Nevada M. & M. Co., Crown King, Ariz. 1907-08, Mining and Timbering for Quartette Mining Co., Searchlight, Nev. 1904-05-06, Cyanide Foreman. Mine Foreman, Contractor and Miner for various companies in and around Searchlight, Nev.

Present position: Supt. of Mine, Bethlehem-Chile Iron Mines Co., Minas Tofo, Chile.

Edgar McCullah Stewart, Lang, Cal.

Proposed by Philip Wiseman, S. W. Mudd, Frank A. Keith.

Born 1882, Denver, Colo. 1890-98, Grammar School. 1898-1901, High School. 1901-04, Colo. School of Mines. 1904, J. Geo. Leyner Engineering Wks. Co. 1905-06, Portland Gold Mining Co. 1906-07, Cumberland Ely Copper Co. 1907, Nevada Consolidated Copper Co. 1907-08-09, Ray Consolidated Copper Co. 1909-16, Sterling Borax Co.

Present position: General Supt., Sterling Borax Co.

Ichiro Suzuki, Mitsubishi Okuyama Mine, Japan.

Proposed by Shiuji Harada, I. Miyasaki, M. Yamashita.

Born 1882, Menuuma, Saitama-ken, Japan. 1909, Finished the whole course of the Mining and Metallurgical Dept. at the Technological College of the Imperial Univ. in Tokyo, Japan. 1909, was employed by Mitsubishi-Goshikaisha as a mining engineer and the same at the present.

Present position: Mitsubishi-Okuyama Mine.

William Edmund Thomas, Austin, Tex.

Proposed by E. Harms, Henry Bornhoft, C. Q. Schlereth.

Born 1885, Austin, Tex. 1907, Graduate E. M. of University of Texas. 1907, Cia. Carbonifera de Sabinas, S. A. 1907-10, Cia. Metalurgica de Torreon, S. A. 1910-16, Cia. Minera de Peñoles, S. A. 1916, Cia. Metalurgica de Torreon, S. A.

Present position: Acting Asst. Supt.

John W. Thompson, Salt Lake City, Utah.

Proposed by J. M. Callow, Ernest Gayford, F. G. Moses.

Born 1875, Quincy, Plumas Co., Cal. 1898, School of Mines, Univ. of Nevada, Degree, B. S. 1898-1901, Milling with Silver King Coalition Mining Co., Park City, Utah. 1901-03, Gen. Supt., Checkmate Mining Co. Ltd., Pearl, Idaho. 1903-04, Kearns Kuth Mining Co., Park City, Utah. 1904-13, Silver King Coalition Mining Co., Park City; Mill Supt. from Jan., 1906, to Oct. 1913. 1913-14, F. E. Marcy as Mechanical Engineer. 1914, Mill Supt., Nat. Copper Mining Co., Mullan, Idaho. 1914-16, Asst. to J. M. Callow.

Present position: Asst. to J. M. Callow.

Lester Charles Uren, Berkeley, Cal.

Proposed by Frank H. Probert, Ernest S. Hersam, Andrew C. Lawson.

Born 1888, Grass Valley, Cal. 1907, Cal. School of Mechanical Arts., San Francisco. 1911, B. S. in mining, Univ. of Cal. 1903, Asst. in Mineral Surveys (2 months). 1904, Asst. in Assay Laboratory (3 months). 1905-06, Mechanic in Mine Machine Shop (5 months). 1907, Making concentrator tests and Extra Man in Mill (2 months). 1908, Concentrator Tests (2 months). 1909, Feeder Tender and Tramway Attendant (2 months). 1910, Frue Vanner Attendant (3 months). 1911-12, Accountant and general mining engineering (16 months). All of above with Eagle Shawmut Mining Co., Shawmut, Tuol. Co., Cal. 1912-14, Asst. in Mining and Mechanical Draftsman, Univ. of Cal. 1914, Instructor in Mining.

Present position: Instructor in Mining, Univ. of Cal.

Alexander D. Walker, Independence, Colo.

Proposed by C. Q. Payne, Justin H. Haynes, H. P. Nagel, Jr.

Born 1888, Coal Creek, Colo. 1898-1903, Public Grammar Schools. 1903-07, High School, Cripple Creek, Colo. 1907-12, Technical Education, Colo. School of Mines, Golden, Colo.; Degree, Mining Engineer. 1902-05, Sorting Ore, Trammings and General work, Hull City Mine, Independence Con. Town & M. Co. 1905-11, General underground work (Machine, Hoist Timbering). 1912-13, Examination, Plain and Mine Surveying for T. R. Countryman, Cripple Creek, Colo. 1913-16, Asst. Supt., Vindicator Cons. G. M. Co., Independence, Colo. 1916, Mine Supt., Vindicator Cons. G. M. Co.

Present position: Asst. Supt., Vindicator Cons. G. M. Co., Independence, Colo.

Chas W. Walters, New York City.

Proposed by G. P. Bartholomew, Edwin S. Berry, C. S. Witherell.

Born 1888, Casselton, N. D. 1906, College preparatory education, Casselton High School, Casselton, N. D. Univ. of Minnesota, Minn. School of Mines, Graduation, 1911; Degree, Engineer of Mines. 1909-11, Intermittent employment in mines at Ely, Nev., Grass Valley, Cal., Bisbee, Ariz., and the Coeur d'Alenes, Idaho, as miner, timberman, etc. 1911-12, Engineer in charge Radersburgh Dis., Mont., for De Lome Gold Mine Co. and Radersburgh Gold Mines Co., both of Butte, Mont. 1912-14, Engineer in charge of construction, foreman and superintendent of construction, Medicine Hat Pottery Co., Ltd., Medicine Hat, Alta, Canada. Plant construction, research work in ceramics and examinations of clay, shale and limestone properties in Alberta, British Columbia and Washington, U. S. A. 1914-15, Anaconda C. M. Co., Great Falls, Mont. Draftsman, B. & M. Smelter; Engineer U. S. R. S., Cody, Powell and Cowley, Wyo., tunnel work, hydraulic installation, canal location and construction. 1916, Engineer of Construction, E. I. du Pont de Nemours & Co., Gibbstown, N. J., and Williamsburg, Virginia. 1916, With Guggenheim Bros. N. Y. C.

Present position: In office of Consulting Mining Engineer.

Paul Sherman Warriner, Wilkes-Barre, Pa.

Proposed by R. V. Norris, J. M. Humphrey, Paul Sterling.

Born 1887, Montrose, Pa. 1906, Completed prep. course at Montrose High School. 1906-07, Lehigh Univ. M. E. and Ch. E. Courses. 1908-09, Study of Mining Engineering with I. C. S. of Scranton. 1908-09, with John Monks & Sons, Contracting Engineers of New York City, as Asst. to Supt. constructing B. & O. R. R. bridge at Havre de Grace, Md. (Operations suspended in 1908). Jan. to June, 1909, Asst. to Carl S. Camp, Contracting Engineer of Phila., Pa. Completed large concrete arch bridge over canal at Trenton, N. J. June to Sept., 1909, Carpenter on house construction. 1909-10, Coal inspection Dept. of L. V. Coal Co. at Wilkes-Barre, Pa. 1910-12, Asst. Chief Coal Insp., L. V. Coal Co. (Principally engaged in prep. of coal.) 1912-16, Engineer Corps, L. V. Coal Co.

Present position: Div. Engr., L. V. Coal Co.

Walter Lucius Whitehead, Institute of Technology, Cambridge, Mass.

Proposed by W. Lindgren, Carle R. Hayward, L. C. Graton.

Born 1891, Pittsburgh, Pa. To 1906, Public Schools, Detroit, Michigan. 1906-09, Waltham Academy, Waltham, Mass. 1909-13, Mass. Institute of Technology, Boston, Mass. 1913, Degree of S. B., Mass. Inst. of Tech., Mining Eng. 1913-16, Graduate student, Mass. Inst. of Tech. 1917, Candidate for degree of Ph. D. in Mining Geology. 1913-14, Assistant, Mass. Inst. Tech. 1914-15, Assistant to W. Lindgren. 1914-15, Aide, U. S. Geological Survey. 1915-16, Research, Secondary Enrichment Investigation. 1916, Mining Geologist with A. R. Whitman, Cobalt, Ont. 1916, Research, Mass. Inst. of Technology.

Present position: Research, Mass. Institute of Technology, Cambridge, Mass.

Edward Augustus Wright, Rancagua, Chile.

Proposed by Frank Cameron, R. K. Stockwell, Wm. S. Conner.

Born 1890, Roxbury, Mass. Public Grammar Schools, Newton and Boston. 1909-10, Steinhart Hall Prep. School, Boston. Mining Engineering course International Correspondence schools. 1915-16, Univ. of Wis. Extension Division Courses in Structural Engineering and for credit towards degree. 1911-12, Mine Surveyor, Lehigh Valley Coal Co., Wilkes-Barre, Pa. 1912, Asst. Engineer, Peerless Coal & Coke Co., Vivian, W. Va. 1912-13, Straw boss and Powderman, American Smelting and Refining Co., Charcas, S. L. P. 1913, Structural draftsman, I. W. Jones, Milton, N. H. 1913-14, Structural Draftsman, Stone and Webster Engr. Corp., Boston,

Mass. 1914-15, Structural Draftsman, Aberthaw Construction Co., Boston, Mass.
1915-16, Office Engineer and Draftsman, Braden Copper Co. Mines.

Present position: Mine Draftsman and Office Engineer, Braden Copper Co., Rancagua, Chile, S. A.

Thaddeus C. Wilson, Butte, Mont.

Proposed by W. M. Rossberg, G. E. Sheridan, J. O. Proctor.

Born 1889, Sedalia, Mo. 1906, Graduated High School, Springfield, Mo. 1906-10, Drury College, Springfield, Mo. B. S., in Genl. Science. 1910-11, Missouri School of Mines. 1912-13, Missouri School of Mines, B. S. in Mining Engineering. 1911, Butte & Superior Copper Co., Basin, Mont. Zinc concentrator, in Hyde Flotation Test Plant and Mill. 1912-13, Colusa-Parrot Ore Testing Plant, Butte, Mont.—Ore Testing. 1914, Timber Butte Milling Co., Butte, Mont., on Mill Construction. 1914-16, Timber Butte Milling Co., Butte, Mont.—Concentrator-Flotation Foreman.

Present position: Flotation Foreman, Timber Butte Mill.

Ernest Wittenau, Morenci, Ariz.

Proposed by Arthur Crowfoot, John Kiddie, Norman Carmichael.

Born 1883, Cheyenne, Wyo. High School Education in Cheyenne, Wyo., followed by a preparatory course at Andover Academy, Andover, Mass. 1906, Yale University. 1910-14, Testing Dept., B. & M. Reduction Dept., Anaconda Copper Mining Co., Great Falls, Mont. 1914-15, Testing Dept., Anaconda, Mont. 1915, Testing Dept., Arizona Copper Co., Morenci, Ariz.

Present position: Testing Engineer, Ariz. Copper Co.

Thos. W. Wright, Lowell, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, D. M. Rait.

Born 1888, Philadelphia, Pa. General grade school education in public schools of Pennsylvania and Colorado. 1905-08, Golden High School, Colo. 1908-13, Colo. School of Mines, E. M. 1911, U. S. Reclamation Service, Routt Co., Colo. 1912, Eng. Dept., Costello Estates Development Co., San Acana, Colo. 1913, Ohio Copper Mining and Milling Co., Lark, Utah, Concentrator. 1914-15-16, Calumet & Ariz. Mining Co., Mucker, Miner, Timberman, Engineer, First aid and Mine Rescue Instructor.

Present position: Mine Sampler.

S. Yasukawa, Sado Mine, Mitsubishi Goshi Kaisha Sado, Japan.

Proposed by C. E. Arnold, Geo. R. Lehman, W. H. Palmer.

Born 1881, Tokio, Japan. 1908, Graduated from Imperial Kyoto University, Japan. 1908-12, Cyaniding and Smelting at Sado Gold Mine, Sado, Japan. 1912-16, Mine Engineer at above Mine.

Present position: Sub. Manager at mine for above company.

Associate Members

Edward W. Beddow, Warren, Ariz.

Proposed by Philip D. Wilson, Ira B. Joralemon, E. E. Whiteley.

Born 1888, Huntsville, Mo. Huntsville Grammar and High School. 1905, International Correspondence School. 1915, Alexander Hamilton Institute. 1904, Miner, trackman and timberman. 1906, Office Manager, Denny Wholesale Jewelry Co., Kansas City, Mo. 1907, Chief Clerk, W. Div., Wabash R. R., Moberly, Mo. 1908, Hoisting Engineer, Rees Beddow and Co., Gallup, N. M. 1909, Calumet & Arizona Mining Co., Warren, Ariz., Bookkeeper.

Present position: Chief Bookkeeper.

Edmund Douglas Brigham, 226 W. Jackson St., Chicago, Ill.

Proposed by W. G. Swart, C. W. Merrill, H. L. Hollis.

Born 1856, Dunkirk, N. Y. Have had no technical education. Have been since 1873 connected with the Chicago and Northwestern Railway. For the first ten years resided in the Upper Peninsula of Michigan, and there associated with the iron and copper interests. For a considerable period have had charge of the Iron Ore, Copper and Coal traffic of the C. & N. W. Ry. Have been interested financially in mining in Mexico, Colorado, California and Oregon.

Present position: Manager of Freight Traffic, C. & N. W. Ry.

George C. Fraser, New York, N. Y.

Proposed by E. E. Oleott, R. M. Taylor, John H. Hill.

Born 1872, New York City. 1893, B. S., Princeton University. 1895, LL.B. Columbian Univ. (Now Geo. Wash. Univ.), Washington, D. C. 1896, Admitted to Bar. Practised law in Washington, D. C. (until 1901), and New York City.

Present position: Member of the firm of Fraser & Speir, Attorneys and Counsellors at Law.

Charles A. Grisham, St. Louis, Mo.

Proposed by Arthur P. Watt, Chas. E. Schwarz, James A. Caselton.

Born 1883, Missouri. University Preparatory School, Columbia, Mo.

Present position: Asst. Treasurer of St. Louis Refining & Smelting Co.

Roscoe Teats, 2701 So. 13th Street, Tacoma, Washington.

Proposed by P. A. Mosman, Willard V. Morse, Brent N. Rickard.

Born 1880, Lawrence, Kansas. 1904, Bachelor of Science in Metallurgy, University of Washington, Seattle, Washington. 1904-05, Assayer, Bunker Hill Mining & Smelting Co., at Index, Washington. 1905-16, Chemist, Tacoma Smelting Co., Tacoma, Washington. 1916, Assistant Superintendent, Tacoma Smelting Co., Tacoma, Wash.

Present position: Assistant Superintendent, Tacoma Smelting Co.

Junior Members

Clifton Thompson Armstrong, 347 Manhattan Ave., New York, N. Y.

Proposed by Robert Peele, R. M. Raymond, Charles P. Berkey.

Born 1895, New York City. 1901-09, P. S. No. 27, Queens. 1909-13, Flushing H. S. at Flushing, L. I. 1913, Columbia University. 1916, Worked as underground laborer at Champion Mine, Painesdale, Michigan.

Present position: Student of Columbia University.

Phillips Brooks Dolman, Rolla, Mo.

Proposed by Ben H. Cody, Karl G. Hamm, Chas. Clayton.

Born 1893, Topeka, Kans. 1906, Graduated from Grammar School, Spokane, Wash.; High School, St. Joseph, Mo.; Tutor for College Prep. 1915-16, Employed as Chemist, Experiment Station, Mo. School of Mines.

Present position: Student.

Arturo C. Fernandez, Box No. 219, Rolla, Mo.

Proposed by Chas. Y. Clayton, Horace T. Mann, H. A. Buehler.

Born 1892, Monterrey, Mexico. Prior to 1911, attended the leading schools and colleges in the following cities: Monterrey, Saltillo and Mexico City. 1911-14, Missouri School of Mines. 1914-16, Employed by the Compania de Minerales y Metales, S. A. 1914-15, Assistant to chief assayer and chemist of the calcination plant at Villaldama, Mexico. 1915, Surveyor of "Minas Viejas" mines near Villaldama, N. L. 1915-16, In charge of the surveying department and laboratory of "Fraternal" mine near Lampazos, N. L.

Present position: Student Missouri School of Mines.

Stuart Grayson Garrett, University, Va.

Proposed by J. S. Grasty, J. H. Batcheller, Thomas L. Watson.

Born 1892, Seattle, Wash. 1912-14, General Scientific Course, Univ. of Virginia, B. Sc. 1915-16, Graduate student in geology, Univ. of Virginia, M. S. 1916 (Summer), Asst. Supt., Valzinco Mine, Virginia Lead and Zinc Corp'n.

Present position: Graduate Student in geology and Instructor in geology, Univ. of Virginia.

Donald C. Gilbert, Columbus, O.

Proposed by H. E. Nold, S. B. Belden, H. W. Linhardt.

Born 1894, Kendalville, Ind. 1913, Grad. High School, Ft. Wayne, Ind.

Present position—1913 to date: Student, School of Mines, Ohio State University.

Tony Frank Golick, Rolla, Mo.

Proposed by H. T. Mann, C. R. Forbes, H. A. Buehler.

Born 1896, Streator, Ill. 1910-14, Canton High School. 1914-16, Missouri School of Mines. 1909, Chas. Golick & Co., Wisconsin Steel Co.

Present Position: Student.

Percy W. Hatfield, Washington, Pa.

Proposed by H. B. Meller, S. L. Goodale, Robert M. Black.

Born 1890, Washington, Pa. 1905-09, Washington High School. 1916, University of Pittsburgh, Pa. 1910, Jessop Steel Co., Washington, Pa.

Present position: Student at U. of P., Clerk at Jessop Steel Co.

Floyd Dixie James, Rolla, Mo.

Proposed by Charles Y. Clayton, Horace T. Mann, H. A. Buehler.

Born 1893, St. Louis, Mo. 1912, Central High School, St. Louis, Mo. 1917, Missouri School of Mines, B. S. in Met. 1913-15, Chemist, Deering Plant, American Zinc Co. 1915, Representative for Butte and Superior Copper Co. 1915-16, Night Foreman, plant of Bartlesville Zinc Co., Collinsville, Okla.

Present position: Experiment Station, working on electrolytic zinc and finishing school work.

Eitar Arthur Miller, Rolla, Mo.

Proposed by Horace T. Mann, Chas. Y. Clayton, Austin L. McRae.

Born 1892, Carbon, Ill. 1906, Ill. Public Schools. 1912, Missouri School of Mines. 1906-10, Mad. Coal Corp., Glen Carbon, Ill., underground work. 1910-12, Mad. Coal Corp., Cartersville, Ill., surface. 1913-14 (Summer), Miami Copper Co., Miami, Ariz. 1915-16 (Summer), O. D. Copper Co., Globe, Ariz.

Present position: Student.

Walter Arnold Rukeyser, Columbia Univ., New York, N. Y.

Proposed by Chas. P. Berkey, Robt. Peele, E. J. Hall.

Born 1895, New York City. 1912-13, School of Mines, Columbia Univ. 1913-16, Princeton University, B. Sc. in Geology. 1915, Summer Session, Schools of Engineering, Columbia Univ. 1916, Summer Session, School of Mines, Columbia Univ. 1914, Worked during summer at the concentrator of the Nevada Consolidated Copper Co., McGill, Nevada. 1914, Visits to Utah Copper Co., Bingham and Garfield, Utah, Leadville and Cripple Creek, Colo. 1916, 4 weeks visiting Dome Mines Hollinger and Vipond Mines at Porcupine, Ore., Penn-Canadian Mines, Townsite Mill, Buffalo Mill, Cobalt, Ont., 1916, Visiting the Tash Orn Mines Co., Ltd.

Present position: Student.

Bert Rice Smith, 1053 E. 13th Street, Brooklyn, N. Y.

Proposed by Robert Peele, R. M. Raymond, Homer L. Carr.

Born 1893, New York City. 1911-13, Worcester Academy.

Present position: Student, Columbia University.

Haifan F. Wang, Columbia University, New York, N. Y.

Proposed by R. M. Raymond, E. J. Hall, Robt. Peele.

Born 1894, Ningpo, China. 1911-15, Pei Yang University, Tientsin, China.

Present position: Student at above University.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Nov. 10, 1916 to Dec. 10, 1916.

This list together with the list published in *Bulletin* Nos. 110 to 120, February to December, 1916, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Dec. 10, 1916.

ADAMS, JOHN H.	512 8th Avenue West, Birmingham, Ala.
AGNEW, JOHN A.	"Royston," West Heath Ave., Hampstead, London, N. W., England.
AHLERS, RUDOLPH O.	Rua do Corpo Santo 13, Lisbon, Portugal.
AMSDEN, O. B.	Kingman, Ariz.
ANDERSON, JOHN C.	Wilhelm Mine, Secret Pass, Ariz., P. O. Box 711, Kingman, Ariz.
ANDERSON, ROBERT J.	10839 Pasadena Ave., N. E., Cleveland, Ohio
BILLICK, DON CARLOS	Thane, Alaska.
BOERICKE, HAROLD	8 National State Bank Bldg., Boulder, Colo.
BOLGIANO, J. RALPH	The Emerson Co., 30 Church St., New York, N. Y.
BORDEAUX, ALBERT F. J.	Fribourg, Switzerland.
BROWN, E. O. FORSTER	Cons. Min. Engr., 706-07 Salisbury House, Finsbury Circus, London, E. C., England.

- BUNKER, CHARLES E. Amador City, Cal.
 CHU, HSING CHUNG, Min. Engr., Mining Dept., Chili Provincial Financial Bureau,
 Central Station, Tientsin, China.
 COGGESHALL, GEORGE W. 2229 California St., Washington, D. C.
 COOK, PAUL R., Care American Institute of Mining Engineers,
 29 W. 39th Street, New York, N. Y.
 CORNELL, ROY LE GRAND, Resident Engr., Telluride Chief Mining Co., Kingman, Ariz.
 CRERAR, GEORGE Care France Inv. Co., Union Oil Bldg., Los Angeles, Cal.
 CREWE, LEONARD C. La Follette, Campbell Co., Tenn.
 DAVENPORT, JOHN 1 Menlo Street, Brighton, Mass.
 DE HORA, MANUEL H. Instructed to hold all mail.
 DIETRICH, WALDEMAR F. 1225 Byron St., Palo Alto, Cal.
 DONDERO, F. NICHOLAS Olinghouse, Nev.
 DUNSFORD, ENSA R. Upham, Kings Co., New Brunswick, Canada.
 DUREHAM, EDWARD B., Min. Engr. Mount Kisco, N. Y.
 DUTTON, CHARLES B., Care Darwin S. Wolcott, Farmers Bank Bldg., Pittsburgh, Pa.
 DWYER, C. EUSTACE, American Smelting & Refining Co., 618 McIntyre Bldg.,
 Salt Lake City, Utah.
 EAMES, LUTHER B. 3009 High St., Pueblo, Colo.
 ELY, FRED B. 108 Bank of Ottawa Bldg., Vancouver, B. C., Canada.
 EMMONS, ARTHUR B. Redwood Library, Newport, R. I.
 EMMONS, CAROLUS D. 524 N. E. 12th St., Portland, Ore.
 EVANS, DAVID J. Bay Cities Garage, Santa Monica, Cal.
 GALLAGHER, FRANCIS J., Mine Engr. Arizona-Comstock Mining Co.,
 Oatman, Ariz.
 GIBBONS, CHARLES A., JR., Andes Exploration Co., Arequipa, Peru, South America.
 GORDY, SHEPPARD B. Care Fritz Mella, Santiago, Chile, South America.
 GORMAN, THOMAS C., Lieut., Canadian Expeditionary Force, Army Post Office,
 London, England.
 GRIFFITHS, HARRY D. 4 London Wall Bldgs., London, E. C., England.
 GRUBNAU, V. CARL Care Hotel Alvarado, Albuquerque, New Mexico.
 GRUGAN, JUSTICE Room 621, 30 Church St., New York, N. Y.
 GUNTHIER, CHARLES B. 309 W. Florence St., Blackwell, Okla.
 HAMILTON, L. A. 29 Broadway, New York, N. Y.
 HARMS, ERNEST 902 Upson Ave., El Paso, Texas.
 HASBROUCK, BERNARD, Petroleum Geol. & Engr., The Carter Oil Co.,
 Central National Bank Bldg., Tulsa, Okla.
 HECKSCHER, A. 50 East 42d St., New York, N. Y.
 HEDGES, JOSEPH H. 653 E. Broadway, Salt Lake City, Utah.
 HOLDERER, GEORGE B. Northern Pyrites Co., Northpines, Ont., Canada.
 HOVLAND, HARRY B. Los Angeles Athletic Club, Los Angeles, Cal.
 HUMMELL, ARCHIBALD S., Met. Wm. Wharton, Jr. & Co., Inc., Easton, Pa.
 INNES, MURRAY 217 Kohl Bldg., San Francisco, Cal.
 JARMAN, ARTHUR Barry Road, Waihi, Auckland, New Zealand.
 JENKS, ARTHUR W., Burma Mines, Ltd., Namtu, Northern Shan States,
 Burma, India.
 JOHNSON, CLIFTON B. 122 W. 72d St., New York, N. Y.
 KAANTA, HENRY W. Bingham-New Haven C. & G. M. Co., Bingham, Utah.
 KEENEY, ROBERT M., Mgr., The Ferro Alloy Co., 603 Symes Bldg., Denver, Colo.
 KLEENIKOFF, KENNETH J. Chloride, Ariz.
 KLINE, HARRY D. Ray Consolidated Copper Co., Ray, Ariz.
 LAWESHE, VERNER T. Mantua Chemical Co., Paulsboro, N. J.
 LELAND, EVERARD Shannon Copper Co., Florence, Ariz.
 LIBBEY, FAY W. P. O. Box 1148, Phoenix, Ariz.
 LIST, ELMER Standard Chemical Co., Cannonsburg, Pa.
 LONGYEAR, JOHN M., JR. P. O. Box 4574, Mt. Dora, Fla.
 LOVEJOY, JOHN M. Producers Oil Co., Wichita Falls, Tex.
 LUNA, GILBERTO, Min. Engr., Res. Mgr., "Angustias, Dolores y Anexas" S. A.,
 Apartado 115, Mineral de Pozos, Gto., Mexico.
 MCCARTY, EDWARD P. 3615 Lyndale Ave., So., Minneapolis, Minn.
 MCINTOSH, FRANK K. Cia. de Minerales y Metales, S. A., Laredo, Tex.
 MALLER, JOHN W. 42 N. Second St., Easton, Pa.
 MATHEWSON, EDWARD P. Royal Bank Bldg., 8 King St. East, Toronto, Canada.
 MAXON, WALTER L., Amalgamated Zinc (De Bavay's) Collins House, 360 Collins St.,
 Melbourne, Vic., Australia.

MAZANT, MARK S.	Chile Exploration Co., 120 Broadway, New York, N. Y.
MILES, JOHN B.	364 South 16th St., Philadelphia, Pa.
MILES, JOHN H., Genl. Supt.	Alaska Mines Corp., Nome, Alaska.
MORRIS, HENRY C.	1868 Columbia Road, Washington, D. C.
NASH, WILLARD H., Engr.	28 German Insurance Bldg., Buffalo, N. Y.
NEAL, WALTER	1553 Harvard Ave., Salt Lake City, Utah.
NEELD, HARPER C.	Cascade Inn, Shawmegan Falls, Quebec, Canada.
NEWCOMB, CLIVE S., Engr.	The Dorr Co., 17 Battery Pl., New York, N. Y.
NICHOLSON, FRANCIS	Charlotte Court House, Va.
O'NEILL, FREDERICK W.	1226 Denmark Road, Plainfield, N. J.
PALLANSCH, ROLLIN A.	U. S. Smelting Co., Midvale, Utah.
PARRISH, SAMUEL F.	5017 Cimarron St., Los Angeles, Cal.
PATERSON, ARTHUR W., Mine Mgr.	1111 W. 12th Ave., Spokane, Wash.
PETERSON, FRANK	Mine Proprietor and Operator, Dos Cabezas, Ariz.
PETTIS, EDSON S.	Mill Valley, Cal.
PLATT, EDWIN H.	208 Colorado National Bank Bldg., Denver, Colo.
POWELL, GUY M., Supt. Coal Mines	Central Iron & Coal Co., Kellermann, Ala.
RAIBER, NICOLAUS H.	127 Madison St., Wilkes-Barre, Pa.
RANSOM, RASTUS S., JR., Min. Engr.	James Ore Concentrator Co., 35 Runyon St., Newark, N. J.
RATHBUN, FRANK DE G.	Detroit Copper Mining Co., Morenci, Ariz.
RICKARD, BRENT N.	American Smelting & Refining Co., East Helena, Mont.
ROBERTS, WILLIAM P.	Wynnewood, Pa.
ROBESON, JACOB H.	303 Ideal Bldg., Denver, Colo.
ROGER, EUGENE, care of Mr. Hudson, 9 Battenberg Ave.	Knighon, Leicester, Eng.
ROGERS, BENJAMIN C.	P. O. Box 233, Mullin, Texas.
ROGERS, CHARLES E.	Fraser & Chalmers, Ltd., Erith, Kent, England.
ROSENBAUM, RUDOLPH R.	1301 Monadnock Bldg., Chicago, Ill.
ROWAND, LEWIS G.	New Jersey Zinc Co., 55 Wall St., New York, N. Y.
SALAZAR, S. L.	Apartado 34, El Oro, Mex., Mexico.
SCHUTLER, WALTER S.	2554 Haste St., Berkeley, Cal.
SCOLLES, JOHN C.	Ironton, Minn.
SEARING, OLIVER P.	P. O. Box 1023, Jacksonville, Fla.
SMITH, HENRY BOYNTON	315 Vandalia Ave., Collinsville, Ill.
SPEER, J. DANA	P. O. Box 435, Jerome, Ariz.
SPIESBURY, PERSIPOR G.	The Aguacate Mines, 55 Liberty St., New York.
STACK, FRANK L.	Instructed to hold all mail.
STANFORD, RICHARD B.	Bonanza Mine, Bluefields, Nicaragua, C. A.
STEEL, DONALD	411 Kipling St., Palo Alto, Cal.
STEWART, MELVILLE B., Min. Engr.	Lumoghi Coal Co., Collinsville, Ill.
STOCKETT, NORMAN A.	Engrg. Dept., St. Joseph Lead Co., Bonne Terre, Mo.
STROHECKER, JOHN W.	Keystone, Mont.
STROUD, BENJAMIN K.	Overall, McCray Ltd., Balmain, Sydney, N. S. W., Aust.
SUTTON, HENRY A.	Ocotillo, Ariz.
SWEETSER, RALPH H.	Met. in Blast Furnace Practice, 106 Wayne Ave., Easton, Pa.
TATE, E. L.	416 Hyde Blk., Spokane, Wash.
TAYLOR, CHARLES H.	315 W. 17th St., Oklahoma City, Okla.
THILL, J. R., Genl. Master Mech., Construction Dept., Braden Copper Co.	Rancagua, Chile, So. America.
THOMAS, DAVID R., Min. Engr., Mgr., Davidson Mines, Ltd., South Porcupine, Ont., Canada.	
TRAUERMAN, CARL E.	832 Colorado St., Butte, Mont.
WEINBERG, SEMEN G.	404 Riverside Drive, New York, N. Y.
WALTER, E. W.	Min. Engr. & Met., Silverton, Colo.
WARFORD, NORMAN L.	2906 Carroll Ave., Chicago, Ill.
WEINBERG, E. A., Min. & Met. Engr.	45 Broadway, New York, N. Y.
WILMOT, H. CLIFFORD	Pleasantville, N. Y.
WILSON, ARDEN M.	Genl. Mgr., The Radium Ores Co., Naturita, Colo.
WITHERELL, CHARLES S., Met. Engr.	Chile Exploration Co., 120 Broadway, New York, N. Y.

MEMBERS' ADDRESSES WANTED

Name.	Last Address of Record from which Mail has been returned.
BAUMANN, A. P.	939 Boulevard East, Weehawken, N. J.
BOUERY, PIERRE	Lagrange Mining Co., Weaverville, Cal.
COMINGS, GEORGE R.	Apartado 126, Oaxaca, Oax., Mexico.
McKANNA, EDWIN A.	Savanna, Okla.
MACAULAY, R. M.	Grande Allee St., Quebec City, Canada.
MURPHY, FRANCIS JOSEPH	Great Cobar, Ltd., Cobar, N. S. W., Australia.
PITTMAN, FRANK L.	Contact, Nev.
ROGERS, GEORGE R.	47 St. Vincent St., Toronto, Ont., Canada.
VAN RENSSELAER, ARTHUR M.	119 East 51st Street, New York, N. Y.
WELLS, RALPH EVANS, JR.	Nogales, Ariz.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Nov. 10, 1916, to Dec. 10, 1916.

Date of Election.	Name.	Date of Death.
1899	**Dennis, Francis James	Nov. 18, 1916.
1912	*Merrill, Frederick J. H.	Nov. 29, 1916.
1887	*Wyatt, Francis	Feb. 27, 1916.
	*Member.	**Life Member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*. PERCY E. BARBOUR, *Vice-Chairman*.
 A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
 C. A. BOHN, *Treasurer*.

JOHN V. N. DORR,

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

W. E. C. EUSTIS, *Chairman*. R. L. AGASSIZ, *Vice-Chairman*.
 E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
 ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.

W. H. LINNEY, *Chairman*. OSCAR LACHMUND, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
 STANLY A. EASTON, S. SHEDD.

Puget Sound

Meets second Saturday of each month.

GLENVILLE A. COLLINS, *Chairman*. H. L. MANLEY, *Vice-Chairman*.
 AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
 I. F. LAUCKS, JOHN N. POTT.

Southern California

C. COLCOCK JONES, *Chairman*. ALVIN B. CARPENTER, *Vice-Chairman*.
 A. B. W. HODGES, *Secretary-Treasurer*, R. A. PEREZ.
 E. A. MONTGOMERY, WILLIAM F. STAUNTON.

Colorado

L. P. HAMMOND, *Chairman*. F. H. BOSTWICK, *Vice-Chairman*.
 P. M. McHUGH, *Secretary-Treasurer*, 812 Cooper Bldg., Denver, Colo.
 G. A. KENNEDY, M. S. MacCARTHY.

Montana

J. L. BRUCE, *Chairman*. W. C. SIDERFIN, *Vice-Chairman*.
 WALTER E. GABY, *Secretary-Treasurer*, 319 North Jackson St., Butte, Mont.
 W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.

T. A. RICKARD, *Chairman*. W. H. SHOCKLEY, *Vice-Chairman*.
 C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
 E. A. HERSAM, H. W. YOUNG.

Pennsylvania Anthracite

R. V. NORRIS, *Chairman*. EDWIN LUDLOW, *Vice-Chairman*.
 CHARLES F. HUBER, *Vice-Chairman*. ARTHUR H. STORRS, *Vice-Chairman*.
 W. J. RICHARDS, *Vice-Chairman*. PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI, *Chairman*. HERMAN GARLICH, *Vice-Chairman*.
 F. W. DeWOLF, *Vice-Chairman*. M. M. VALERIUS, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
 C. R. FORBES, F. D. RASH.

Chicago

CHARLES H. MACDOWELL, *Chairman*. LUTHER V. RICE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
 ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
 GEORGE P. HULST, FREDERICK T. SNYDER.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 29th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Potash as a Byproduct from the Blast Furnace

BY R. J. WYSOR,* SO. BETHLEHEM, PA.

(New York Meeting, February, 1917)

SINCE the outbreak of the European war, few problems of raw-material supply have commanded more nation-wide attention than potash. It is well known that before the war the domestic production of potash was an insignificant percentage of the imports. The average annual importation of raw potash salts for several years prior to 1914 was slightly over 300,000 net tons, and of other potash manure salts about 700,000 tons. The sudden and almost total cessation of these imports created a unique and stringent situation. Methods of recovering potash from feldspar and other native mineral sources, and reclamation from hitherto waste products, have received a marked impetus. Soaring prices have been a keen incentive to research and industrial development.

Although several brief articles concerning the possibilities of potash recovery have appeared in recent trade journals, it still may be a matter of some surprise to the average technical mind that potash might be reclaimed as a profitable byproduct in the manufacture of pig iron. At the present time it may be of special interest to present some data, largely of a technical nature, on this subject.

A fairly thorough search of the literature reveals a number of articles concerning salts of the alkali metals in the blast furnace. Many of them deal with the theoretical rôle of alkali cyanides in the furnace. Several discuss the probable effect of the alkalis on the furnace brickwork. Two or three ambitious patents have been granted for reclaiming potash or other products from blast-furnace gas. However, we have heard nothing further as to the practical application of these patents. Little literature of importance has appeared during the last 10 years concerning alkalis in blast-furnace practice, and I have discovered no record whatsoever as to the actual sale or commercial disposal of flue dust for its potash content until this was inaugurated in our plant at Bethlehem. A bibliography of the most important articles discovered is appended.

About 4 years ago, in the course of investigating blast-furnace stove efficiencies, I analyzed the fine, yellowish fume of which a considerable

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quantity was removed from the bottom of the stove checkerwork. The sample was found to contain, among other constituents, about 15 per cent. water-soluble potash, which was somewhat surprising. This induced an investigation, which showed that considerable quantities of this material, hitherto a waste product, could be recovered from our stoves and gas-fired boiler settings. A search for a market was made without immediately satisfactory results. One fertilizer dealer claimed that the alumina content was too high. Others wished to make practical tests, and we furnished large samples for a full season's demonstration, satisfactory results being reported. But the pre-war-time prices offered hardly seemed to justify the trouble of reclaiming the dust.

With the beginning of the war and the subsequent spectacular rise in the potash market, conditions were changed. Knowing just what dust to recover, reclamation was immediately commenced, a satisfactory contract was negotiated, and this company has been disposing of the dust at a good profit ever since.

Having introduced the subject thus far in the same order in which it was initially investigated, we may now turn logically to an inquiry as to the source of potash in the raw materials charged, behavior in the furnace, and methods for recovering it as a byproduct, or of eliminating it as a nuisance. As a matter of both scientific and practical interest, a consideration of sodium will also be attempted, though not so thoroughly as its more important kindred metal, potassium.

Occurrence of Alkalies in Raw Materials

Potash doubtless occurs almost entirely in some form of feldspar or clay in all the materials entering the blast furnace.

At Bethlehem we receive ores from many quarters of the globe, and an attempt was made to discover whether the relative percentages of alkalies in the various ores bear any special relation to the source, mineralogical nature, or to the silica and alumina contents of these ores. No striking relation was discovered, except that the manganese ores, from widely separated sources, were found to contain relatively high percentages of potash as compared to most iron ores. Iron ores of this country, including those of the South, containing upward of 1 per cent. potash, or over, seem to be restricted to small areas.

The percentage of potash in different varieties of limestone and dolomite used as flux, varies greatly, largely on account of intermixed clay, and may be surprisingly high. Per unit weight, the potash content of our flux charge at Bethlehem is considerably higher than in either the average ore or coke charge.

Few data are available on the alkali content of coke from different localities. Various standard Connellsville cokes which we have examined

have shown a much lower alkali content than our local byproduct coke made from West Virginia coals.

It may be a matter of some surprise to note that, with one exception, the soda contents of all the iron ores, the coke and the flux, subsequently listed, are much higher than the potash contents.

Before presenting any analyses, it may be mentioned that the difficult analytical determinations of potash and soda, and the complete analyses of the various miscellaneous samples, were executed almost entirely by two expert analysts, who literally spent several weeks in this work. Most of the determinations were duplicated to insure their accuracy.

Table 1 shows analyses representing 3 months' shipments in 1916, including hundreds or thousands of car loads of each variety, of the chief materials charged into our blast furnaces. Besides the alkali metals, percentages of silica and alumina are given, also iron contents of the iron ores, all analyses being expressed on the "natural" basis.

TABLE 1.—*Analyses of Materials of Blast-furnace Charge*

Materials	Kind	Source	Fe	SiO ₂	Al ₂ O ₃	K ₂ O	Na ₂ O
<i>Iron Ores:</i>							
Tofo.....	Magnetite	Chile	65.0	2.7	0.9	0.21	0.48
Juragua.....	Magnetite	S. Coast Cuba	54.5	13.0	2.7	0.29	0.77
Mayari.....	Limonite	N. Coast Cuba	38.0	2.0	10.0	0.06	0.25
Swedish.....	Magnetite	Northern Sweden	66.0	2.8	1.1	0.27	0.82
Port Henry Conc.....	Magnetite	Northern New York	63.5	4.4	1.8	0.28	0.68
Harmony Cobbed.....	Magnetite	Northern New York	63.0	10.0	3.4	0.25	0.55
Barton Hill.....	Magnetite	Northern New York	64.0	8.3	1.9	0.22	0.77
Cheever.....	Magnetite	Northern New York	59.5	11.4	1.0	0.21	0.92
Sterling.....	Magnetite	Southern New York	55.0	8.6	2.0	0.33	0.95
Manganate.....	Hematite	Lake Menominee	45.5	7.2	3.3	0.24	0.47
Norman.....	Hematite	Lake Gogebic	55.5	5.4	1.5	0.29	0.32
Eureka.....	Hematite	Lake Gogebic	54.5	7.1	1.5	0.26	0.35
Mary.....	Hematite	Lake Marquette	50.5	7.8	2.5	0.28	0.55
<i>Manganese Ores:</i>							
Brazilian.....		Brazil		7.5	3.0	0.88	0.21
Cuban.....		S. Coast Cuba				1.12	0.77
Oriental.....		India				1.68	0.19
<i>Fuel:</i>							
Lehigh byproduct coke.....		West Virginia Coals		5.1	3.4	0.28	0.30
<i>Flux:</i>							
McAfee limestone.....		Northern New Jersey		2.5	0.9	0.36	0.70
Average dolomite.....		Bethlehem Vicinity		3.5	1.5	0.82	0.97

Two or three of the above ores are mixtures of magnetite and hematite, with the former predominating.

In the course of this investigation, samples of slag and flue dust from all of our Bethlehem furnaces were collected continuously over a period of 4 weeks during June and July, 1916, and analyzed for the alkalies. The

analyses in Table 1 of ores, fuel and flux, are fairly representative of the materials charged during this period. The same coke and stone (half limestone and half dolomite) are used in all the furnaces. The percentages of potash and soda in the ore mixture charged in each furnace during the above period are as follows:

Furnace	A	C	D	E	F	G
K ₂ O in ore mixture, per cent.	0.28	0.27	0.28	0.29	0.28	0.28
Na ₂ O in ore mixture, per cent.	0.63	0.43	0.65	0.50	0.51	0.63

From the foregoing data, and from known weights of materials charged, Table 2 has been constructed, showing the weights of potash and soda charged per ton of pig iron produced.

TABLE 2

	Potash (K ₂ O)			Soda (Na ₂ O)		
	Gross Ton	Pounds	Per Cent. of Total Contributed	Gross Ton	Pounds	Per Cent. of Total Contributed
Average weight in ore mixture, all furnaces, per ton pig	0.0046	10.3	46	0.0092	20.6	58
Average weight in coke all furnaces, per ton pig	0.0026	5.8	26	0.0028	6.3	18
Average weight in stone, all furnaces, per ton pig	0.0028	6.3	28	0.0039	8.7	24
Total.....	0.0100	22.4	100	0.0159	35.6	100
Ratio potash to soda charged.....	1:1.6					

The ratio of potash to soda is considerably higher in the coke than in the ore and stone. From Table 2 it will be seen that for each ton of pig iron produced, nearly 60 lb. of the alkali oxides are charged, constituents which certainly are of considerable importance in the working of the furnace, yet of which relatively little cognizance has been taken in the past.

However, due largely to the relatively high percentage of alkalis in our fuel and flux, and on account of the considerable quantity of alkali-bearing flue dust removed, it is my opinion that more potash and soda are charged into our blast furnaces, per unit of iron produced, than in any other large plant in this country.

Action of Alkalies Within the Furnace

It is probable that a considerable part of the potash and soda charged into a blast furnace is evolved from the top by direct volatilization or

heat decomposition, though alteration by chemical reaction of the alkaline salts or compounds liberated may occur before they have left the furnace. It is certain that a large part of the alkalies is carried down into the hotter zones of the furnace and converted into cyanides by reaction with red-hot carbon. Some investigators have attributed an appreciable part of their ore reduction to the action of cyanides, inferring that after being oxidized and driven to the cooler upper portion of the furnace, they condense and are again carried down into the reducing zone. Whatever the action, it is self-evident that eventually the same amount of alkalies must be carried out of a furnace that is charged, and this takes place through the following avenues of escape:

1. *In Chemical Combination in the Slag.*—The literature is almost barren with reference to the presence of alkalies in blast-furnace slag. On account of the readiness with which the compounds of these metals are sublimed and carried out of the furnace with the gas, it might be inferred that only a negligible proportion would be found in the slag. This is far from the truth. Average samples from each of six furnaces in operation for two bi-weekly periods were carefully analyzed for potash and soda, with the results shown in Table 3. The slags were all normal for our practice, averaging about 35 per cent. silica, 13 per cent. alumina, 13 per cent. magnesia and the remainder lime, etc. In order to give an idea as to the hearth temperatures, the average silicon contents of the pig iron produced over the same periods are also tabulated.

TABLE 3

Furnace	Period Included, 1916	Per Cent. Alkalies in Slag		Per Cent. Silicon in Pig Iron
		K ₂ O	Na ₂ O	
A	$\frac{9}{25}$ to $\frac{7}{8}$	0.48	0.96	1.07
A	$\frac{7}{8}$ to $\frac{7}{22}$	0.38	0.70	1.06
C	$\frac{9}{25}$ to $\frac{7}{8}$	0.17	0.22	1.62
C	$\frac{7}{8}$ to $\frac{7}{22}$	0.18	0.22	1.70
D	$\frac{9}{25}$ to $\frac{7}{8}$	0.42	0.36	1.22
D	$\frac{7}{8}$ to $\frac{7}{22}$	0.40	0.40	1.20
E	$\frac{9}{25}$ to $\frac{7}{8}$	0.56	0.60	1.50
E	$\frac{7}{8}$ to $\frac{7}{22}$	0.65	0.66	1.35
F	$\frac{9}{25}$ to $\frac{7}{8}$	0.29	0.36	1.27
F	$\frac{7}{8}$ to $\frac{7}{22}$	0.40	0.64	1.27
G	$\frac{9}{25}$ to $\frac{7}{8}$	0.34	0.68	1.27
G	$\frac{7}{8}$ to $\frac{7}{22}$	0.35	0.72	1.05
Average for all furnaces, first period		0.38	0.53	1.32
Average for all furnaces, second period.....		0.39	0.56	1.29
Average ratio potash to soda in slag.....				1:1.4

The calculated weight of slag produced during this period was about 0.52 ton per ton of pig iron produced, which would account for a loss of alkalis as follows:

	Pounds	Per Cent. of Total Charged
Average potash in slag per ton pig.....	4.5	20
Average soda in slag per ton pig.....	6.3	17

The percentage loss of the two alkalis in the slag is seen to be about the same, being somewhat greater in the case of potash.

In the case of "C" furnace, almost the same weight of alkalis was charged and the same weight of slag was produced per ton of iron, as at the other furnaces, yet it will be noted that the percentages of alkalis in the slag is much lower. This probably may be attributed to the higher hearth temperatures, as indicated by the higher silicon iron produced. However, I am unable to generalize this theory because further data are lacking at present.

On old blast-furnace slag dumps, deposits of a yellowish or almost white substance frequently may be found in sheltered crevices or small grottoes, into which water has percolated through overlying masses of slag, thus becoming more or less charged with soluble salts. The substance left by evaporation may be in the form of stalactites, thin acicular crystals, or of a dry powder. Considerable quantities of this material may be seen at Bethlehem where portions of our slag dump are being removed for concreting and other work. Two representative samples, one of light-yellow material, and the other nearly pure white, were obtained and, after being dried, were found to have the following proximate composition:

Sample	SiO ₂	Fe ₂ O ₃ + Al ₂ O ₃	MnO	CaO	MgO	K ₂ O	Na ₂ O	Total Sulphur	Sulphide Sulphur	Cl
White.....	0.20	0.36	0.04	26.47	None	8.66	4.27	38.3	14.5	trace
Yellow.....	0.38	0.13	trace	17.60	None	17.64	8.76	37.6	25.6	trace

The above analyses are interesting and instructive. We note the presence of nearly twice as much potash as soda in these samples, whereas in the raw slag the figures are almost reversed. The simultaneous presence of large amounts of potash, soda and sulphur indicate that, to a limited extent, the alkalis are efficient desulphurizers in the blast furnace.

Although not directly connected with the subject of this paper, I wish to call attention in passing to the apparent significance of the relatively

large percentage of lime and sulphur, and the absence of magnesia in the above leachings. We know that practically without exception magnesium salts are more soluble in water than the corresponding salts of calcium. Our flux at Bethlehem for years has consisted of half limestone and half dolomite; as previously stated, the magnesia content of the slags averages about 13 per cent. The above facts furnish new evidence, so far as can be ascertained, as to the superiority of lime over magnesia as a desulphurizing agent in the blast furnace. Further corroborative tests were made by subjecting fresh, powdered samples of slag to 24-hr.



FIG. 1.—WHITE CRYSTALS FROM COOLING PLATES ABOVE MANTEL.

leaching tests in hot and cold water. In every case, appreciable amounts of sulphur (in both sulphide and sulphate state) and lime, were found in about equivalent ratio for combination, and never more than a trace of magnesia.

2. *As Cyanide or Other Volatile or Inflammable Compound through the Iron and Cinder Notches.*—Part of the fume arising from the molten iron, and especially from the slag running from the furnace, is undoubtedly alkali compounds. I have often noticed a peculiar lavender or violet flame around the iron and cinder notches during casting or flushing, which I believe is due to some alkali salt or salts burning. The fume

arising from the iron and cinder runners certainly contains a considerable percentage of alkalies.

3. *By Liquid Exudation or Deposition from Gas Around the Tuyères, Coolers, Mantel and Cooling Plates.*—Occasionally while removing a tuyère or cooler, a stream of liquid "cyanide," resembling water, will run out of the furnace, or can be seen exuding from the lining.



FIG. 2.—YELLOWISH AND YELLOWISH-RED CRYSTALS FROM COOLING PLATES ABOVE MANTEL.

When cooling plates above the mantel burn out, due to the failure of the water supply, and a little gassing has commenced, beautiful white and yellow or yellowish-red crystals often build up around the apertures. In mass, all of these crystals appear to be strongly deliquescent, and hence it is difficult to obtain photographs showing clearly defined crystal faces, though the crystalline nature of the material is clearly apparent from Figs. 1 and 2. When examined on the hot furnace shell with a good hand glass, there is seen sometimes a mixture of different

kinds of crystals and in different systems. Several samples of brilliant yellow crystals were secured and a few reddish-yellow which are very likely potassium ferrocyanide. Usually, however, the crystals are pure white, and have been found to be chiefly ammonium chloride. An analysis was made of some of these white crystals, slightly contaminated, taken from two different furnaces at different times, the results being: SiO_2 , 0.78 per cent.; $\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$, 2.49; CaO , 0.29; MgO , 0.28; K_2O , 0.24; Na_2O , 0.12; Cl , 62.39; CO_2 , 0.72; CN , none; NH_4 , 30.69 per cent.

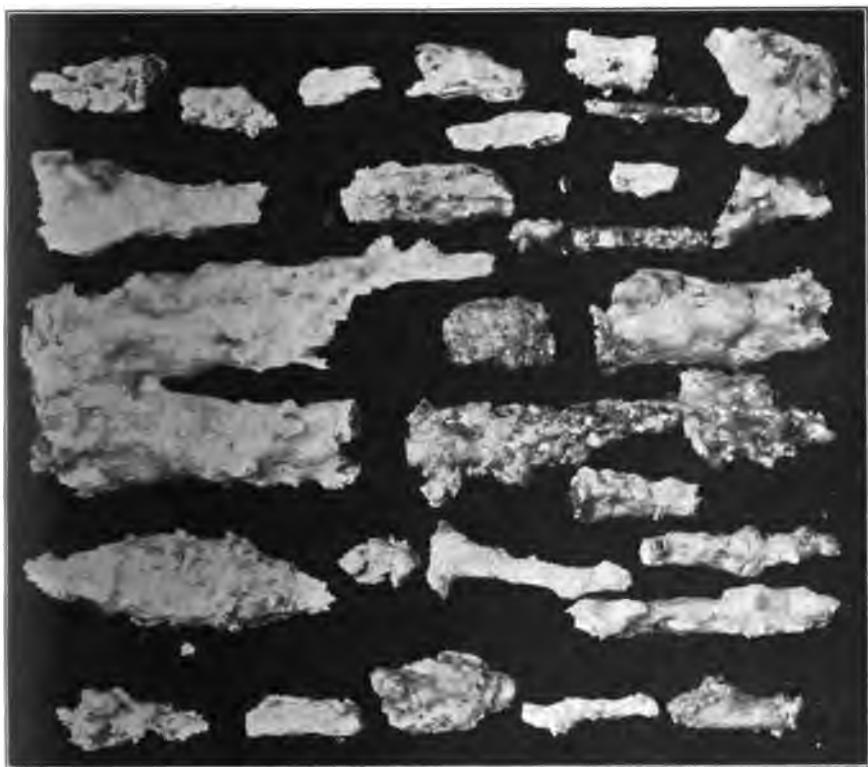


FIG. 3.—WHITE "CYANIDE" STALACTITES FROM SHELL ABOVE MANTEL.

Liquid material sometimes also exudes from around the mantle plates and solidifies in heavy columns or stalactitic masses on the shell. This substance containing some cyanides, is highly deliquescent. Upon several occasions, samples which have been brought into the office in the evening have disappeared into puddles of liquid during the night. Fig. 3 shows some fine specimens. A fairly complete analysis of this material showed the following complex composition: SiO_2 , 1.11 per cent.; $\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$, 3.17; CaO , trace; MgO , 0.22; K_2O , 46.15; Cl , 16.94; Na_2O , 18.08; CO_2 , 9.64; CN , 7.84; NH_4 , none; $\text{Fe}(\text{CN})_6$, 5.25; CNS , 0.17.

The dissimilarity between the composition of this material and the crystals is striking. The excess of potash over soda is noteworthy, as is also the presence of considerable percentages of chlorine in both stalactites and crystals. As will be seen from later analyses of flue dust, an appreciable amount of chlorine, doubtless practically all charged, is carried out of the top of the furnace. Although no special investigation has been made, we know some of it originates in the coke, which is quenched with water rich in chlorides; also, it is doubtless carried by some of the ores as chlor-apatite or other chlorine-bearing mineral.

Whereas certain other investigators have shown considerable percentages of cyanide compounds in dust samples collected through the shell of the blast furnace, or from the flue dust, only the last-mentioned sample, of all collected in this investigation, has shown more than a trace of cyanide. While probably present in considerable quantities in the hearth and bosh, the alkali cyanides forced into the upper zones are evidently almost entirely decomposed before leaving the furnace.

In the case of one furnace which has recently completed a long campaign, most of the cooling plates above the mantel burned out because of failure of water at one time or another. After the wall had worn thin, it was necessary to use a spray cooling pipe on the outside of the shell. To prevent water from entering the furnace, the feed-water pipes were cut off and the holes through the shell covered with "scab" plates on top of asbestos pads. After several months, when these plates were removed, it was found that some of these asbestos pads were dyed deep yellow, and several showed greenish and bluish tinges, indicating complex cyanides. When an old furnace is laboring under high pressure, and is seen to be gassing freely about the mantel, a strong acrid odor, doubtless ammonium chloride, is sometimes evident.

4. *By Combination with the Brickwork or as an Accretion in the Form of Cyanide, Etc., and Removal when the Furnace is Blown Out.*—This is, of course, a relatively small but very interesting part of the alkalis charged during a campaign. Several authors have suggested that destruction of brickwork in certain stacks, particularly in the middle and upper zones, was due to action of alkalis. However, so far as chemical action is concerned, rather more importance has been attributed to reaction between carbon monoxide and iron oxide spots in the brickwork, with subsequent disruption of the brick mass. The destruction of linings is doubtless hastened somewhat in furnaces in which the burdens are relatively rich in alkalis.

The average sample of the hearth brick used in one furnace lining at Bethlehem by careful analysis showed a total alkali content of 1.60 per cent. After blowing out, a sample was secured of a number of these brick near the bottom of the hearth, appreciably discolored when broken, which showed a total alkali content of 3.36 per cent. Higher up in the

lining the alkali enrichment is greater. Samples of pure white, mixed cyanides, etc., have been recovered in various plants from protected crevices in the brickwork, around the boshes after furnace campaigns, and a great deal more would be found if the contents of the stack could be removed dry.

The presence of ammonia around furnaces directly after being blown out is, of course, well known. Sometimes enormous quantities are evolved, the odor being apparent until the entire lining is removed and the hearth cleaned out. The chief source of ammonia gas is undoubtedly the alkali cyanides and ammonium chloride which decompose in the hot water or water vapor introduced to cool down the furnace. In passing, it is interesting to recall that a very small part of the cyanide produced in a furnace is fixed and carried down into the salamander in the form of the peculiar compound, titanium cyano-nitride. The excess accumulation of alkalis in the brickwork, or in salt deposits in a furnace lining after a campaign, may literally amount to several tons.

5. *By Evolution in the Gas.*—The heavy flue dust which is carried out of the top of the furnace by the gas current, of course, contains approximately its normal percentage of alkalis. The finer portions of ore, stone and coke dust doubtless average somewhat higher in alkali content than the entire materials as charged. All the remaining potash and soda in the burden, not previously accounted for, is sublimed and passes out in the form of various salts or compounds as a fine fume.

Various investigators have studied blast-furnace flue dust with special reference to its potash content, but almost entirely from a scientific standpoint. It is true that at least one American and one foreign patent have been issued for the recovery of potash in flue dust, but no evidence has been found that any practical application has thus far been made. And, so far as can be ascertained, the first commercial disposition of the flue dust as a fertilizer was made by us at Bethlehem somewhat more than 2 years ago.

The progress of the alkalis in our practice will now be traced from the furnace top, through the gas mains, washers, stoves, boilers and stacks.

Alkalis in Dry Dust Between Furnace and Washer

Besides the dust carried out by the relatively small amount of gas escaping from the furnace top, and the insignificant quantity permanently deposited along the mains, all the dust under the above heading is removed from the dust catcher. Representative bi-weekly samples of the dust-catcher flue dust were taken over the same period during which the slag samples were secured and analyzed, with the results shown in Table 4.

TABLE 4.—*Analyses of Dust-catcher Flue Dust*

Furnace	Period Included, 1916	Fe	Ign. Loss	CN	Total		Water Soluble	
					K ₂ O	Na ₂ O	K ₂ O	Na ₂ O
A.....	$\frac{3}{8}$ to $\frac{7}{8}$	40.7	17.3	None	0.63	1.45	0.40	0.35
A.....	$\frac{1}{6}$ to $\frac{1}{2}$	47.2	11.5	None	0.64	1.47	0.27	0.16
C.....	$\frac{3}{8}$ to $\frac{7}{8}$	59.8	1.7	None	0.50	0.94	0.29	0.30
C.....	$\frac{1}{6}$ to $\frac{1}{2}$	53.8	8.8	None	0.69	1.07	0.24	0.16
D.....	$\frac{3}{8}$ to $\frac{7}{8}$	40.5	18.1	None	0.59	1.30	0.40	0.42
D.....	$\frac{1}{6}$ to $\frac{1}{2}$	43.3	12.9	None	0.88	2.10	0.29	0.22
E.....	$\frac{3}{8}$ to $\frac{7}{8}$	49.3	9.8	None	0.50	0.90	0.31	0.30
E.....	$\frac{1}{6}$ to $\frac{1}{2}$	46.3	16.4	None	0.54	1.10	0.17	0.20
F.....	$\frac{3}{8}$ to $\frac{7}{8}$	46.6	14.6	None	0.42	0.85	0.20	0.22
F.....	$\frac{1}{6}$ to $\frac{1}{2}$	50.6	8.9	None	0.59	1.20	0.22	0.18
G.....	$\frac{3}{8}$ to $\frac{7}{8}$	47.4	8.1	None	0.55	1.50	0.25	0.20
G.....	$\frac{1}{6}$ to $\frac{1}{2}$	43.3	12.2	None	0.87	1.60	0.29	0.24
Average for all furnaces first period.....					0.53	1.16	0.31	0.30
Average for all furnaces second period.....					0.70	1.42	0.25	0.20
Average for all furnaces both periods.....					0.61	1.29	0.28	0.25
Ratio average total potash to average total soda.....					1 : 2.1			
Ratio average water-sol. potash to average water-sol. soda.....					1 : 0.9			
Ratio average water-sol. potash to average total potash....					1 : 2.2			
Ratio average water-sol. soda to average total soda.....					1 : 5.2			

A much greater percentage of the total potash than of soda is present in a water-soluble condition.

The entire absence of cyanides in this dust, and even in concentrated alkaline samples, recovered further along in the gas system, furnishes further corroboration of the fact that in normal operation practically no cyanide is carried in the gas current from the furnace top. That it may be an occasional constituent of flue dust, because of abnormal furnace conditions, is evidenced by several records at hand, notably by the following abstract.¹

The greater part of the CN compound dissociates higher up in the furnace and aids in the ore reduction. A portion passes away unaltered with the gases and is deposited in the flues. CN in small quantities was found in flue dust by Ledebur. It may, however, occur in considerable quantities as the Austrian Alpine Co. found to its cost. In the summer of 1901 about 20 tons of flue dust from the Hiefiau charcoal blast furnace in Styria was tipped into the river Ems, and an enormous quantity of fish were killed. Cyanogen was found in the dust and 2,800 pounds damages had to be paid.

Also, an instance of a large quantity of cyanides being evolved from

¹Sir I. Lowthian Bell: *Journal of the Iron and Steel Institute*, No. 2, 1882, p 555.

the furnace top occurred at the Colebrook Furnaces, Lebanon, Pa., several years ago. I trust that this occurrence will be enlarged upon in the discussion.

It will further be noted that there is a considerable enrichment of this flue dust by the alkali fume. At one furnace where dry gas is used in the stoves, there are two Brassert Witting dry whirlers and a long dry dust main with numerous dust legs between the dust catcher and stoves. This dust is increasingly fine and somewhat lighter in color at each of the points removed. Potash determinations at different times have shown a gradual increase in the successive samples, running as high as 5 per cent. at the stove burners.

Tests made at different times show that our dust catchers remove an average of about 100 lb. of dust per ton of pig iron produced. Using the average percentage of alkalis in flue dust for the four weeks' period, we find:

	Pounds	Per Cent. of Total Charged
Average potash in flue dust from dust catcher per ton of pig.	0.6	1.8
Average soda in flue dust from dust catcher per ton of pig.	1.3	2.3

The ratio of total potash to total soda present is roughly about the same as in the average charge entering the furnaces.

Effect of Primary Washers on Alkalies in Dust

At Bethlehem we have in service a type of tower spray washer, in common use in this country. All of the gas for stoves and boilers, as well as for gas engines, is washed, except from one furnace. Naturally, it would be thought that practically all of the alkaline material in the dust, most of it readily soluble in water, would be removed in the wet washers. The bulk of it is washed out, but it is a remarkable fact that much of the water-soluble alkalis remains in the gas current after leaving the washers. Calculations based on alkalis charged into the furnaces, lost in the slag, dust catcher, stack gases, etc., and recovered from the stoves and boiler settings, indicate that about 20 per cent. of the total potash (apparently only about 5 per cent. of the soda) entering the primary washers, passes through them. The explanation for this fact is that the particles of fume are in such an exceedingly fine state of division that they escape contact with the relatively large drops of water. In my opinion, any washer that will successfully clean blast-furnace gas rich in this fume must employ spray nozzles or other devices which will discharge water in a fine mist, thus insuring intimacy of contact between dust and water particles. Our washers at present perform the

function of selective precipitation, eliminating the relatively coarse and heavy iron ore and coke particles, while delivering the lighter particles of dust and fume into the primary clean gas main.

Alkalies in Dirt from Primary Clean Gas Mains

The mains carrying gas from the primary washers to stoves, boilers and secondary gas-cleaning plant accumulate dirt or mud gradually, and are washed out at intervals of every 2 or 3 months. This dirt is black when wet but after being dried is dark gray, and after the fine particles of coke dust are burned out, it is of a light-gray or reddish-white color. Sometimes drippings from a clean-out door along a gas main will form beautiful long stalactites. This material is in general appearance

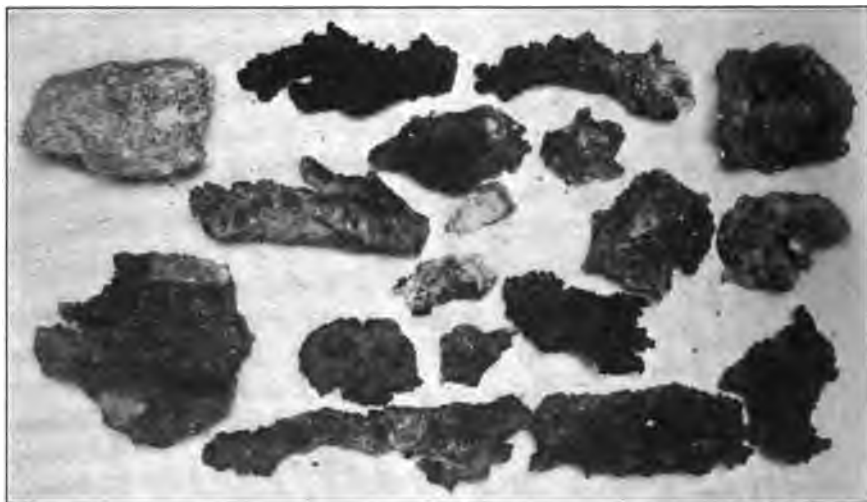


FIG. 4.—STALACTITES, RESULTING FROM DRIPPINGS FROM GAS MAINS.

similar to the stalactites mentioned as exuding from the furnace shell. A typical sample was found to have the following proximate composition, Fig. 4 showing its general appearance: SiO_2 , 0.28 per cent.; $\text{Fe}_2\text{O}_3 + \text{Al}_2\text{O}_3$, 0.68; CaO , trace; MgO , 0.25; K_2O , 44.63; Na_2O , 7.13; Cl , 37.22; CO_2 , 7.78; CN , trace; NH_4 , none. The almost utter absence of cyanide in this concentrated alkaline material is worthy of note.

Alkaline Dust in Stoves and Boiler Settings

The dark, wet dirt carried into the stoves, which at our plant are of the three-pass type, is dead burned, of course, the combustible or volatile constituents, etc., being expelled. The residue, in the form of an impalpable powder, collects to some extent in the bottom of the combustion chambers, a relatively large amount gathers in the bottoms of the second

and third pass checkers while the largest portion is carried out in the stack gases.

In the combustion chambers the dust frits together in friable masses. The accumulation is not great, the wells being cleaned out at intervals of about 2 months. However, the action on the brickwork is serious. All over the inner surface of the combustion-chamber walls, the lower surface of the dome, and the second pass checkers for a depth of several feet, a deep-green glaze is formed, or the brickwork is honeycombed.



FIG. 5.—FRAGMENTS OF LOW SILICA STOVE BRICK, HONEY-COMBED BY ALKALI FUME.

Along the fire-clay joints between the brick, the action is especially marked. Stove brick of a high silica content invariably glaze or slag, whereas those with a lower percentage of silica are more readily honeycombed or excoriated by the alkaline fume. Fig. 5 shows a typical sample of excoriated brickwork, while Figs. 6 and 7 exhibit surface glazed brick and a completely fused green slag respectively. Analyses of these materials show the following interesting results:

	SiO ₂	Fe ₂ O ₃	Al ₂ O ₃	K ₂ O	Na ₂ O
Green slag No. 1	60.66	2.30	14.55	16.30	6.03
No. 2	58.44	2.50	12.60	18.07	8.29
Original brickwork.....	79.70		19.70		0.25
Excoriated brick.....	40.17		26.70	20.84	8.04
Original brickwork.....	52.50	2.00	43.0		1.95



FIG. 6.—HIGH SILICA STOVE BRICK, GLAZED BY ALKALI FUME.



FIG. 7.—GREEN SLAG, RESULTING FROM COMPLETE FUSION OF HIGH SILICA STOVE BRICK BY ALKALI FUME.

The thermal efficiency of the stoves is reduced, of course, by this glazing, erosive and honeycombing action, though repairs seldom have to be made during a furnace campaign.

The fine, light-colored dust, which accumulates in the bottom of the second and third passes, is the material of commercial interest to us. It is cleaned out at intervals of 3 or 4 months. A typical pile of this dust freshly removed through one of the stove clean-out doors is shown in Fig. 8.

In the boiler houses the dust accumulates and is recovered every few days from the combustion chambers, from the passes and occasion-



FIG. 8.—TYPICAL PILE OF FLUE DUST REMOVED FROM STOVE, SHIPPED AS FERTILIZER.

ally from the horizontal flues leading to the stacks. It becomes increasingly fine and richer in potash in its progress. The boiler tubes, of course, become coated with the dust and must be blown off with a steam or air lance.

The white fume continuously issuing from the top of the stove and boiler-house stacks is approximately of the same composition as that reclaimed in the last passes.

Properties and Quantity of Alkaline Dust Recovered for Sale

As previously indicated, the dust recovered from the combustion chambers and stoves and boilers is a light, friable sinter, whereas the

larger amount reclaimed from the passes and flues is in an exceedingly fine state of division. When drawn out of the stoves or boiler settings hot, it runs almost like quicksilver, but it absorbs a certain amount of water readily, and becomes somewhat clammy and heavy.

An attempt was made to obtain some data on "angle of repose" of the fine dust. However, the cold, freshly dried dust behaves very differently from the hot material. A large, glass funnel at a fixed height over a plane surface was employed in the experiment. Comparative tests with white sand and two kinds of ore, prepared to various degrees of fineness, down to ultra 100-mesh, all yielded heaps with angles of repose of about 40° , whereas the stove-dust samples varied from 43° to 50° .

Sieve tests were also attempted. By careful sifting, it was found that all of the alkaline dust would pass a 300-mesh sieve. It is a true fume. The small residues remaining on the various sieves, when examined under a magnifying glass, were found to be chiefly eroded brickwork.

Specific-gravity tests, also, cannot be made with accuracy. The lightest dust, without compression, will run well below 0.50 apparent specific gravity. Samples taken from the checkerwork of five different stoves, and showing potash contents of 10 to 14 per cent., when tamped down gently in large glass measuring cylinders, showed the following varying densities: 0.68, 0.69, 0.73, 0.89, 0.96.

Three typical samples of fine stove dust were taken from different furnaces, and after being screened through a 300-mesh sieve and dried, were sent to an expert microscopist, skilled in examining such material. All of the samples showed the same characteristics, and 12 typical photomicrographs are reproduced herewith (Plates 1, 2 and 3) together with quotations direct from the microscopist's report.

"The general characteristics in this dust are: Very fine, spherical particles; the large proportion of them as single particles, but many united into groups and held together as by a gelatinous mass; the large majority of the particles are nucleated, the nuclei, however, varying greatly in size, as do the particles themselves; the nuclei may be several or but one in a particle, and are of a dark orange or brown color; the form of the particles is spherical to pear-shaped, while the form of the nuclei is practically spherical. The nuclei are not always confined to the inner or central part of the particles, but may be on its surface.

"One photograph of each of the samples A, B and C was taken by means of a paraboloid dark ground illuminator, of approximately 500 diameters. They tell nothing of the detail of the individual particles, but they do show clearly the grouping of the particles into aggregates of greatly varying sizes and forms. These aggregations are merely formed by the mutual attractions; that is, each particle is within the radius of attraction of the other. Photograph A shows particularly many single and individual particles, and represents a characteristic field of the samples. Photograph B shows a larger group, or one of the largest groups, while photograph C represents smaller groups. It should be said that a larger group is reduced easily to smaller ones by agitation or stirring or crushing. No detail can be possible at this magnification.



FIG. A.— $\times 325$.

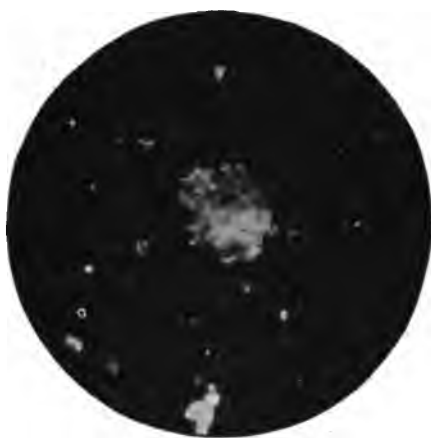


FIG. B.— $\times 325$.



FIG. C.— $\times 325$.

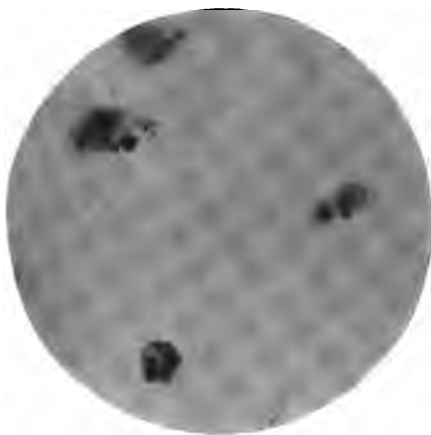


FIG. D.— $\times 1300$.

PLATE 1.—PHOTOMICROGRAPHS OF FINE FLUE DUST RECOVERED FROM STOVES.

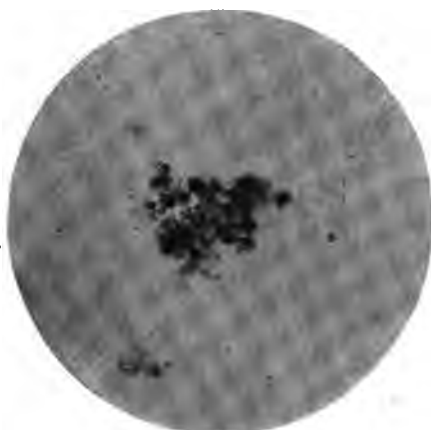
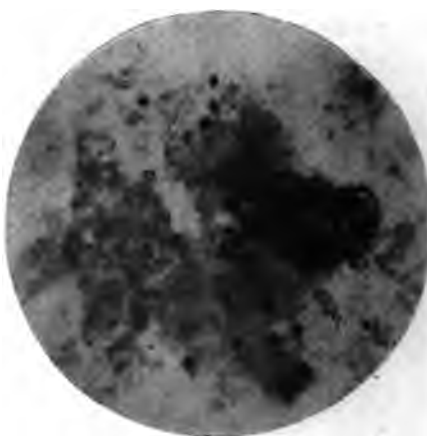
FIG. E.— $\times 1300$.FIG. F.— $\times 1300$.FIG. G.— $\times 1300$.FIG. H.— $\times 1300$.

PLATE 2.—PHOTOMICROGRAPHS OF FINE FLUE DUST RECOVERED FROM STOVES.



FIG. I.— $\times 1300$.

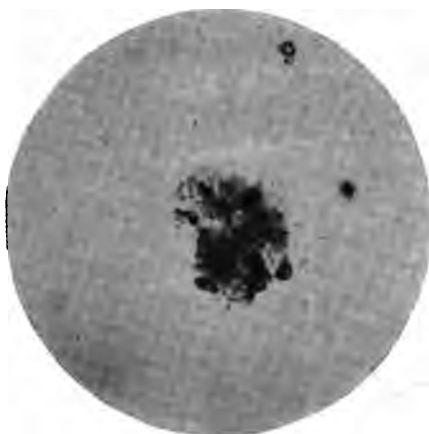


FIG. J.— $\times 1300$.



FIG. K.— $\times 1900$.

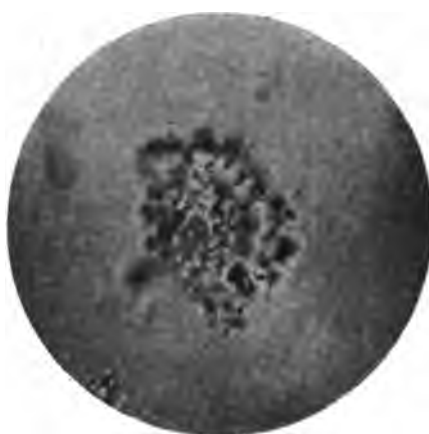


FIG. L.— $\times 1900$.

PLATE 3.—PHOTOMICROGRAPHS OF FINE FLUE DUST RECOVERED FROM STOVES.

"For detail, higher magnifications, with proper color screens and apochromatic lenses are essential."

Difficulty was experienced with these lenses in getting all the particles in a group into focus. The greatest trouble, however, was in getting both the vitreous groundmass and the orange-brown nuclei to show up together. Under the circumstances the photographs are creditable and interesting.

Aside from the three above-mentioned, all of the other photographs were taken under a magnification of 2,000 diameters (1300 as reproduced), with the exception of K and L which were taken under 3,000 diameters (1900 as reproduced). The three samples are each represented by four photographs (Plates 1, 2 and 3).

Knowing the exceedingly fine state of division and smooth, spherical nature of the ultimate particles in this fine fume, there is little cause for wonder that so much of it passes through a creditable wet washer untouched.

The water-soluble potash content of the dry, ignited dust, recovered from the stoves and boilers, having previously passed through the wet washers, will vary from about 5 to 20 per cent. Our practice has been to store the material in a large bin, capable of holding two or three carloads. The commercial recovery, begun in 1914, was not thoroughly systematized until the ensuing spring. Table 5 shows the weights and water-soluble potash contents, calculated on the dry basis, of 36 carloads

TABLE 5

Weight of Carloads in Net Tons	Percentage Water-solu- ble Potash (K ₂ O)	Weight of Carloads in Net Tons	Percentage Water-solu- ble Potash (K ₂ O)
33.0	10.05	34.7	7.04
31.3	11.20	32.8	6.22
26.1	9.95	36.4	10.24
24.9	10.65	33.7	7.15
25.4	10.10	32.5	12.00
24.0	11.31	27.3	7.27
25.7	13.96	29.9	11.38
21.1	8.32	31.3	14.22
27.4	9.32	35.0	7.69
27.7	12.17	34.1	9.01
27.9	9.62	26.3	7.55
24.2	11.91	35.1	7.82
26.1	15.92	39.4	6.05
24.0	15.07	27.8	7.48
34.5	11.00	26.4	8.76
28.2	10.27	32.1	8.60
30.6	10.37	32.5	8.58
33.9	8.73		
30.2	14.35	Total 1,073.5	Average 9.90
Total weight of water-soluble potash		106.3 net tons.	

of alkaline dust shipped in the period of 15 months, Apr. 1, 1915, to July 1, 1916. This represents all the dust removed from stoves and boilers serving four 500-ton furnaces, which could be recovered in normal operation, without special effort. Instead of being dumped into a dirt car, the dust was simply emptied into a central bin. A number of other carloads of lower-grade material, contaminated by flue dust from unwashed gas, were also shipped from the other furnaces, but are not included in this tabulation.

Complete analyses of three typical carload samples and one especially rich stove-dust sample, containing approximately 20 per cent. water-soluble potash, are given in Table 6.

TABLE 6.—*Analyses of Typical Carload Samples and a Rich Stove-dust Sample*

Sample No.	1 Per Cent	2 Per Cent	3 Per Cent	4 Per Cent
SiO ₂	22.22	18.07	18.50	22.66
Fe ₂ O ₄	15.24	11.25	2.31	2.30
MnO.....	0.67	0.67	0.75	0.79
Al ₂ O ₃	11.58	11.04	11.22	11.60
TiO ₂	0.06	0.05	trace	trace
CaO.....	11.69	12.22	13.14	12.49
MgO.....	7.35	8.49	12.35	6.76
ZnO.....	2.57	0.82	4.29	2.04
C.....	5.04	5.43	0.50	0.40
SO ₃	6.35	6.13	7.60	10.80
CO ₂	1.61	1.94	2.40	0.58
Cl.....	3.34	5.29	5.40	4.86
CN.....	None	None	None	None
CNO.....	None	None	None	None
CNS.....	None	None	None	None
NH ₄	None	None	None	None
Total K ₂ O.....	10.87	14.40	16.94	19.71
Total Na ₂ O.....	3.74	7.30	7.06	7.34
Water-soluble K ₂ O.....	7.55	12.00	15.92	17.04
Water-soluble Na ₂ O.....	3.28	3.94	4.44	3.98
Ratio water-soluble to total K ₂ O.....	0.69	0.83	0.94	0.86
Ratio water-soluble to total Na ₂ O.....	0.88	0.54	0.63	0.54
Ratio total K ₂ O to total Na ₂ O.....	1:0.35	1:0.51	1:0.42	1:0.37

No readily volatile constituents, such as may occur in the unignited dust, of course, are to be found after roasting.

In general, the ratio of water-soluble to total potash is higher than in the case of soda.

The preponderance of both total and water-soluble potash over the corresponding soda determinations is impressive, which seems to show the much greater removal of the latter in the wet washers.

If we assume that the total soda content in the stove and boiler dust averages 40 per cent. of the total potash content, and the water-soluble potash 80 per cent. of the total potash, as appears reasonable from the foregoing analyses, then the total alkalis recovered in the fertilizer material in the above period is found to be:

Total potash.....	118 gross tons.
Total soda.....	47 gross tons.

Calculating on the basis of pig iron produced in the above 15-month period, the alkalis recovered in terms of total charged are as follows:

	Per Ton Pig, Pounds	Per Cent. of Total Charged
Total potash recovered in fertilizer material.....	0.3	1.3
Total soda recovered in fertilizer material.....	0.1	0.3

The amount of potash recovered, though of considerable tonnage and value, is an insignificant percentage of the total charged.

At furnace plants using burdens rich in potash where the gas is unwashed, a considerable recovery still can be effected, though some precautions must be taken to avoid the coarser, raw flue dust. Several other furnace plants in the East, acting upon our suggestion, made directly or indirectly, have been recovering potash-bearing flue dust during the last year and a half.

Alkaline Dust Escaping from Stove and Boiler-house Stacks

Some idea of the relative amount of alkalis discharged per unit value of gas from blast-furnace stove and boiler stacks may be gained by observing the color and depth of the fume. Dirty gas, of course, will obscure greatly the fine, white, alkaline fume. In Western practice, with washed gas, the escaping stack gases show a thin, white color. In our practice the stack fume is much heavier.

About 2 years ago we made a series of tests, extending over a period of about a week, to determine the approximate amount of potash lost through our boiler-house stacks. The average results on the two largest stacks were as follows:

Average amount of dust per cubic foot flue gas (62° and 30''), grains.	0.12
Average percentage water-soluble K_2O in this dust.....	14.70
Average percentage total.....	18.60

Knowing the average volume of flue gas discharged, and the average volume of fuel gas consumed by the boiler houses, and produced by the furnaces, we can calculate roughly the weight of potash lost through the boiler-house stacks per ton of pig iron.

No actual tests have been made on potash losses from the stove stacks, but they should be slightly less per unit volume of fuel gas burned on account of the greater baffling action of the stove brickwork. We will assume these losses per unit volume of fuel gas to be 90 per cent. as great as from the boiler-house stacks. The soda-potash ratios will be assumed to be the same as in the average stove dust.

	Per Ton Pig. Pounds	Per Cent. of Total Charged
Total potash lost in boiler-house and stove-stack gases....	2.6	11.2
Water-sol. potash lost in boiler-house and stove-stack gases.	2.2	
Total soda lost in boiler-house and stove-stack gases.....	1.1	2.8

Alkalies Lost in Secondary Wet Washers

A considerable percentage of the primary cleaned gas at Bethlehem is further cleaned in Theisen scrubbers for gas-engine use. Practically all of the fine dirt is thus removed, the small amount passing through the fine gas main and into the combustion chambers of the engines having practically no cutting or scouring action, even when as much as 0.03 grain per cubic foot of gas is present. Samples of this dust, recovered near the engines, were of a smooth talcy consistency. The composition is similar to the finest stove dust.

The amount of alkalies removed in the secondary scrubbers, per unit volume of gas, is practically the same as the sum of the alkalies precipitated in the stoves (or boiler passes) and escaping through the stacks. Knowing the gas-engine consumption of fuel gas, the alkalies removed in the wash water, calculated on the same basis as for the stoves and boilers, is readily found to be as follows:

	Per Ton Pig. Pounds	Per Cent. of Total Charged
Total potash lost in secondary scrubbers.....	0.5	2.2
Total soda lost in secondary scrubbers.....	0.2	0.6

Balance Sheet for Alkalies Charged, Removed and Lost

Bearing in mind the extremely difficult nature of the problem, and the necessity for including a number of estimates, I will endeavor to sum-

marize a rough balance sheet for alkalies, charged and produced in our average blast-furnace practice. Further, it will be remembered that this balance sheet is approximately representative of our local conditions only.

TABLE 7.—*Balance Sheet for Alkalies Charged, Removed and Lost*

	Potash (K ₂ O)		Soda (Na ₂ O)	
	Pounds per Ton Pig Iron Produced	Per Cent. Total Charged	Pounds per Ton Pig Iron Produced	Per Cent. Total Charged
Total charged.....	22.4		35.6	
Lost in slag.....	4.5	20.0	6.3	17.0
Lost in fume, etc., from iron and cinder notches, and shell (est. 10 per cent. of slag loss).....	0.4	2.0	0.6	1.7
Lost in gases from top of furnace (est. 5 per cent. total gas losses).....	0.9	3.9	1.4	4.1
Lost by combination with brickwork of furnace, stoves, etc.....	negligible		negligible	
Recovered in dust-catcher dust.....	0.6	2.7	1.3	3.6
Lost in primary washers (est. by difference).....	12.5	55.9	24.6	69.5
Lost in secondary washers.....	0.5	2.2	0.2	0.6
Recovered in stove and boiler passes.....	0.3	1.3	0.1	0.3
Wastage from mains, stoves and boilers (est. 30 per cent. of quantity recovered).....	0.1	0.4		0.1
Lost in stack gases.....	2.6	11.2	1.1	3.1
Total.....	22.4	100.0	35.6	100.0

Apparently there is a greater loss of soda than of potash in the primary washer. This may be due to the greater solubility of the sodium salts, or to the larger size or possibly to the difference in contour of the fume particles of sodium compounds.

Cause of "Smoky" Gas

Every blast-furnace man is familiar with the phenomenon of "smoky" or "calico" gas. This title refers to a peculiar flecked, mottled or streaked appearance of the gas flame and is concomitant with more or less imperfect combustion. In the worst cases the gas will not burn at all in the boiler combustion chambers, even in the presence of a good wood fire. It is more frequently noticed in boiler than in stove practice, on account of the greater surface of hot brickwork, which promotes combustion, in the latter. To my knowledge, no trouble in gas engines has been attributed to smoky gas. The more or less imperfect combustion of the gas is accompanied by corresponding amounts of carbon monoxide in the flue gases. The phenomenon occurs in both dirty and primary washed gas, in some plants much more frequently than in others, but at no regular intervals. This gas is not only a curiosity, but, in plants where blast-furnace gas is at a premium for steam development, its appearance is the cause of concern to the steam engineering and blast-furnace de-

partments. The condition of smoky gas may continue for only a few minutes, or it may last for several hours. In a large plant, it can usually be traced to one, two or three furnaces, seldom being manifest in all the boiler houses, unless they are closely segregated and fed from a common gas main.

So far as I am aware, the cause of smoky gas has not been determined definitely and proved. We have taken numerous samples of gas when it was burning badly under the boilers, but the composition was always found to be normal. The temperature of the washed gas was not found to be above normal, indicating high moisture content, nor was there an abnormal amount of entrained water present.

I am indebted to C. H. Rich, Metallurgist, Alan Wood Iron & Steel Co., for the suggestion that smoky gas is concomitant with increased amounts of fume, in the gas, including cyanide or chloride. Determinations, made under his supervision, showed an enrichment of the fuel gas with one or both of these constituents when the gas was burning badly; elimination of the fume automatically restored a free-burning gas.

To test this idea further, I had several large samples of potassium chloride, sodium chloride and potassium cyanide prepared, in part by grinding to about 40-mesh size, and in part to ultra 100-mesh size. A suitable aspirating spray nozzle, operated by compressed air, was inserted through the breech of a boiler gas burner. The salts were introduced easily in a thin stream through this device. In the coarser condition, none of the salts had any appreciable effect on the character of the flame. However, the ultra 100-mesh product of all three produced in a clear flame exactly the same phenomenon as observed in smoky gas. In the case of sodium chloride, which was dry and easily pulverized, the flame could almost be extinguished.

The logical reasoning from these tests is that the cause of smoky gas is a purely physical one, due to the presence of an unusual amount of exceedingly fine fume in the gas, preventing proper contact for combustion between the particles or molecules of fuel gas and oxygen.

It has been our observation that smoky gas is simultaneous with high top temperatures in the furnace, but at irregular intervals. As earlier investigators have suggested, part of the alkali salts probably circulate in the furnace many times, on account of condensation in the upper zone. But the excess accumulation must be expelled at intervals, which naturally correspond to high top temperatures, low stock lines or high hearth temperatures. The fact that cold air admixture in the hot blast, when a furnace is producing smoky gas, will alleviate the condition temporarily, constitutes additional evidence that alkali fume, in considerable quantity, is being evolved during such periods.

A considerable amount of fume also will be noticed when smoky gas is burning. It is difficult or impossible to determine the approximate

average size of the ultimate fume particles in blast-furnace gas. The clammy, hygroscopic nature of the material makes microscopic work laborious. Judging from the photomicrographs made at 3,000 diameters, accompanying this paper, there are many minute particles which appear to have a diameter of about 0.01 in. in the photograph, and many others which do not exceed, say 0.04 in. The actual diameter of such particles would therefore be 0.0000033 in., and 0.0000133 in. diameter respectively. Assuming 0.3 grain of fume per cubic foot of fuel gas, and an intrinsic density of 2.0, the number of particles of fume per cubic foot of gas is found to reach the enormous total of over 30,000,000,000,000 and about 500,000,000,000 for the two sizes above mentioned. There would be about 32,000 and 8,000 particles per linear foot respectively, and the distance between centers of the particles would be only 0.000385 in. and 0.00124 in. respectively. These calculations are purely speculative, but they indicate the physical interference possible in the proper combustion of gas by the introduction of a quantity of fine fume particles. It will be readily understood that coarse, heavy particles of dust, such as exist in ordinary unwashed gas, per unit weight, will have much less effect in breaking up the continuity of the gas stream than fine fume.

In connection with Mr. Rich's original suggestion, I offer the above theory in explanation of the phenomenon of smoky gas. If there are other theories, or objections to this one, I trust that proper publicity will be made.

The obvious method of preventing smoky gas is simply to clean the gas more thoroughly, though this is more easily said than done.

Methods for Further Recovery of Potash

In our present gas-cleaning practice, it appears that there is a loss in the primary washers alone of over half of the total potash charged, or about 12 lb. per ton of pig iron produced, though, as mentioned later, this amount is probably a little high. The amount recovered, while appreciable, and representing almost clear profit, is seen to be an insignificant part of the total, less than 2 per cent. Potash lost in the slag, around the shell and from the top of the furnace, for all practical purposes, is lost beyond recovery. The greater part of the alkali content of the flue dust removed from the dust catchers could be recovered by leaching in water, but the percentage is too low to justify reclamation in this way. However, the potash now lost in wet washers and from stove and boiler-house stacks offers a legitimate and inviting field for its recovery. According to our balance sheet, it appears that about two-thirds of the total potash charged is now lost in the wash water and stack gases, or about 15 lb. per ton of pig iron produced.

About 2 years ago we obtained estimates for cost of recovery of the flue dust from several of our large boiler-house stacks, both by filtering

and electrical precipitation methods; but the relatively high cost and the uncertainty in the potash market, deterred an actual installation. Later it was realized that the principal loss of potash was not from the stacks.

At intervals during the course of the last year or more, we have had in operation an experimental Cottrell electric dust precipitator, connected to the raw gas main leaving one of the dust catchers. It is not my purpose in this paper to discuss the operation of the unit, except to state that practically all the dust and fume entering the treater could be precipitated successfully. The color of the dust recovered varied from a light to a dark gray. Several samples were analyzed and showed a potash content of about 10 per cent. The total dust leaving the dust catcher is evidently very much richer in potash than the relatively heavy particles constituting the dust in the dust catchers. However, with the above knowledge at hand, and judging from check calculations on total dust leaving the dust catchers and its theoretical potash content (by difference), it is my opinion that the estimated weight (by difference) of potash lost in the primary washers is somewhat, though not greatly, too high. For our average practice, there should be not less than 12 lb. of total, and probably about 9 lb. of water-soluble potash, now lost in washers and stacks, per ton of pig iron produced.

It may be mentioned that by weak acid treatment part of the insoluble potash content in flue dust may be rendered water-soluble, though this is not likely to be of practical application. Also, the soluble alkali salts can be recovered in tolerably pure form by leaching and evaporation.

It is not my purpose to develop at length the commercial phase of the subject. With the foregoing figures and present potash-fertilizer value as a basis, anyone can readily determine what a very attractive proposition is the recovery of potash from blast-furnace gas at the present time. A word of caution may be appropriate. As previously indicated, the weight of potash charged per unit of iron produced is above the average at Bethlehem. There are only two or three apparently practical methods for recovery on a large scale of potash from blast-furnace gas, and they are expensive and as yet untried for blast-furnace conditions. The price of potash is certain to fall after the war.

On the other hand, the recovery of potash in connection with the thorough dry cleaning of blast-furnace gas, with certain obvious advantages as against wet cleaning, is an attractive proposition to plants now suffering from burdens rich in alkalies. I venture to predict that in the future dry cleaning will be adopted in many blast-furnace plants, and that many thousands of tons of potash, hitherto wasted, will be reclaimed. Thus will our national resources be strengthened in this important raw material, and the blast furnace will have added another material to its increasing list of byproducts.

The subject of alkalies in blast-furnace practice is of a difficult and

complicated nature. It will afford much food for thought, both for the philosopher and the practical furnace operator.

In conclusion, I wish to acknowledge especially the coöperation of F. O. Kichline, Chief Chemist, who supervised the difficult analytical work presented in this paper.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Anthracite Stripping

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(New York Meeting, February, 1917)

Introduction

STRIPPING is the name given to the process of removing clay, rock, or other cover from deposits of coal or ore. In this paper it is intended to cover the methods used in carrying on this operation in the anthracite coal regions of Pennsylvania.

In doing this, attention will be chiefly devoted to those methods wherein practice in different sections of the region differs, or is susceptible to change or improvement, though it is intended to give as complete data as possible concerning all stages of the operation.

The data from which this paper has been prepared have been collected and submitted by a committee representing the larger users of stripping methods, of which committee the writer has acted as chairman. Data and assistance have also been obtained from various mining men, engineers, contractors and manufacturers of stripping equipment, to all of whom our appreciation is due.

Early History and Statistics

The earliest mining operation on a commercial scale in the anthracite region was a stripping operation. This was the famous Quarry Mine at Summit Hill, where this method was used only because no other method of mining a deposit of the nature found there seemed feasible to the early operators. A description of this stripping from a report published in 1821 is as follows:

"The coal mine at present worked by the company lies on the top of a mountain, and appears to extend over some hundred acres of land covered by about 12 ft. of loose black dirt resembling moist gunpowder, which can be removed by cattle with scrapers, and thrown into the valley below so as never to impede the work. The thickness of the coal is not known, but a shaft has been sunk in it 35 ft. without penetrating through. More than an acre of mine has been uncovered and presents a huge rock of coal, which is easily quarried without blasting."

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PLATE A.—PANORAMA OF STRIPPING SHOWN IN FIGS. 6 AND 6A.

There is also an early record of a similar quarry at Beaver Meadow and there may have been others whose record has not come down to us. After these first strippings, however, there is no record of such operations for many years. The first strippings, as we now know them, were commenced about 1864 when William Milnes carried on a hand-stripping operation at Jeanesville. Even this was a comparatively isolated instance and not until the late '70s did such operations become a general practice. About 1874, strippings were carried on at Beaver Meadow and in the Panther Creek Valley, as well as in other parts of the region. Thereafter, throughout the middle and southern fields until the advent of steam shovels, and afterward, immense quantities of overburden were removed in such operations, as is attested by numerous overgrown and forgotten spoil banks yet remaining along the hillsides of the coal valleys.

The literature of anthracite strippings, which is very meager, begins about this time with articles in technical journals descriptive of certain individual operations. Also, in the records of the Second Geological Survey of Pennsylvania there is a report by H. M. Chance on Anthracite Mining Methods and Appliances, published in 1883, which gives interesting plates and descriptions of early strippings.

The U. S. Geological Survey has recently commenced the collection of stripping data for the anthracite and bituminous fields, but the anthracite record is yet very incomplete, owing partly, at least, to a lack of detailed information in the reports furnished by the anthracite operators.

The earliest strippings were all hand excavations and the methods employed were much the same as those now used for very small operations. The operation consisted merely of digging out the stripping area with pick and shovel and removing the excavated material to the nearest available dumping place in carts or wheelbarrows. Horse-drawn scrapers were used in the larger jobs, some of which were extensive. Where the stripping became of considerable depth, and especially in rock, a horse-operated derrick was used to elevate material from the lowest levels.

The first steam shovel came into the anthracite region in 1881 and was used at Hollywood by L. E. Klotz, Contractor for C. Pardee. This was 6 years earlier than the first date on which steam shovels were used in ore mining in the United States. It was one of the early type of Oswego shovels with a dipper of 1 yd. capacity and weighing 30 to 35 tons. The next shovel was used by J. W. Crellin at Silver Creek. This was larger, and operated a dipper of $1\frac{3}{4}$ yd. capacity. Shovels came into use rapidly after this and this general type of equipment has remained in use since, though the sizes of the shovels have gradually increased until now 70- and 80-ton shovels are the rule and larger equipment is coming in.

During the latter part of the '90s some experiments with aerial tramways were made, but their capacity proved low and they were soon abandoned. These have also been tried in ore-mine stripping with equal

lack of success, though they are used almost to the exclusion of other equipment in quarries.

Anthracite strippings are a notable example of the way in which labor-saving devices have held down operating costs in the face of steadily advancing labor costs. In the early days when \$1.10 was the regular wage for a 10-hr. day, unit stripping costs were around \$0.26 per cubic yard for clay excavation and \$0.60 for rock. With present wages nearly double the early rate, stripping costs range from \$0.16 to \$0.20 for clay and from \$0.35 to \$0.40 for rock. This is due, largely, to the use of steam shovels and the increase in the size of the shovels employed.

Stripping is at the present time, and has been for very many years, one of the most important adjuncts to the mining of anthracite coal. In the northern regions, that is, the Lackawanna and Wyoming Valleys, stripping methods have been employed but little, because the veins are, as a rule, of limited thickness, and also because the pitch is so light as to render crop strippings unnecessary. In certain parts of these regions, however, these generalities do not hold good and stripping methods could be employed to economic advantage. Already a few areas have been stripped and others are contemplated.

In the middle and southern fields, on the other hand, strippings are an essential part of mining operations and form about 5 to 10 per cent., on a tonnage basis, of all mining done in these regions.

During the year 1914, the total yardage removed in stripping operations in the anthracite regions was 8,370,174 cu. yd. The division of this yardage between hand and shovel strippings was about 5 per cent. of the former to 95 per cent. of the latter. Due to the fact that a great deal of this work is done by contract on an unclassified basis, the percentage of rock removed as compared to other overburden is not obtainable, but is approximately 35 per cent. In the years since stripping operations were commenced in the anthracite region it is probable that at least 300,000,000 cu. yd. have been removed by stripping. Moreover, this figure does not include coal excavated by hand and by steam shovels from stripping areas, which is a substantial quantity.

Immense quantities of overburden have been removed in individual stripping operations in the anthracite field and some of these operations have extended over many years. One or two million cubic-yard strippings, and larger, are not unusual, and some of these have been in continuous operation from 12 to 20 years. To arrive at the record in respect to continuous operation, however, it is necessary to go back to the old quarry mine at Summit Hill again. This was in operation nearly 30 years and the total output of coal from it was nearly 2,000,000 tons. In cubic yards of overburden removed, however, and in depth of excavation, this old operation does not compare at all with more modern ones. Total excavation in the largest modern anthracite stripping is 4,500,000 cu. yd.

exclusive of coal, with a surface area of 53 acres and a maximum depth to the top of the coal of 150 ft.

Reasons for Stripping

The economical problem of producing the largest quantity of coal for the least expenditure of money can be assigned as the fundamental reason for the employment of stripping methods. There are, however, many factors that contribute to this fundamental one. A higher yield of prepared sizes is usually obtained by mining the coal in the open under conditions of natural light; a cleaner product is obtained for the same reason and also because of the removal of overlying impurities; also, the total output of a colliery can be increased by this method and fluctuations of output eliminated for the reason that the entire body of coal is thus exposed to view and can be attacked in few or many places as desired. It is usually one or the other of these considerations that determines the stripping of any area. In some cases these are such potent reasons as to induce the stripping of certain areas even where the profitability is otherwise doubtful. The doctrine of conservation also, is a consideration of some influence in certain cases.

Strippings on an economic basis, however, fall into two general divisions: Those that are undertaken to obtain coal at a cheaper cost than by ordinary mining, and those that are undertaken to recover coal that is unobtainable by ordinary methods, or obtainable only at a prohibitive cost. The latter is the larger division, because unless the coal to be uncovered is to be mined by other than the ordinary chute methods of inside mining—and often when the coal is to be mined by other methods—the output from a stripping is at a higher price than the average for the colliery. This is because the cost of inside mining is not appreciably decreased by uncovering the coal and because the cost of removing the cover must be added to the mining cost. There are, of course, many stripping operations, wherein the unit cost or the cost per ton of coal obtained or recovered is less than the average cost of inside mining. These occur, however, when conditions are exceptionally favorable; when, for instance, the coal after being uncovered can be excavated by hand or by steam shovel at a very low unit cost, which, added to the cost of removing the cover, is still lower than the cost of mining by ordinary methods. These cases, though numerous, are exceptions, and are undoubtedly becoming fewer in number.

It is, nevertheless, the preconceived idea of a great many people, even of some more or less intimately connected with coal mining, that coal is produced by stripping operations at a greatly reduced cost over general mining operations. That this is erroneous in the majority of instances is shown by the fact that lessors of coal lands often lower their royalty

rates in order to induce the stripping of areas that otherwise would remain unmined.

Limits in Area and Depth

The economical limits of stripping in area and depth are affected by so many factors, that this question is without exception the most important to be faced in stripping operations, and the variations in the values assigned to these different factors by those engaged in stripping operations are the most marked of all differences in practice. Some of the factors can be named, as: Quality of coal; margin of profit; classification of material to be removed; excavation of coal by chute mining, or by steam shovel; refuse to be handled, etc. As a rule the sum of the equation containing all of these is resolved for each operating company or district into an empirical ratio of cubic yards of cover to tons of coal which

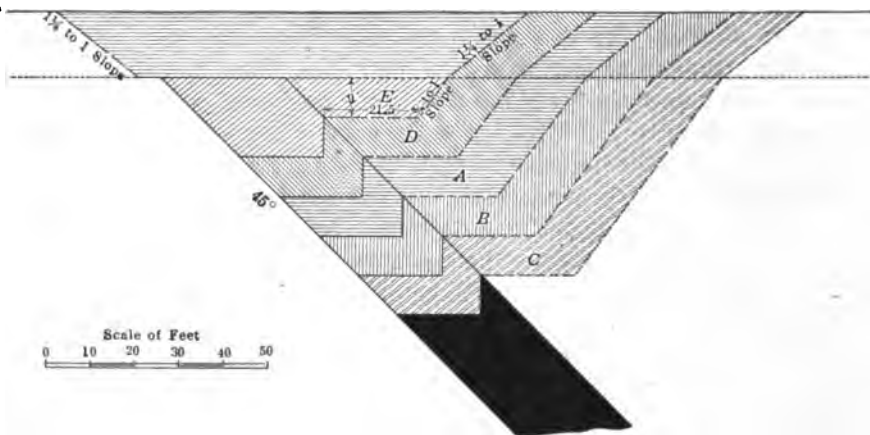


FIG. 1.

is then applied to other stripping work with perhaps slight variations to allow for local changes in conditions. Such a ratio once established for any region or operation sometimes remains unchanged for a long period and is applied in turn to each new stripping operation that presents itself. This method is not incorrect and in most cases it serves as well as a more complicated calculation, but there are many possibilities of a failure in the basic principle if a false value has been assigned originally to certain factors, or if certain factors have been neglected. Suppose, for instance, that it is decided to expend on coal recovered from the stripping shown in Fig. 1, an amount per ton equal to the average margin of profit of the colliery, the return on the investment being considered to be secured by certain factors or advantages that do not lend themselves readily to calculation in exact figures. Perhaps the output is falling off, or possibly the coal to be uncovered is of exceptionally good quality, and may improve the

car yield of the colliery a point or two. Then this cost per ton figure is translated into a ratio of cubic yards overburden removed per ton of coal uncovered, say, still merely as an instance, $2\frac{3}{4}$ yd. per ton. Then the limits in area and depth are set to produce this ratio. This, on its face, appears to be satisfactory, but not when the operation is resolved into component parts as is shown on this diagram by the dotted lines. There, the lowest component part which comes within the limits of the ratio is

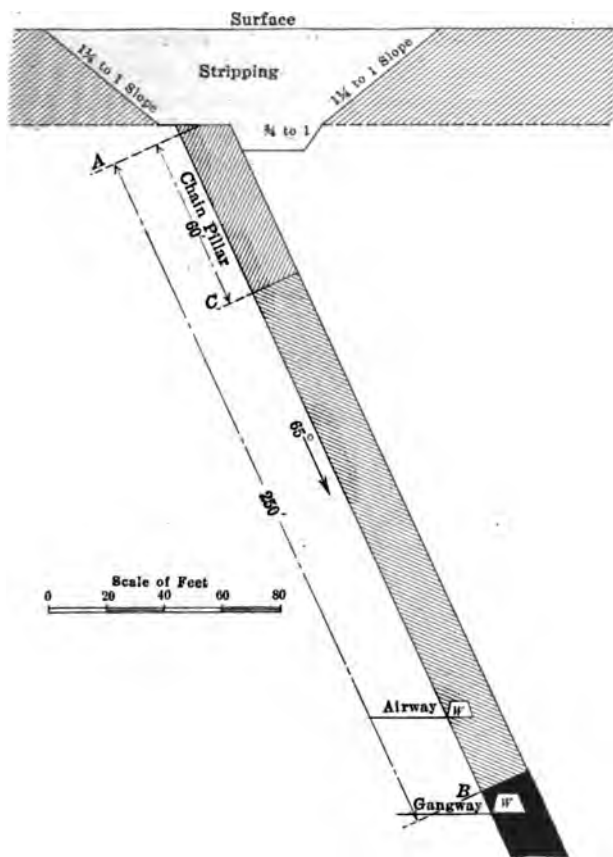


FIG. 2.

marked A. The component parts B and C are 3 cu. yd. per ton uncovered and $3\frac{1}{2}$ cu. yd. per ton uncovered, respectively, and their average is $3\frac{1}{4}$ cu. yd. per ton. In other words, they are operated at a loss, regardless of the fact that component parts D and E are operated at a considerable profit. It is required that certain marked advantages be gained by the removal of B and C to justify this operation. In these days of increasing costs of coal and narrow margins of profit, conservation can hardly be considered one of these advantages.

Another condition is illustrated in Fig. 2. This is a crop stripping of a character common to the southern fields, where the clay and gravel overburden must either be removed or a chain pillar of coal left below the surface to prevent the contamination of the prepared coal. It is to be assumed that the stripping is for a virgin area.

In this illustration it would be necessary to leave a chain pillar of about 60 ft. if stripping were not resorted to. The coal below this chain pillar can be mined at as low a unit cost for cutting and loading as would be obtained by mining all the coal from the gangway to the surface provided the cover were removed. By the latter method, lower unit development costs would be obtained, but the reduction would amount in this case to only a few cents per ton. Still, taking $2\frac{3}{4}$ cu. yd. of overburden to 1 ton of coal as our economical ratio, we find the ratio in this illustration to be 4 cu. yd. to 1 ton of coal in the chain pillar, and 1 cu. yd. to 1 ton of coal in the entire area between the gangway and the surface. The coal that can be strictly classed as stripping coal is, therefore, secured at a loss, and assuming that no other factors need be considered, the stripping should not be undertaken.

Of course, in presenting this illustration, I do not consider certain veins in the southern anthracite fields that are very thick and in which the character of the coal is so loose and friable that any excavation in them is likely to cause a run of the coal that is not stopped short of daylight. A chain pillar is obviously of little value in such a case, and if the coal is to be mined at all stripping must be resorted to.

These are only two instances of possibilities of fundamental error in preliminary calculations, but they serve to illustrate the point. In general, in the anthracite region, veins that are too thin to be worked at a

TABLE 1

Pitch, Degrees	Thickness of Vein, Feet	Remaining, Per Cent.
1. 15-25	10-20	55
2. 15-25	20-30	60
3. 15-25	Over 30	65
4. 25-35	10-20	50
5. 25-35	20-30	55
6. 25-35	Over 30	60
7. 35-45	10-20	40
8. 35-45	20-30	50
9. 35-45	Over 30	55
10. 45-60	10-20	40
11. 45-60	20-30	50
12. 45-60	Over 30	55
13. 60-80	10-20	50
14. 60-80	20-30	55
15. 60-80	Over 30	60

profit are not mined, even though such mining might not cause the colliery to be operated at a loss, and stripping operations should undoubtedly be treated in the same manner.

The above illustrations were assumed to be of virgin veins in order to simplify the explanations. Unfortunately, the majority of large stripings are of veins worked and reworked in days past in such a way that an accurate calculation of the amount of coal remaining is impossible. This problem can only be solved by a guess, as is done where contracts are let on the basis of payment by the ton for coal recovered. Fortunately, however, many stripings of worked-over areas have been completed at this time, or are so far advanced that a record of the coal actually recovered or remaining in the worked-over areas can be obtained. A provisional tabulation has been made of all these records that are available, which gives the average results shown in Table 1. Some of the figures from which this tabulation was derived are given in Table 2.

TABLE 2

Kind of Stripping	Thickness of Vein, Feet	Pitch, Degrees	Extent Worked	Percentage of Original Coal Remaining
Anticline.....	55	20	Robbed and robbed...	60.0
Crop.....	50	60-80	Mined and robbed.....	60.0
Anticline.....	40	15	Mined and robbed.....	50.0
Crop and basin.....	40	40	Mined and robbed.....	40.0
Crop.....	40	30	Mined and robbed.....	60.0
Crop.....	35	40	Mined and robbed.....	50.0
Crop.....	30	40	Mined and robbed.....	50.0
Crop.....	25	45	1st mining.....	82.5
Crop and basin.....	25	20-50	1st mining.....	70.0
Crop.....	25	20	Mined and robbed.....	50.0
	11			
	10			
Crop-3 splits.....	15	50	1st mining.....	70.0
Crop.....	16	20-35	1st mining.....	55.0
Crop.....	16	25-35	1st mining.....	50.0
Crop.....	15	50	1st mining.....	68.0

Tables 1 and 2 consist entirely of examples of stripping from areas mined 20 years or more ago. In more modern mining better extraction has been obtained, and this fact must be taken into consideration where the stripping of such an area is contemplated.

With all these factors to be considered, and with the varying margins of profit at different collieries kept in view, it can readily be seen that no universal ratio of yards of cover to tons of coal can be set as the profitable limit for stripping operations. At each operation this limit has to be worked out for itself. The ratios in use throughout the entire region

vary usually from 2 to 4 to 1, with an average of about 3 to 1. In some instances ratios as high as 5 to 1 have been used, but this means that the cost per ton, for stripping only, would be \$0.80 to \$1.50, to which must be added mining, preparation, and overhead costs.

Equipment and Operation—Characteristic Strippings

The equipment required for a one-shovel operation is about as follows:

1 70-ton shovel.	1 steam drill.
3 18-ton locomotives.	1 water tank.
20 5-yd. dump cars.	1 boiler.
1 star drill.	1 blacksmith shop.

Necessary rails, sills, pipe lines, tools, etc.

The total capital outlay for such an outfit is approximately \$30,000.

The average force required to operate a one-shovel stripping consists of about 35 men, roughly as follows:

1 foreman.	4 jackmen.	2 drillers, 8 helpers.
1 shovel engineer.	3 locomotive engineers.	1 boiler fireman.
1 craneman.	1 dump boss.	1 blacksmith and helper.
1 fireman.	6 dumpmen.	2 coal diggers.
1 watchman.	1 track boss.	1 driver.
2 laborers.	2 trackmen.	1 switchboy.

The wages paid these men amount to \$2,100 per month. The shovel engineer is paid \$140 a month, the craneman \$95, locomotive engineers \$0.25 per hour. These rates are all subject, however, to the recent increases granted the mine workers, ranging from 7 to 15 per cent.

When a stripping is decided on and its limits staked out by the engineers, an inspection of the ground determines the method of opening it. Usually the cuts at the higher elevations are made first. After that the problem is almost entirely a transportation one. Steady operation of the shovel or shovels is the object to be secured. Everything must contribute to this end—tracks and rolling stock must be in good condition, turnouts must be maintained, and the grades must be as easy as the nature of the ground will permit. If in rock, drilling and blasting must be kept well in advance.

The method of opening a stripping with either a Bucyrus 70-ton shovel or a Marion 60-ton shovel, which are the two types most widely used in anthracite stripping work, is as follows (Fig. 3A): For the first cut the track is laid on the surface along one limit of the stripping, usually the bottom rock side, and the shovel cuts down grade alongside the track until a depth of 9 ft. is reached, this being the maximum cut that the shovel can take and load overhead. When the first cut is completed for the length of the stripping, the track is laid in this cut and the shovel

again cuts down grade until a depth of 9 ft. below the first cut is reached. The shovel then continues cutting toward the other limit, the additional depth being determined by the depth of surface over the vein up to 30 ft., which is considered the proper maximum height for a clay cut. In working by the above method, it is necessary to leave a bench at least 13 ft. in width for the laying of the track. Local conditions, as a rule, render it impossible to maintain any such plan for the entire life of a stripping.

The first cut as described above is always the first made in a stripping except in the case of what is known as a side-hill stripping. Here the track is laid on the surface and the shovel started at an elevation that will give the required cut at the vertical limit.

Rock cuts are usually made from 22 to 25 ft. in height, though more recent practice is to keep the height down to 12 or 15 ft. This height depends somewhat on the nature and hardness of the rock. The lower height seems better for very hard rock, the reason being that the rock is broken up by blasting better than in the case of a higher bank, especially the upper portion of the bank, and consequently is more easily loaded. A saving of 25 per cent. in cost per cubic yard is claimed for this method.

Drill holes for blasting are arranged in parallel rows, usually three, with the holes 12 to 20 ft. apart and the holes in each row staggered. From 15 to 25 holes are fired in a battery, or more if possible. After the holes have been drilled a charge of from three to eight sticks of 40 per cent. dynamite is used to "spring" each hole. By "springing" is meant the blasting of a pocket at the bottom of the hole to take a sufficient charge of black powder. It is sometimes necessary to spring a hole two and three times. The charge of black powder then used varies greatly, according to the nature of the rock and the amount of powder that can be put into the hole. The pocket itself, and the hole for about 2 ft. above the pocket, is filled with powder and the remainder of the hole with clay or coal dirt.

For 25- and 30-ft. holes, "Star" type churn drills are usually used with a customary diameter of 4 in. In solid rock the progress is about 30 ft. per shift of 9 hr., or an average of $3\frac{1}{3}$ ft. per hour. For holes 12 ft. deep or under, a steam tripod drill is used and about double the progress of the churn drill, or $6\frac{2}{3}$ ft. per hour, is made.

Average costs for drilling and blasting are as follows per cubic yard of rock excavated:

Labor drilling and charging, depreciation equipment, etc....	\$0.045-\$0.065
Powder.....	\$0.055-\$0.080
	\$0.100-\$0.145

The tracks to the dump are always on an ascending grade of at least 1 per cent., though usually higher. Four per cent. is common and grades as high as 7 per cent. have been used. The grade of the tracks in the

stripping pit is governed by the necessary rise in elevation to reach the dump. The locomotives used vary in size up to 20 tons, the latter being about the heaviest type that can be used safely on a dump of any height. A 20-ton locomotive will push:

- 10 $4\frac{1}{2}$ -cu. yd. cars on a 1 per cent. grade.
- 8 $4\frac{1}{2}$ -cu. yd. cars on a 3 per cent. grade.
- 6 $4\frac{1}{2}$ -cu. yd. cars on a 4 per cent. grade.

The general, and best, practice for stripping tracks is to use 60-lb rails and nothing under a No. 6 frog. Curves should be kept to under 10° , though 20 to 25° curves are used, especially in forming a dump.

Dumps are made of all heights and sizes, though there is less maintenance cost with heights of about 25 ft. Dumps of greater height settle and slip easily, especially in wet weather.

The cars most widely used in stripping work are the Eastern and Western type side-dump cars. The Eastern type is of $4\frac{1}{2}$ to $5\frac{1}{4}$ cu. yd.

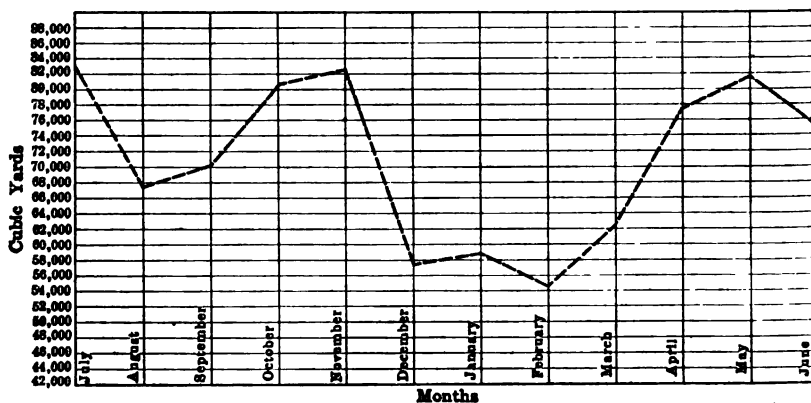


FIG. 3.

capacity and the Western type of 4 cu. yd. Some 8- to 10-cu. yd. cars are in use and the results obtained seem to be satisfactory.

Under proper conditions, outputs as high as 30,000 cu. yd. per month have been obtained for one shovel in clay. The average, however, is only about 18,000 cu. yd. for clay and 10,000 to 12,000 for rock. The output from strippings varies considerably according to the season of the year. The curve shown on Fig. 3 illustrates this for the entire stripping operations of one company averaged by months over a period of 4 years. This curve, while affected by other factors, serves to illustrate the drop in output during the winter months.

If the stripping is not too deep, all the excavated material can be removed by locomotives. In many cases, however, this is not feasible and hoisting planes must be resorted to. Practically without exception,

even in the largest operations, these are single-track planes operated by small second-motion hoisting engines with a capacity of about 150 dump cars per day, or about the output from one shovel. The practical problem involved in putting these planes down along the steep sides of the average pit is often a serious one. Some of the planes are anchored on a slope of 50° to 60° pitch by bars sunk into the solid rock to which the road-bed is tied, presenting a very interesting sight. While nothing can be said against these small hoists for a one-shovel stripping, it is undoubtedly bad practice to use them in the larger operations employing two or more shovels. There are practically none of these that cannot be laid out so that the output from two shovels can be brought to the foot of one plane, and this plane should be equipped with a hoist capable of handling with ease 300 and more cars per day. This plane can be either single-track or double-track, but the grade should be maintained at about 20°, which is the average for the single-track planes now in use. Some figures have been worked up showing the comparison of the cost of the two varieties of planes, taking a double-track plane handling only the output of two shovels which would allow the greatest advantage of comparison possible to the small hoist. The first cost of the small-hoist job is very low, as the hoist itself is usually picked up second-hand around the collieries. It would be something as follows for a 300-ft. length of plane:

Hoist.....	\$500
Tracks, track material, rope, etc.....	700
Grading for hoist and plane.....	1,000
	\$2,200

For the double-track plane with the larger hoist the figures would be:

Tracks, track material, ropes, etc.....	1,100
Hoist.....	5,000
Hoist house, pipe lines, etc.....	800
Grading for hoist and plane.....	3,000
	\$9,900

To operate the single-track plane two top-men, two bottom-men, one locomotive engineer, one hoist engineer, four men and a boss on the dump, are required, while the double-track plane would require three top-men, three bottom-men, two locomotive engineers, one hoisting engineer and seven men and one boss on the bank.

The comparative cost would be as follows:

	Single Track	Double Track
Labor per day.....	\$17.88	\$26.21
Power.....	4.30	6.48
Interest and depreciation, 15 per cent..	1.00	4.00
	\$23.18	\$36.69

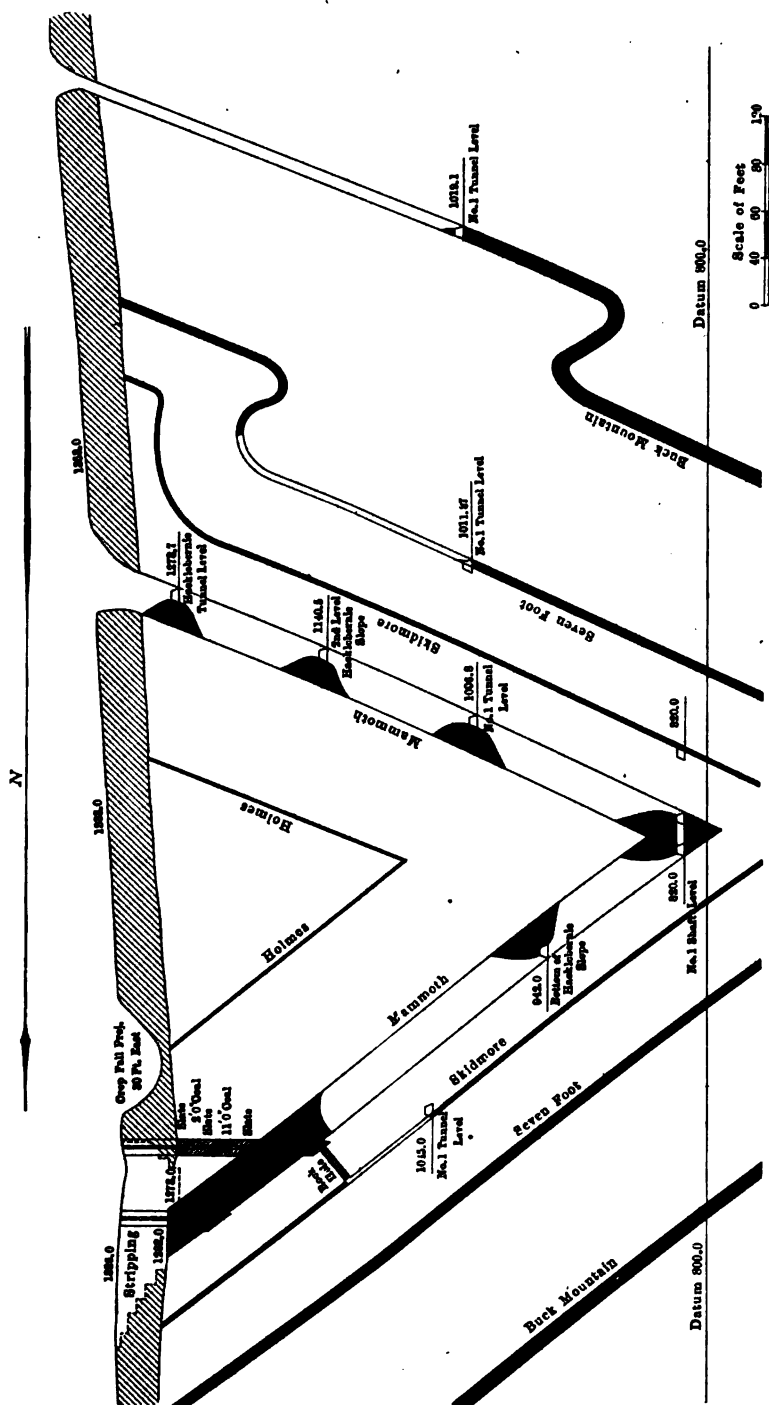


FIG. 4.

and therefore a gangway is driven in a small underlying vein from which chutes are driven up to a point opposite the lowest edge of the chain pillar and rock holes are then driven through into this pillar.

Fig. 5 is of a large basin stripping. This was operated for many years and the various stages of excavation are shown illustrating the characteristic methods of opening strippings of this kind. This stripping is of a virgin vein. Its width is 300 ft. and its length, 4,800 ft., maximum depth 100 ft. of cover, and cubic yards of cover per ton of coal, 3.46. This stripping was in operation from 1900 to 1915, starting with an Oswego shovel and changing in 1909 to a 70-ton Bucyrus.

Fig. 6 illustrates an anticlinal stripping. It is of a worked-over area where it is estimated that 60 to 70 per cent. of coal remains. Upon this

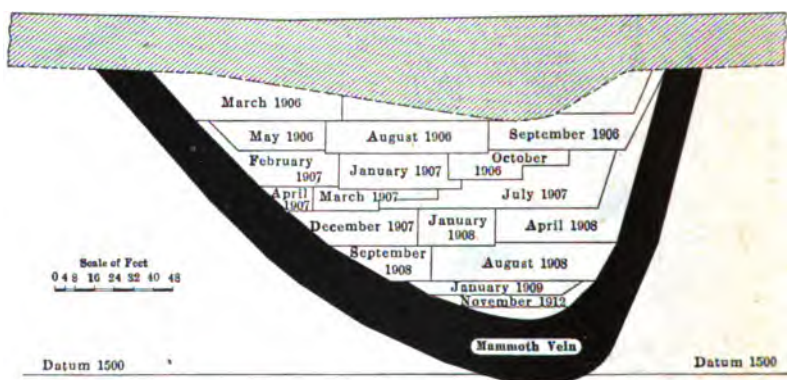


FIG. 5.

percentage depends its profitableness, as it has been undertaken primarily to form a final barrier against a fire that has been raging to the east of it for many years. The vein is 55 ft. thick and is on 20° pitch. It has been robbed and robbed again, but because of its thickness and the unhandy pitch, as well as the time of the mining, which dates back to early '50s, it is thought that not over 35 per cent. of the coal has been extracted. Fig. 6A shows a plan view of this stripping and Plate A is a panorama photograph taken looking north from the south side during the progress of the operation.

Engineering Methods

Engineering methods begin with the location of a stripping and the preliminary estimates of its profitableness, and continue through the calculations of yardage removed in the various stages of excavation down to the final tabulation of results obtained. In laying out the stripping and making the periodic estimates of excavation, a high degree of standardization prevails throughout the anthracite region. A base line is laid out

parallel to the length of the stripping, with stakes at regular intervals of 20, 25 or 27 ft. These are numbered in order, beginning with one. At each of these stakes lines are laid out at right angles to the base line

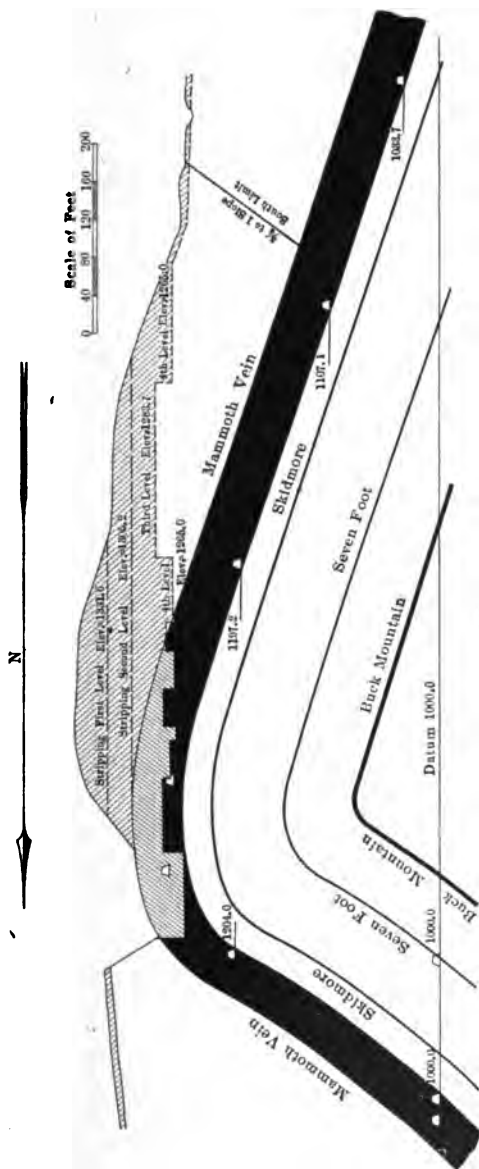


FIG. 6.

across the width of the stripping and stakes set at 20- or 25-ft. intervals. Sometimes these are numbered, but the best way is to assign letters to them, beginning with A. At any stage of the stripping operation these

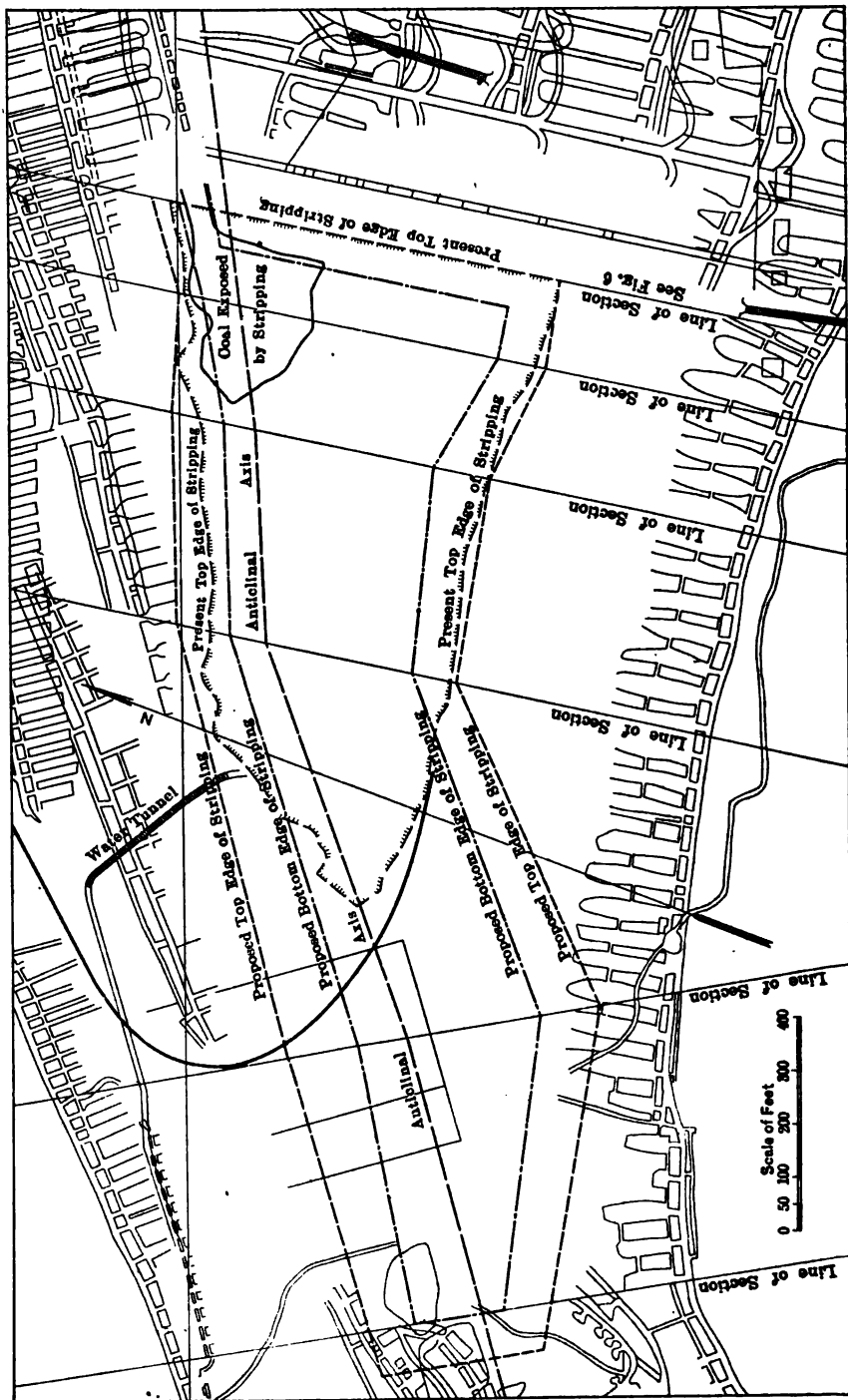


FIG. 6A.—MAP SHOWING PLAN OF STRIPPING.
(To accompany cross-section shown in Fig. 9.)

points can be readily relocated and levels run over them to ascertain the yardage removed. Cross-sections on each of the right-angle lines are plotted in the office on rolls of printed cross-section paper or tracing cloth. These plottings are invariably on a scale of 10 in. to 1 ft.

The use of 27-ft. intervals along the base line is worthy of note, as it is rapidly coming into practice. By this method the area in square feet of the cross-section on each right-angle line can be translated into cubic yards without any multiplication.

At the lateral limits of a stripping it is customary to place limit stakes, using large stakes painted red or white to give them special distinction and permanence.

Classification

The overburden in stripping operations is classified usually as rock and earth. This classification is purely for convenience in carrying out the contractual relations when the contractor has undertaken to remove earth at one price and rock at another. Where the price of removing rock and earth is the same, no classification is required and the rate established is known as an 'unclassified rate. The definitions of rock and earth vary. As a rule, boulders of 1 cu. ft. and over in size are classified as rock, and all smaller as earth. Most variations in classifications occur at the gradation zone between earth and rock where the material to be excavated, though stratified and of the appearance of rock, is soft and of the consistency of earth. In some cases a third classification known as loose rock is employed, as is witnessed by the following definition:

"Loose rock will include all stone and detached rock found in separate masses, containing not less than 3 cu. ft., nor more than 1 cu. yd.; also all slate, coal or other rock, soft or loose enough to be removed without blasting, although blasting may be resorted to; also stratified rock in layers of 8 in. thick and under, separated by strata of clay."

As a rule, however, rock and earth are the only classifications needed. One particularly complete definition of these two classifications is as follows:

"Excavations shall be paid for under the following classifications:

"**EARTH**, which shall include clay, sand, gravel, loam, decomposed rock and slate, whether lying in place or not; stones or boulders containing less than 1 cu. yd., indurated clay or other earthy material, cemented gravel, and all coal, shale, slate, soft friable sandstone, and all other material in place except rock as hereafter defined; also stratified solid sandstone in layers of 8 in. or less in thickness when separated by stratified earth as above defined.

"**ROCK**, which shall include all solid sandstone in place in layers greater than 8 in. in thickness, whether separated by layers or earth as above defined or not, and all boulders containing more than 1 cu. yd."

The loose-rock classification is undesirable and the tendency is, properly, away from it.

The average rates paid for classified strippings have been touched on earlier in this paper. For unclassified work the average rate varies from \$0.20 to \$0.25 per cubic yard.

1916 Stripping Society Monthly Statement of Stripping																			
Month	Average Depth of Cut	Surface Area	Excavation	Mining of Coal	Loading	Time of Coal	Total Cost of Stripping per Ton	Cost of Mining per Ton	Cost of Hauling per Ton	Cost of Loading	Cost of Hauling	Cost of Loading	Cost of Hauling	Cost of Loading	Cost of Hauling	Cost of Loading	Cost of Hauling	Cost of Loading	Cost of Hauling
Year 1916	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S
January	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S
December	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S
Year 1916	A	B	C	D	E	F	G	H	I	J	K	L	M	N	O	P	Q	R	S

A — Stripping for Month
B — Coal Mined During Month
C — Coal Stripped and Completed
D — Coal Stripped and Completed
E — Coal Stripped and Completed
F — Coal Stripped and Completed
G — Coal Stripped and Completed
H — Coal Stripped and Completed
I — Coal Stripped and Completed
J — Coal Stripped and Completed
K — Coal Stripped and Completed
L — Coal Stripped and Completed
M — Coal Stripped and Completed
N — Coal Stripped and Completed
O — Coal Stripped and Completed
P — Coal Stripped and Completed
Q — Coal Stripped and Completed
R — Coal Stripped and Completed
S — Coal Stripped and Completed

Fig. 7.

Prospecting

Prospecting by diamond drills is of the utmost importance and the most successful stripping operations are those that have been most thoroughly laid out as the result of drilling. Holes are placed at proper intervals on cross-section lines sufficiently close to each other to insure, so far as possible, avoidance of error in the location and character of the vein or veins to be stripped. These cross-sections in badly distorted areas should often be as closely placed as 100 ft. apart. Prospecting by steam shovel is not to be recommended.

Special Devices, Methods, Accounting Systems, Etc.

Various special methods for keeping a record of the progress of strippings, their cost, economy, and other related data are in use. Some of the anthracite companies have kept records in great detail for many years covering the actual cost of stripping work, and the results obtained in the greater efficiency of their stripping operations are strong evidence of the value of such thoroughness. In some instances the contractor's costs and profits are watched as closely as are those of the company.

Keeping a careful record of the actual results from each individual

stripping cannot be too strongly urged. Fig. 7 is a reproduction of a sheet of this kind kept by one anthracite operator. The first columns can be passed over as merely giving tabulated data of value as reference. The final columns, however, show at each successive stage of the stripping operation the unit costs, based on the coal tonnage recovered, or to be recovered, of results to date, results for a completed section and probable final results. As the stripping progresses, a glance is sufficient to show the exact financial standing of the particular operation covered.

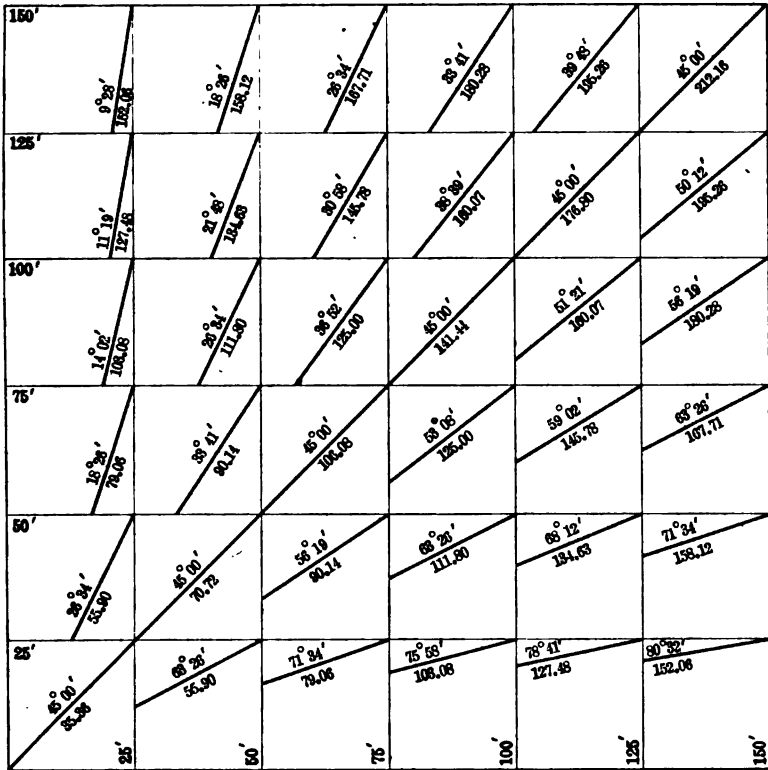


FIG. 8.

Another special form which is of great labor-saving value to the engineers engaged in calculating the monthly yardage excavations is shown in Fig. 8. The stripping area is laid out in squares by parallel and right-angle lines. The illustration shows squares of 25 ft., but a similar chart could be worked up for any dimension. By means of this chart the intersection of any two lines can be found with the minimum effort and with field calculations eliminated.

Yield and Waste

The yield from a stripping in per cent. of the total is naturally high. There are, however, certain losses such as the fuel loss; dump losses, where the coal is excavated by steam shovel; and various losses in digging and handling.

The fuel loss is, as a rule, the most important of these, as a large amount of coal must be used to supply fuel to the various shovels, locomotives, drills, etc., employed in stripping work and in a majority of cases this coal is cut by the contractor from the uncovered areas of the vein. In one large stripping employing eight shovels the consumption per day ran about 50 tons, which means a total consumption of nearly 15,000 tons a year. Another employing four shovels used 20 tons per day or about 6,000 tons per year. This loss is never less than 2 per cent. of the total coal uncovered and runs over 5 per cent. at times. A steam shovel of 60-ton class will use 2 to 3 tons per day. A 12-ton locomotive uses about 1 ton and churn drills and blacksmithing forges use about $\frac{1}{2}$ ton each. Little economies can be effected in this connection, such as requiring the contractor to furnish well-constructed boxes or bins wherever coal is deposited or stored. These not only save considerable tonnage in the course of a year but also have a desirable moral effect on all concerned.

When fuel is not cut from the strippings, and always until coal is uncovered, fuel must be furnished from the colliery or from other strippings. This question of fuel is a rather important one and there is some difference of opinion as to whether or not all coal used in the operation of the stripping should be furnished from the colliery, rather than cut from the uncovered vein. In cutting from the vein there is always waste, because only the best coal is taken. Moreover, the shovels and locomotives use none of the fine coal that is produced. In spite of this, it is impossible to figure the value of coal in the ground at anything except the profit that would be obtained by mining and preparing it for market, while against coal furnished from the colliery all the costs of handling it in the various stages of mining and preparation must be charged and the comparison in cost is distinctly in favor of allowing the contractor to cut his own coal from the exposed vein.

The second loss or the dump loss applies only to vein areas previously worked over. There the loss occurs in the zone between pillars and old openings filled with gob and coal that has sloughed from the ribs of the pillars. No satisfactory separation between rock and coal can be made here with a steam shovel and the percentage of loss is high—perhaps 10 per cent. would not be an abnormal figure in some instances. This loss can be reduced by various methods. All cars containing an appreciable percentage of coal can be sent to the breaker and the separation made there, or a separate cleaning or separation plant can be erected for

the handling of all cars too high in refuse to be handled economically at the breaker. Still another way is to dump all cars containing an agreed on percentage of refuse on a definite section of the dump and to employ laborers to pick the coal from the refuse by hand. This method has been employed with success at various places. A single laborer has loaded as high as 2 tons a day by this method and if this average can be obtained the method is profitable. The dumps for the old Summit Hill Quarry Mines, as mentioned earlier in this paper, are an example of the amount of coal that can be wasted into stripping dumps, as these dumps afterward took fire and burned with intense heat for many years. Evidently some progress has been made since that time in avoiding waste, but much yet remains to be done.

The other losses such as handling and blasting run as high as 5 per cent. at times, and the sum of all losses varies from 5 to 20 per cent., though the latter figure is unusual. Any reduction secured in any of these losses is true conservation.

Mining Coal from Strippings

The consensus of stripping experience is that it is more economical to mine coal from an uncovered area by hand loading into cars, or by steam shovel, than by the ordinary methods of inside mining. This is undoubtedly true as regards virgin coal areas and is usually true as regards worked-over areas, though here the difference is not so marked. When excavating coal in worked-over areas, it is necessary to excavate also all refuse and rock in old breasts, some of which might be left behind in chute mining. The recovery of coal from the strictly stripping area might also be less in a steam-shovel operation because of the various factors of waste that have been discussed previously. If, in addition, it is necessary to construct expensive planes to handle the product from the steam shovel, such an operation may be the more expensive, all factors considered, of the two. Ordinarily, however, the outside transportation cost will be found a little cheaper than the inside, unless the inside transportation can be handled without additional equipment, or men, especially if planes have been installed for the handling of the overburden.

It is a peculiar fact that loading into mine cars by hand in the open costs about the same per ton as coal recovered in excavation by steam shovel. At least such is the case in worked-over areas. There are no data on similar work in virgin areas, but the difference there would probably be slightly in favor of the shovel.

The objection to hand loading is, of course, its limited capacity, and its advantage is a cleaner separation of rock and slate from the coal.

In steam-shovel excavation of the coal, practice is about equally divided between contract and company work and circumstances to a large

extent determine the procedure. If the cover has been removed by a contractor it will usually mean a complete change of track gage and equipment for the company to take over the work, and if the cover stripping is still in operation, as is frequently the case, such a division of operations would be very undesirable. If possible, however, coal excavation in worked-over areas should be handled by the company and, if the output required will permit, the smallest size of shovel should be used. A cleaner product will be obtained with less waste in the dump and less handling of rock at the breaker. By the use of mine cars, also, a rehandling of the coal between the stripping and the breaker is often avoided with attendant breakage and rehandling cost. The rehandling cost is a considerable item and has equalled 7 c. a ton in certain cases. The minimum is not below 5 c. per ton.

Contracts

Written contracts between company and contractor are now nearly standard in essential form in spite of various local differences. It is, for instance, the universal practice for the company to furnish fuel and water though the mining of the one and the distribution of the other are generally required of the contractor. Where planes are required the company also furnishes, in most instances, the power to operate them. Practically all contracts contain certain standard clauses that aim to prevent waste of coal by the contractors. Enforcement of such clauses, however, varies.

The contractor in assuming the risk of profit or loss in the operation is supposed to assure himself so far as possible of the local conditions to be encountered, and no claim for extras is ordinarily allowed for unforeseen difficulties met with. On the other hand, no reduction in contract rates is asked by the company in operations where the natural conditions especially favor the contractor. As a rule, contracts are not made for each individual operation, but instead, cover all work done by the company over a period of years.

The usual form of contract provides for payment by the cubic yard, for all material excavated, at certain definite rates. Some contracts have been made, however, that provide that the contractor shall be paid by the ton, delivered at a fixed point to the company, a sum supposed to cover all costs of removing cover and mining coal. While such contracts are justified under certain conditions, they are not to be recommended. In their nature they are a gamble and their only virtue is that they sometimes secure the stripping of areas that would otherwise be left untouched.

Rates by the cubic yard are sometimes on a classified and sometimes on an unclassified basis. These terms have already been dis-

cussed. The unclassified rate is like the rate per ton of coal, somewhat of a gamble, and the classified rate is probably more satisfactory in the long run.

Contract Vs. Company Strippings

For actual stripping of the overburden from coal deposits, the nearly universal practice among coal companies is to let the work out on a contract basis. This has been the accepted practice since the earliest days of hand stripping. Indeed, this probably accounts partly for the fact that such is still the practice, as customs that once gain a foothold in conservative mining regions are likely to go unquestioned for long periods, and to spread from small beginnings until they become the dominating note in immense operations. The contract system for stripping operations, however, has had other arguments than this in its favor. Coal operators have always been reluctant to have their attention distracted from straight coal mining by the injection of interests foreign thereto. Moreover, the margin of profit on coal recovered by the stripping method is so small, at the best, that it is practically essential before undertaking such work to know, within a cent or two per ton, just what the final cost will be and such is accomplished when by the contract system all risks of fluctuation in such costs are assumed by the party of the other part. Another argument is the usual one in favor of contract work, that such work is pushed more vigorously than company work, due probably to the fact that there is more individuality to the management of a small contracting company than to that of a large mining corporation. This argument does not apply to what is known as the individual operator, but another and more potent argument in his case is the high first cost of stripping equipment and his inability to keep it steadily employed. Some, however, of the individual operators do maintain their own equipment and operate their own strippings, and many of these have succeeded in excavating overburden at a figure under the usual contract rate for the region. That the large companies could secure similar results is probably true, though to do so it would be necessary to devise some method of handling the work entirely outside of the regular organization of the company. The question is an uncertain one, however, and the solution of the problem of securing decreased stripping costs must probably be looked for elsewhere. It is becoming more and more essential to the larger operator in the anthracite regions to consider carefully means of meeting the increasing cost of producing coal, which would be helped by stripping the remaining available areas to greater limits in both area and depth than is economical under present practice. If this increasing demand cannot be met by the contractor, then the operator himself must assume the burden.

Possible Improvement in Practice

Past history of stripping operations, as recited briefly at the beginning of this paper, has shown that in spite of large increases in the wages of labor and in the cost of materials the cost of stripping has been reduced by improvement in methods, and the employment of adequate labor-saving equipment. It can be confidently expected that the future of anthracite stripping will repeat this history. Already at one operation in the region this is the case. The problem was met here by the installation, at a cost of over \$30,000 of one of the large drag-line excavators that have been so successfully operated in Middle Western coal strippings and in ship-canal excavations. Fig. 9 is a cross-section showing the measures stripped at this particular operation. Briefly, this machine is about 255 tons in weight and is of the revolving type. It is electrically operated through a complete control system which protects the motors from disaster due to overloading or sudden strain of any kind. The machine revolves in a complete circle with a radius of 125 ft.

Where the ground is at all regular this machine can be readily moved along when a cut is finished by casting the bucket to a point in advance and with the anchorage thus secured dragging the entire apparatus over wooden rollers. For uneven ground, or ground broken up by crop falls, the machine can be equipped as a shovel and results nearly as satisfactory secured.

Wherever it is possible to cast the excavated material to one side, rather than to load it into cars, labor is largely dispensed with. The labor item in ordinary strippings is probably 70 per cent. of the total cost of stripping, but in such an operation as described this percentage is reduced to about 50 per cent. From the results obtained to date it is believed that present stripping costs can be reduced by these advanced methods at least 10 c. per cubic yard, allowing amply for interest and depreciation items. The cost for power has been proven to be only 1 c. per yard and the first cost of the equipment is little greater than for an ordinary 70-ton shovel operation requiring a full complement of locomotives, dump cars, rails, etc. The principal item of cost is the moving of so large a machine from one stripping operation to another, or to and from the railroad. If the distance is great or if hills intervene, the machine must be taken down and carried piecemeal to its new location and there set up. This represents no insuperable difficulty, however, and it is probable that the number of these machines in the anthracite region will gradually increase. It is reported that in Kansas ratios of 10 cu. yd. of cover to 1 ton of coal have been handled profitably by means of this equipment. Many areas remain to be stripped in the anthracite region, for the end of stripping operations is not yet in sight, and if costs can be reduced to the extent outlined above there will be many

areas not now considered as stripping propositions, that will be the big operations of the future.

Various methods now employed in other stripping and excavating operations could perhaps be used to advantage in anthracite stripping

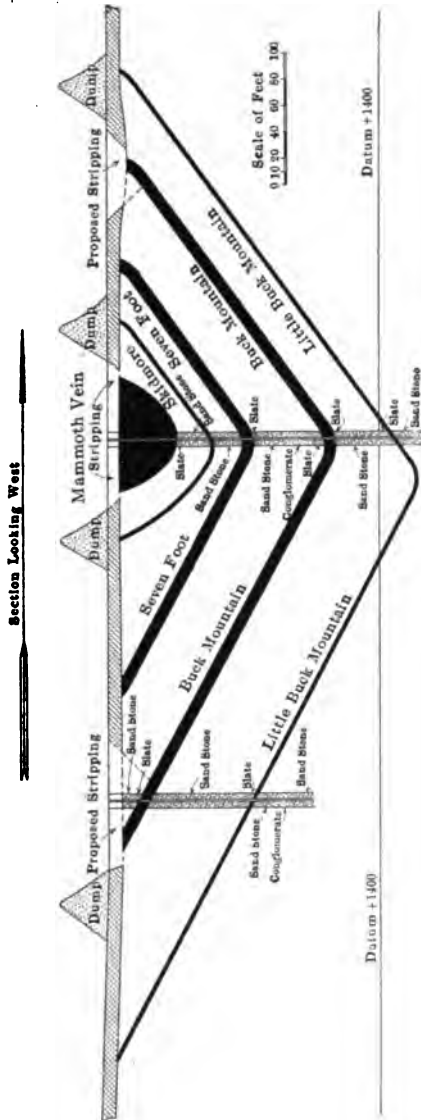


Fig. 9.

operations. Space, however, permits only the briefest mention of two of these.

Geared locomotives are practically unknown in the anthracite region.

There is a record of a stripping operation on an area of only 20 acres, where by the use of these locomotives a difference in elevation of 300 ft. between the bottom of the stripping and the top of the dump was overcome without passing outside the lines of the property.

Another possible improvement would be the use of hydraulicking methods. Scarcity of water and of areas on which to deposit and settle the refuse render this method in most sections of the region out of the question. Very low costs have been secured, however, in the West by this method and areas where it would prove feasible might be discovered. Where the refuse or spoil could be flushed into mine openings to support the surface a double value would be obtained.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Notes on the Heat Treatment of High-Speed Steel Tools

BY A. E. BELLIS,* S. B., SPRINGFIELD, MASS., AND T. W. HARDY,† S. B., NEW GLASGOW, N. S.

(New York Meeting, February, 1917)

THE problem of heat treating high-speed steel becomes more and more important as the design of cutters becomes more and more complicated in increasing the efficiency of mechanical operations. Hundreds of dollars are spent in the design and manufacture of milling cutters of special form for rapid production of duplicate or interchangeable parts, and then, as the heat treatment is faulty on the one hand, or scientifically executed on the other, the tool fails after a few operations, or its efficiency is greatly increased. The practical operation of giving these complicated tools the right temperature necessary to bring out the best cutting qualities, and at the same time bring the tool out "clean," is a difficult one and calls for no small amount of skill. In order to be on the safe side the average tool hardener uses a temperature much too low to give the best results with the high-speed steel he uses. In the case of cutters which are finished to a given diameter before hardening, it is impossible to grind the tool after hardening, so that it is essential that the surface be protected from oxidizing or de-carbonizing. It is the aim of this paper to describe some experiments on hardening high-speed steel, in which metallographic means were used to determine the correct hardening temperatures.

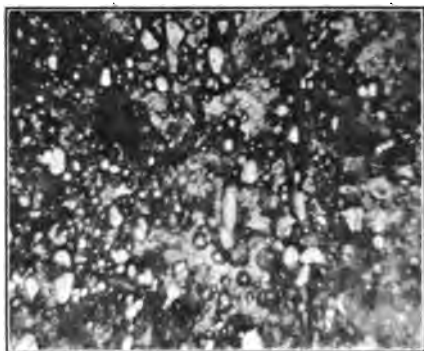
The accompanying table and plates show the results of hardening experiments on five different high-speed steels. The analyses for carbon, tungsten, chromium and vanadium of each steel are given in the accompanying table. Six specimens from the same bar of each kind of steel

Analyses of High-Speed Steels Used in Hardening Tests

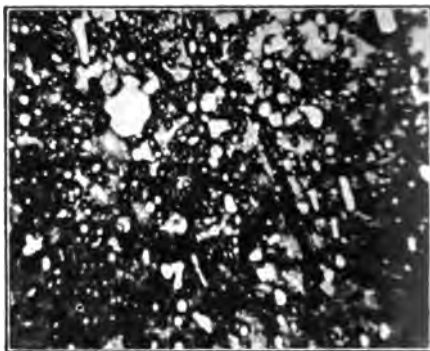
	C, Per Cent.	W, Per Cent.	Cr, Per Cent.	Va, Per Cent.
A	0.58	17.4	3.11	1.14
B	0.60	13.3	3.32	3.58
C	0.53	13.0	4.69	2.45
D	0.75	17.7	3.30	0.85
E	0.60	16.5	3.55	0.70

* Metallurgist, New England Westinghouse Co.

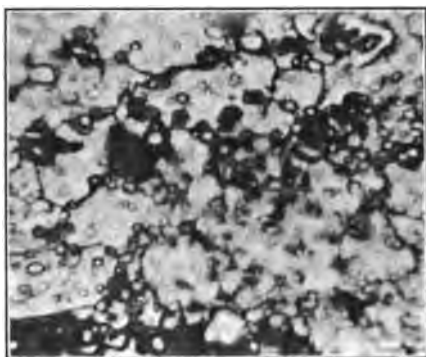
† Metallurgist, Nova Scotia Steel & Coal Co.



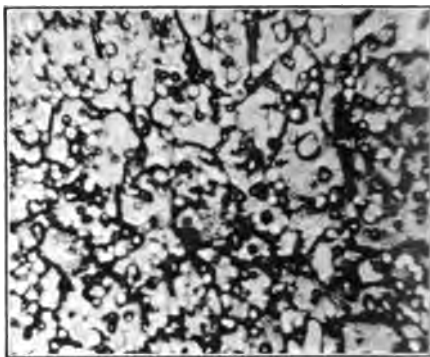
2,050° F.



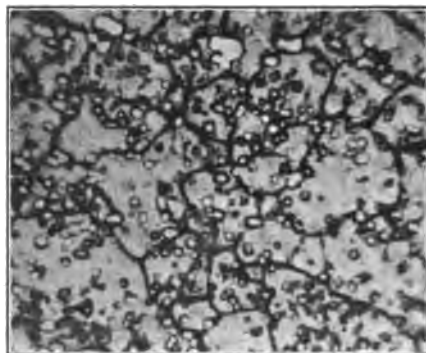
2,100° F.



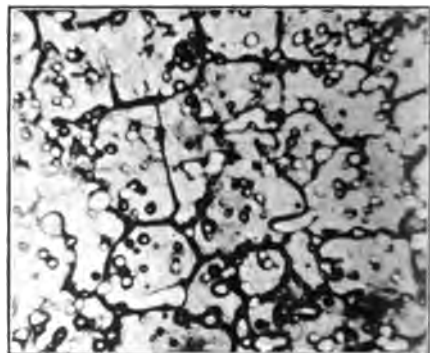
2,200° F.



2,250° F.

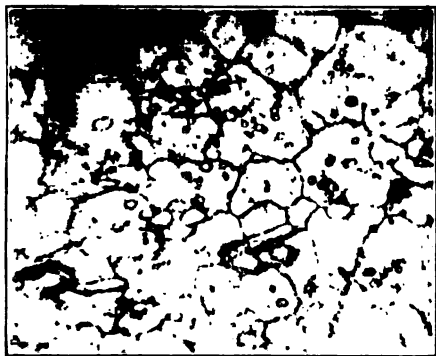


2,300° F.

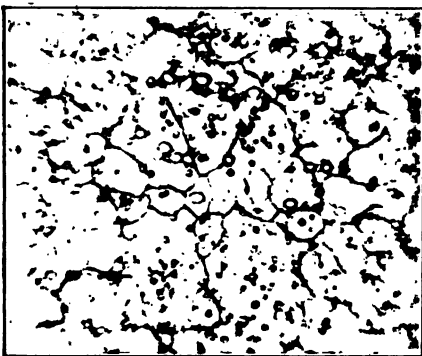


2,350° F.

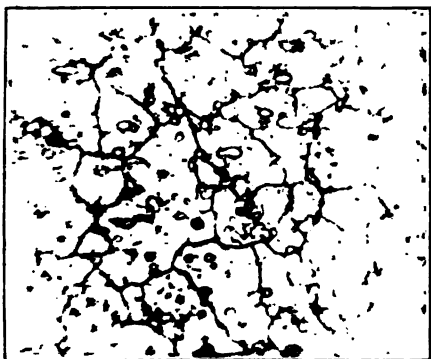
PLATE A.—STEEL A HARDENED AT DIFFERENT TEMPERATURES.



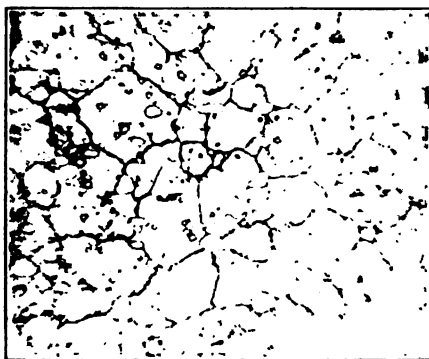
2,100° F.



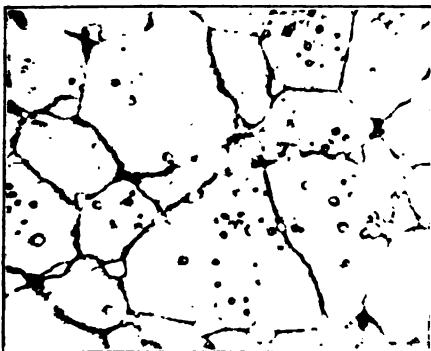
2,150° F.



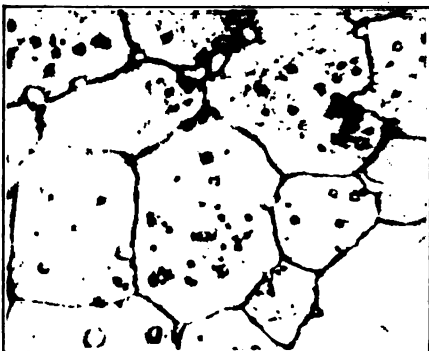
2,200° F.



2,250° F.

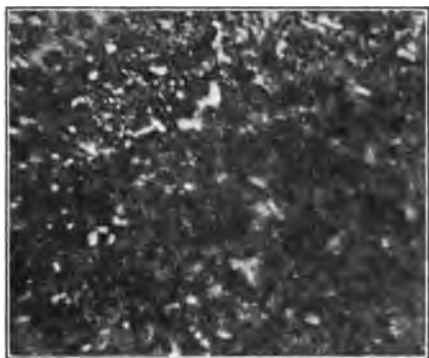


2,300° F.

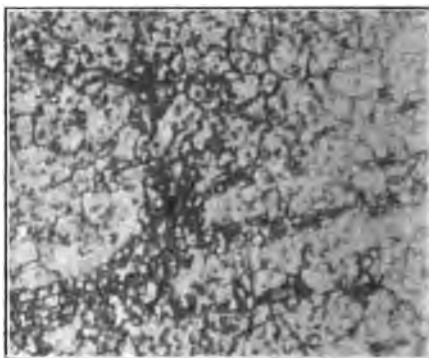


2,350° F.

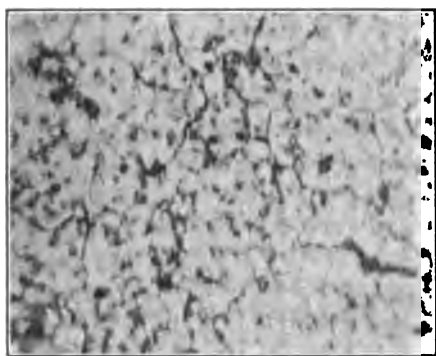
PLATE B.—STEEL B HARDENED AT DIFFERENT TEMPERATURES.



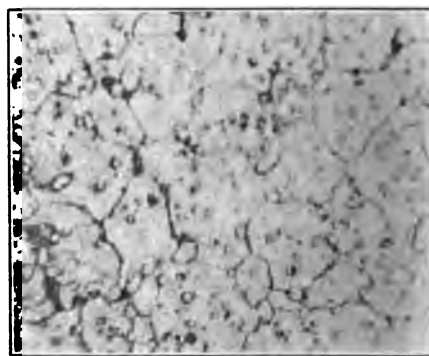
2,000° F.



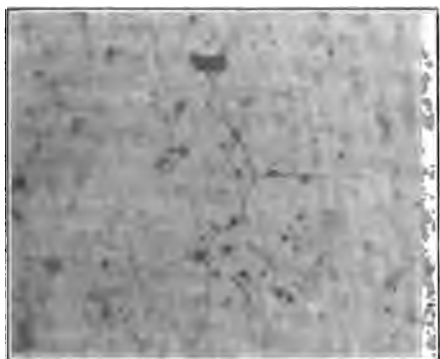
2,100° F.



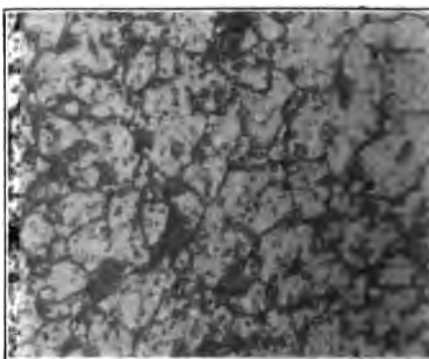
2,150° F.



2,200° F.



2,250° F.



2,300° F.

PLATE C.—STEEL C HARDENED AT DIFFERENT TEMPERATURES.

were hardened from different temperatures, and photomicrographs made. The temperature from which the piece was hardened is given under the photomicrograph in each case. Thus the illustration marked A-2,100 is steel giving analysis under A, and hardened from 2,100° F. The six samples were taken from the same bar in the annealed condition regularly furnished the tool maker. Photographs were made of the longitudinal section, care being taken to grind off the outer surface. The specimens, $\frac{1}{4}$ -in. square in section, were first preheated at 1,500° F., and then quickly placed in the high-speed furnace already heated to the desired temperature, left at this temperature for 1 min. and quenched in oil. The temperature was controlled by a standard pyrometer, consisting of a rare-metal couple and potentiometer. An optical pyrometer of the Holborn-Kurlbaum type gave excellent checks with the standard pyrometer, and proved more convenient and durable. A precision of 10° F. was attained throughout. The polished specimens were etched for 15 min. in 4 per cent. alcoholic solution of nitric acid, and photographed under a magnification of 1,000 diameters (750 diameters as reproduced).

In interpreting the micrographs, it is convenient to regard A-2,100 as a dark matrix showing a large excess of a free carbide; as the temperature is raised, more and more of the carbide dissolves, and the network structure of austenite is more noticeable (A-2,200). The overheated structure shows a coarse grain with black (burnt?) spots and wide intergranular spaces. We have refrained from drawing definite conclusions regarding variations of structure with analysis and hardening temperatures.

In general, the steels that show some excess carbide even at the maximum hardening heat are the most efficient. These, as a rule, are the steels with high tungsten content; they harden from a higher temperature and over a wider range than the lower-tungsten steels. For this reason they do not require as careful treatment and are, therefore, more popular than the lower-tungsten steels. The steels with lower tungsten and higher vanadium give better results when hardened at the lower temperatures than do the higher-tungsten steels when these are hardened at the same low temperatures, but the comparison is not so advantageous to the lower-tungsten steels when the steel with higher tungsten is given the proper hardening heat.

The importance of carefully controlling the hardening temperature, and of varying it for the particular steel used, cannot be overemphasized. The custom of using one "high-speed temperature" for all tools is very poor practice, for, as shown by the photomicrographs, the best structure may be obtained with one steel at a temperature which will "burn" another, or not harden a third. Thus 2,300° F. or over is necessary to give A or E a good structure, but this temperature gives a coarse grain in the other steels, or "burns" them. Again, a temperature as low as 2,150° F. can satisfactorily harden B or D, but tools made of other steels

would not stand up if hardened at this temperature. More extreme examples could have been shown, but the samples chosen are typical of the most widely used brands of high-speed steel.

The average hardener rarely obtains the best result from his steel. The reason for this is, especially in the cases of tools that cannot be ground after hardening, that oxidation becomes a serious problem at higher temperatures. The use of a barium chloride bath to eliminate this difficulty has the disadvantage that the surface of the tool becomes decarbonized. A method that has proven satisfactory is to place the tools after preheating in the reducing atmosphere of a carbon resistance electric furnace already heated to the required temperature. The very short time necessary to get the tool to the temperature of the furnace eliminates deleterious surface effects. Pack hardening often gives good results but, owing to the great affinity of iron for carbon at high temperatures, care must be taken to reduce this carbonizing action to a minimum. This may be done by selecting a packing material of little or no carbonizing power, and by cutting down the time during which the metal is in contact with the packing material.

The increased efficiency and cutting power of tools that have received the proper heat treatment is out of all proportion to the time given to the study of the particular steel involved, and to the care given the work.

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The Manufacture of Weldless Steel Tires for Locomotive and Car Wheels

BY GUILLIAEM AERTSEN,* PHILADELPHIA, PA.

(New York Meeting, February, 1917)

THE derivation of the word tire (or tyre, as it is spelled in England) is obscure. Some dictionaries suggest that it is the aphetic form for "attire, covering," so called as being the outside covering or band of a wheel; also as a corruption of "tier," the band which holds, or "ties," the rest of the wheel together. One dictionary states: "The supposed derivation of the word is that bands were first used on wheels in the City of Tyre, Syria."

In the infancy of railroads, the tracks consisted of flat bars of iron, spiked or bolted to the upper surfaces of longitudinal timbers, which, in turn, rested upon masonry foundations. Stone blocks installed for this purpose on the old Germantown & Norristown Railroad, now the Chestnut Hill branch of the Philadelphia & Reading Railway, were to be seen for many years on the right-of-way, north of what is now Wayne Junction, Philadelphia, Pa. The "cornerstone" of the Baltimore & Ohio Railroad was laid Apr. 4, 1828, and the first locomotive in America was built by Peter Cooper at the St. Clair Works, Baltimore, Md., in 1829 "to demonstrate its adaptability to a curved road." It is believed that the first successful locomotive was built in Philadelphia by Tyler & Baldwin in 1832 for the Germantown & Norristown Railroad. Rufus Tyler and Matthias Baldwin were brothers-in-law; the latter's name will be identified with American locomotives until locomotives are forgotten.

The wheels on these first locomotives, designed to run on this primitive rail, were shod with plain iron bands, bent and welded. The flanges consisted of separate iron bands, bent sidewise, welded, and bolted to the sides of the wooden felloes, or rims.

Before many years had elapsed the tires were fabricated of iron bands, rolled in one piece with the flange projection on the side. The tire for each wheel was bent and welded and subsequently attached to the wheel center by shrinkage, bolts, rivets and various mechanical devices.

Germany is generally given the credit for first making such tires of

* Assistant to Vice-President, The Midvale Steel Co.

steel without a weld, the ingot being cast solid, flattened, punched and subsequently spun, or rolled, on rolling mills designed for the purpose. Similar, or perhaps the same, mills had previously been used for "rounding up" the bent and welded iron tires. There is a tradition that weldless steel tires were made in the same manner in England about the time that their manufacture was being attempted in Germany, but the writer has been unable to discover any trustworthy record of this practice in England at so early a date.

As early as 1853 the German process was exploited, and before 1860 weldless steel tires were being turned out in considerable quantities there and in England. The popularity of these tires seems to have been instantaneous, their success being due not only to superior wearing qualities and accuracy of finish, but also to increase in safety.

Steel for tires was originally made in the crucible exclusively and the principal German maker had been engaged in their manufacture for many years, sending them to all parts of the world, before he ventured to introduce tires made from open-hearth steel. The ordinary practice was to cast individual ingots, poured from the top into cast-iron molds of great variety of shapes. The popular mold was pear-shaped, or the frustum of a cone with the small end up; this for the purpose of reducing the area affected by shrinkage. The ingots were subsequently flattened under a steam hammer, a small hole punched in the resultant slab, or disk, and this hole enlarged by hanging the punched disk on the horn, or beak, of the anvil, or lower die, on which it was rotated while the hammer struck successive blows on the outer edge. During this operation, the flange, where one was required, was roughly shaped. After this operation, the "beaked bloom," as it was called in England, was rolled to required dimensions in the tire mill.

Some of these early tire mills seem to have been horizontal two-high bar mills altered by extending the rolls through the housing farthest from the engine and hanging the tire on this extension. Many manufacturers followed this idea in designing new tire mills, the rolls being horizontal and the tire in the air while rolling.

It is interesting to note that even in these early days, tire manufacturers realized the improvement in quality that would be secured were it possible to cast material for tires in long ingots, cutting each into several blocks, one for each tire, and discarding the upper portion. As early as 1864, an English patent covering this was taken out. Each ingot was to be cut into four or five blocks by means of circular saws. A subsequent patent avoided the loss in sawdust by using, instead of saws, circular shears, or disks, the ingot being previously heated. These revolving disks were carried upon a shaft which, in turn, was supported at either end in eccentric bushings revolved by spur gearing so as to feed the shaft and its disks toward the center of the ingot until the cut was

completed, when they were lifted to clear it. The hot ingot was supported by two corrugated rollers, driven, thus causing it to revolve.

The shaft bearing the disks could also be driven either by separate gears or by the friction between the disks and the revolving ingot.

The records do not show how successful this process was, but it seems to embody several ideas perfected by the next generation.

To avoid scrap in punching, a method was proposed in which a sharp-pointed instead of a flat-pointed punch was used. Ingots were top-cast in conical molds, small end up, the neck being closed after pouring by a plate clamped or cotted down upon it. Dead-melted steel did not seem to be expected in tire ingots.

The pioneers appear to have had great difficulty, at first, in punching, in the slab or upset ingot, a hole sufficiently large to admit the horn of the beaking anvil. One of the early methods was to drive a cutter, or wedge, about $\frac{3}{4}$ in. thick through the slab and afterward stretch and enlarge this slit by the introduction of tapered punches or mandrels. After this difficulty was overcome and the practicability of punches of circular cross-section proved, it seems to have been good practice if the four operations of upsetting, punching, beaking and rolling could be accomplished in four heats.

In the United States, prior to 1867, there were one or two mills designed for rolling, or rounding-up, iron tires. The manufacture of weldless steel tires had not yet been attempted. In that year, a member of an English family which had been long identified with the manufacture of cutlery, was induced by some Philadelphia capitalists to come to that city and establish a tire plant. From that beginning, 50 years ago, have sprung, directly or indirectly, all the other tire works in America.

At first, crucible steel was employed exclusively. The open-hearth furnace was at that time not sufficiently perfected to justify its use for such purpose, although but a few years elapsed before it practically supplanted the crucible for this specialty.

At first, ingots of circular cross-section were poured from the top in cast-iron open molds, one ingot for each tire. To insure accurate weights, pouring was stopped when the level of the fluid metal reached a chalk mark on the inside of the mold. Subsequently these molds were improved by the introduction of a cast-iron stopper, or heavy disk hung on the inside at the required height. This stopper, having a vent at its center to permit the air to escape, was suspended from a spider attached to the mold. After some years, top-casting was abandoned and all ingots were poured from the bottom. Still later, the practice of casting individual ingots, one for each tire, was replaced by the practice of casting long ingots, each of which was sliced cold into as many blocks as necessary and the top portion, enough to eliminate all danger from segregation and pipe, discarded. The advantage of this practice is self-

evident. The slicing operation is stopped before the center of the ingot is reached, the remaining core being broken, so that the fracture at the center of the ingot can be examined.

In those early days, furnaces for heating high-carbon steel, as tire metal was then considered, were crude, and the circular cross-section of the ingots added to the difficulty. Consequently, it was thought necessary to give the ingots a preliminary, or "saddening" heat. They were charged into a comparatively cold furnace, brought slowly to a dull red heat, transferred to another hotter furnace and brought to the temperature necessary for punching. After this operation they were allowed to become cold; reheated and beaked; again allowed to become cold; reheated and rolled.

In a tire works with modern equipment, the sliced block is very slowly heated in a continuous furnace, upset, punched, beaked (when necessary) and rolled, without being allowed to become cold at any intermediate stage. The various operations are performed at a press, or hammers, and two or three mills, the partly finished bloom being transferred from one to the other with no appreciable loss of heat during the transfer. The results of this method are eminently satisfactory, the metal being continuously worked from the high forging heat to the proper temperature at which it should be laid down to cool.

Rolling tires differs from other rolling operations in one important particular not always appreciated by those unfamiliar with the details of the operation. In rail, structural or bar mills the billet is led from one pass to another, each designed to deliver the cross-section best suited for that following. Thus, in well-designed rolls there should be no risk of tearing or pulling the metal or setting up strains in one part of the section because its reduction has been less rapid than that of the adjoining section. On the other hand, in a tire, or other ring (except when it is possible to transfer from one mill to another), it is almost always necessary to accomplish the reduction in the same pass, without removing the tire therefrom, the reduction in the section being accomplished by forcing the rolls closer together while the metal is still between them. Any one familiar with ordinary rolling-mill practice will appreciate the difficulties this involves, and these difficulties are aggravated by the fact that in tires we are dealing with the cubical contents of the entire ring, not with the area of its cross-section alone.

The old tire mills were designed for tires of comparatively small diameter and thin sections and usually consisted of only four rolls, two of which might be called working rolls and two, guide rolls. One of these working rolls, the main roll, contained a single box pass, turned to fit the contour of the outside of the tire, the contour including the tread and flange with the front and back faces contiguous thereto. The other working roll, the pressure roll, was turned to coincide with the bore of the tire,

being of exact width to enter the box pass in the main roll. One of these working rolls, usually the main roll, was rotated, or driven, by the engine, its bearings remaining stationary; the other working roll, usually the pressure roll, was rotated by friction between its periphery and the inside of the bloom as the latter rotated by friction with the main roll. The bearings supporting the pressure roll were carried on a slide, which, operated by hydraulic or screw power, advanced as required toward the main roll, thus reducing the opening between the two and the cross-section of the bloom or tire being rolled. The two guide rolls merely served to preserve the circularity of the tire by bearing against two points on its circumference equidistant from the main roll. They were driven only by friction between the tire and themselves and were carried back and forth on slides or carriages operated by screw or hydraulic power.

The more modern mills have four guide rolls touching the tire at four instead of at two points; have a pressure roll similar to that described above except that its face is longer than the width of the bloom or tire being rolled; have a main roll containing a very shallow box pass, not covering or overlapping the front and back faces of the tire, and, in addition, have two horizontal rolls, one driven by power, the other driven by friction, the latter being raised and lowered by screw or hydraulic power. With this universal type of mill, pressure is brought upon the section of the tire from all four sides, with beneficial results. These modern mills are, as may be imagined, more substantial and powerful than their predecessors. As a result the quality as well as the quantity of work turned out shows much improvement.

Many changes, great though gradual, have occurred in the demands made by consumers upon tire manufacturers since the birth of the industry, and, as might be expected, most of these are for the better. The standardization of sections and diameters insures more satisfactory deliveries and eliminates many causes of error. The "mileage guarantee," usual in the early days, is now seldom demanded and still more seldom granted. Railroad-operating men appreciate that there are important factors affecting the hardships inflicted upon locomotive driving tires which can seldom be calculated, more seldom foreseen, and never observed or recorded. Safety can be demanded and secured. Wearing qualities would better be considered among the general results. Locomotive driving tires have sometimes shown abnormal wearing qualities, being apparently indestructible, and yet the efficiency of the locomotive shod with them has been reduced to a minimum by loss of tractive power.

One of the most successful manufacturers in Europe in the early days guaranteed that all the tires sent out from his establishment would do a certain amount of work before any repairs were placed upon them.

This mileage guarantee, based upon the weight of the tire, he reduced to a formula in which W represented in pounds the weight of the tire and M the mileage it was guaranteed to run before turning. The formula was:

$$M = \frac{W \times 248.55}{2.2048}$$

which simply meant that for every pound of metal in the tire it would run 113 miles. This probably worked well in practice in those days, when the effect of brake action was nil, the loads were light, and practically the only difference in tires was their diameter, their section and the service to which they were subjected being the same. The heavier tire, larger in diameter though equal in thickness, would naturally outwear its smaller competitor, their lives being directly proportional to their weights.

In few specialties of the steel industry have producers and consumers worked so harmoniously in endeavoring to improve the quality of the product, and a comparison, could one be made, between tires manufactured in 1870 and those turned out today, would prove that this coöperative labor has not been in vain.

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Reservoir Gas and Oil in the Vicinity of Cleveland, Ohio

BY FRANK R. VAN HORN, *CLEVELAND, OHIO

(New York Meeting, February, 1917)

It is customary to ascribe two general modes of occurrence to natural gas, namely shale gas which, as the name indicates, is found in shale, and reservoir gas, which occurs in sandstone, conglomerate and dolomitic limestone. Both of these types are found in the Cleveland district. The shale gas has been produced on a small scale for a long period of years, probably dating back as far as 1883. Shale gas is a low-pressure gas, small in volume with variable pressure, and is limited to no definite horizon. It is generally independent of oil, may last for a long time, and is not dependent to any great extent on the structure of the rocks. There are many such wells in the district which range from 400 to 1,840 ft. in depth, although most have been completed at about 800 ft. This indicates that the wells pass through the Cleveland and Chagrin shales into the Huron or Portage shale of the Upper Devonian.

During the last 2 years great interest has developed in a deeper-seated reservoir gas.

What is commonly called reservoir gas is generally a high-pressure gas occurring in large volume at a definite horizon. Oil may, or may not, be associated with the gas, and in most regions in which reservoir gas is found, the geological structure is of great importance. This type of gas occurs in all large fields such as those of West Virginia which have been supplying Cleveland until the recent discovery of local gas at great depths in commercial amounts.

HISTORICAL OUTLINE

The oldest well producing reservoir gas in the district was drilled in 1886 in Newburg for what was then known as the Cleveland Rolling Mill Co. This well was 3,000 ft. deep, and was thought to have reached the "Clinton" formation. Two "sands" showing small amounts of oil were recorded in the Niagara limestone at depths of 2,658 and 2,686 ft. This is about the horizon of the so-called Newburg sand which has proved to be of some importance as a source of gas and oil by recent drilling operations. The rock pressure was reported to be 400 to 500 lb. per square inch, but since the volume was only 14,000 to 16,000 cu. ft. daily, the well was not only considered unsuccessful but was thought to prove the absence of high-pressure gas in commercial quantities in the Cleveland district.

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The discovery of gas and oil in Clinton rocks in central Ohio between 1900 and 1907 proved an incentive to the continuation of prospecting northward, and as a result, several thousand acres west of Kamm's Corners and Berea were leased for gas and oil rights between 1905 and 1907 by the East Ohio Gas Co., and the Logan Gas and Fuel Co. The former company did most of the drilling, and was still at work in 1908. It also purchased the site of the Newburg Salt Co. in Mill Creek Valley, and in 1907 and 1908 deepened the old salt wells to the Clinton horizon. In most places the wells were unsuccessful, and most of them were not even capped. One of these wells was fairly successful, however, and furnished the second example of deep-seated reservoir gas in the district. The well was drilled by the East Ohio Gas Co. at North Ridgeville, Lorain County, and was finished in June, 1908. It was sold to A. L. Mills, who furnished the driller's record which has been interpreted as shown in Table 1.

TABLE 1.—*Partial Record of Well Drilled in June, 1908, at North Ridgeville, Ohio*

	Thickness, Feet	Depth, Feet
Drift.....	18	18
Bedford and Ohio shale.....	1,032	1,050
Devonian and Cayugan limestone (water in limestone at 1,300 ft.).....	563	1,595
Salina.....	80	1,675
Niagara limestone (show of oil at 2,200 ft.; water in limestone at 2,250 ft.).....	825	2,500
Cataract sand "Clinton".....	8	2,508
Cataract and Queenston.....	44	2,552
Richmond.....	47	2,599

At the time of casing, the rock pressure was 840 lb. per square inch, and the open flow of gas was reported to be 250,000 cu. ft. a day. The supply from this well was used to light and heat several houses. The "show of oil" at 2,200 ft. is undoubtedly the horizon of the so-called Newburg sand of present-day operations.

The next development in deep drilling for gas in this district, as far as the writer knows, was on Oct. 17, 1911, when the Newburg Brick and Clay Co., near Warner and Canal Roads, South Newburg, "brought in" a gas and oil well at a depth of 2,520 ft. It was found in a "sand" between limestones of Niagara age, and this horizon has since been called the Newburg sand. The well was still furnishing both gas and oil in December, 1915.

Present Developments

The oldest wells of the present boom were finished to the "Clinton" sand at a depth of 2,740 ft., in February, 1912, by the National Carbon Co. and the Winton Motor Carriage Co., near Highland Avenue, Berea Road and the New York Central Railroad. Both wells furnished about 1,000,000 cu. ft. of gas daily, and the pressure was about 1,100 lb. per square inch. There was but little drilling during 1912, or until well along in 1913 when several good wells were drilled in Lakewood. Early in 1914, several other successful wells were drilled in Lakewood and West Park, but no actual excitement or boom started until Jan. 30, 1914. On this date, at



FIG. 1.—VIEW ON LORAIN STREET, CLEVELAND, SHOWING SIX DERRICKS. THIS GIVES SOME IDEA OF THE CONGESTED DRILLING DURING THE GAS EXCITEMENT OF 1914-1915.

the plant of the J. L. and H. Stadler Rendering & Fertilizer Co., South Brooklyn, gas was found at about 2,400 ft. in the Newburg sand. The initial flow of the well was about 12,000,000 cu. ft., and the rock pressure was 950 lb. per square inch. The production dropped to about 3,000,000 cu. ft. within 6 months, and when the service from this well was discontinued in August, 1915, the pressure was only about 100 lb. per square inch. In April, 1914, there were already 55 producing wells, 10 of which were in the Newburg, and the remainder in the Clinton sand. Drillers came from all parts of the country, and many people inside the western limits of the City of Cleveland, as well as in Lakewood and West Park, insisted on having private wells in their own back yards. As a result of this situation, drill holes were placed too near each other, and the production of the older wells rapidly decreased (see Figs. 1 and 2). There did not seem to be very much decrease in the initial pressure and flow of the

newer wells when compared with the earlier ones, but there was a notable decrease in the life of the newer wells. In June, 1916, over 1,000 wells had been drilled, about 900 of which were drilled between January, 1914, and January, 1916, so that only a little over 100 wells were drilled during the first 6 months of 1916. This indicates a very decided curtailment of drilling operations, which is better understood when the statement is made that nearly 50 per cent. of all wells drilled between January and June, 1916, were unsuccessful. The average cost of a well in the district is about \$5,500, so that the total amount invested in drilling alone has



FIG. 2.—VIEW ON LORAIN STREET, CLEVELAND, SHOWING FOUR DERRICKS, CLOSELY SPACED.

been over \$5,500,000, to say nothing of the cost of land, leases, or other expenses. It is fairly certain that the value of the gas produced has returned but a small percentage of the original capital invested, much less any interest on it. Only the drillers and a few others have profited by the discovery. Of the total number of wells drilled, about 230 holes were dry or yielded so little that they were immediately abandoned. Of these, 34 were in the Newburg, and the remainder in the Clinton sand. About 150 of the successful wells had been abandoned before May, 1915, at which time many others were producing very little. The average life of the wells of the district is said by an official of the East Ohio Gas Co. to be about 8 months.

The record well of the Cleveland area is said to be that finished by M. F. Bramley and others, on Feb. 22, 1915. It is located on the Aulenbach property, on Harrington Road near Puritas Springs Road. The Clinton sand was reached at a depth of 2,712 ft., and the initial flow was said to be

nearly 14,000,000 cu. ft. daily. Two other wells produced 13,000,000 each, and the Stadler well had an initial flow of 12,000,000 cu. ft. a day. Six other wells are said to have started with a flow of 10,000,000 cu. ft. each. Over 50 wells are reported to have had an initial flow greater than 3,000,000 to 4,000,000 cu. ft. daily, but on the other hand, many wells have not produced 1,000,000 cu. ft. a day, and the number of dry holes has averaged more than one out of every five producing wells. No accurate figures could be obtained about the total production of the Cleveland field, but it was reported that late in 1914 the district furnished close to 100,000,000 cu. ft. daily, which is about the amount required by the City of Cleveland and its suburbs. The increase in production in Ohio during 1914, over the previous year, was 17,657,974,000 cu. ft. This averages 48,275,271 cu. ft. daily and the increase was due largely, if not wholly, to the production from the Cleveland field. It is very probable that the amount produced by December, 1915, was hardly one-quarter of the maximum production. Practically all the gas is purchased by the East Ohio Gas Co., although some manufacturing plants have supplied their own gas. Since January, 1915, the center of drilling operations has gradually moved from West Park and Kamm's Corners to the southeast, toward Brook Park, Berea, and Middleburg township where each well is protected by considerable acreage. This should result in a longer life for each well than was experienced in the northern part of the district where the overdrilling was so pronounced.

PRODUCING SANDS

The chief sands of economic importance have been called the Newburg and Clinton. Some people have applied the name "Stadler Sand" to the Newburg, but the latter term has priority as well as more common usage. There is also one unimportant horizon called the "Stray" sand, and the Trenton limestone has been prospected to a slight extent.

Stray Sand

This sand has been found in at least three places in Lakewood, at depths ranging from 1,355 to 1,400 ft., and its position must, therefore, be in the lower Devonian or upper Silurian limestones. The sand is about 3 ft. thick and at two places produced oil, while at the other locality an initial flow of 4,000,000 cu. ft. of gas was obtained. Water caused more or less trouble, and the wells were abandoned after a time, or were deepened to the Clinton.

Newburg Sand

The horizon of the Newburg sand is found at depths ranging from 2,300 to 2,600 ft. The Newburg is said to attain a thickness of 30 ft. in places, but in other localities it is apparently absent. In the Lakewood

region, its thickness is rarely greater than 15 ft., but it becomes thicker to the south and east toward the Denison-Harvard district, where the maximum thickness is found. In that vicinity its depth below the surface also increases, owing to the southeast dip. According to reports made by the East Ohio Gas Co., the Newburg varies from 3 to 17 ft. in thickness. At the pioneer well of the Newburg Brick & Clay Co., the sand was penetrated to a depth of 15 ft., reaching a total depth of 2,520 ft. The gas sand from this well was of grayish color, soft, brittle, and showed abundant cleavage planes. In polarized light even the smallest fragments showed high interference colors which characterize minerals of very high double refraction. This sand dissolved almost wholly in cold dilute hydrochloric acid, and the solution showed few, if any, traces of iron. The properties given previously are those of a fairly pure limestone, composed of calcite or dolomite. Another specimen of Newburg sand was of grayish-red color, and contained particles which were grayish, reddish-brown, and grayish-black, so that it might have been a mixture of three rocks. This sand was easily crushed but some of the particles were hard enough to scratch glass. The grayish particles showed high double refraction, and dissolved readily in cold dilute hydrochloric acid. The reddish-brown particles dissolved less readily and the other particles were insoluble. The solution was colored yellow with iron chloride. This sand was calcareous, but certainly originated from a more impure limestone than that obtained from the Newburg Brick & Clay Co.'s well. The Newburg sand is, therefore, a calcareous or dolomitic limestone, more or less pure, and not a quartzose sandstone like the Clinton horizon. It probably consists of a porous limestone similar to the Trenton limestone of the Findlay-Fostoria region. The Newburg sand occurs in what is called the Big Lime, and belongs to the Niagara epoch of the Silurian. The horizon probably corresponds to that of the Lockport dolomite. About 124 wells have been drilled to this sand, and only 34 have failed to find gas. Many of the latter were continued down to the Clinton. Chemically, the gas from the Newburg sand is said to be little, if any, different from the Clinton gas, an analysis of which is given in Table 3. The Newburg has been most successfully prospected in the southern and southwestern parts of Cleveland, known as Newburg and Brooklyn, especially in what is called the Denison-Harvard district, where the sand is thickest. The three largest wells came in with 13,000,000, 12,000,000 and 6,000,000 cu. ft. of gas daily. One of these was the Stadler well, located at Denison Avenue and the Belt Line, which really started the gas boom in the Cleveland territory. Although this well came in with a large volume of gas, it lasted only from Jan. 30, 1914, to about August, 1915. Because of the general interest in this well, its log, as furnished by the drillers, with interpretation, is given in Table 2.

TABLE 2.—*Log of Stadler Well, Denison Avenue and Belt Line, Cleveland. In Newburg Sand. Completed Jan. 30, 1914. Volume 12,000,000 Cu. Ft. Rock Pressure 950 Lb. per Square Inch*

	Feet		Thickness, Feet	Depth, Feet
Drive pipe.....	70	Drift.....	70	70
Shale.....	70-1,106	Devonian shale....	1,036	1,106
Top of lime.....	1,106	Devonian and Cay- ugan limestone....	689	1,725
Lime very hard.....	1,360			
Water (filled to top of hole)	1,395			
Gas.....	1,415			
Through lime.....	1,555			
Broken lime.....	1,555-1,725	Salina.....	390	2,115
Salt.....	1,725-1,915			
Lime.....	1,915-2,010			
Salt.....	2,010-2,025			
Slate.....	2,025-2,070			
Lime shell.....	2,070-2,080	Niagara above Newburg.....	240	2,355
Slate.....	2,080-2,115			
Lime.....	2,115-2,250			
Lime very hard.....	2,250-2,260			
Lime, broken.....	2,260-2,320	Newburg sand.....	20	2,375
Lime, broken.....	2,320-2,355			
Gas.....	2,365			
Total depth.....	2,375			

Clinton Sand

Although through long usage the sand at this horizon has been known as the Clinton sand, it is very certain that it does not belong to the Clinton, but to the Cataract formation of the upper Medina, and, like the Newburg, is also a Silurian horizon. On account of the common use of the term Clinton, it will probably be best retained. The sand is found at depths ranging from 2,700 to 2,900 ft. In the northern part of the Cleveland district it is reached quite constantly at about 2,750, but toward the south and east the depth increases, owing to the dip of the rocks in those directions. According to reports, the thickness ranges from 8 to 50 ft. The East Ohio Gas Co. gives results varying from 5 to 35 ft. At the plant of the Cuyahoga Brick & Shale Co. in Parma township, between Ridge and Pearl Roads, the top and bottom of the Clinton were reported at 2,823 and 2,867 ft. respectively, giving a thickness of 44 ft. The Twin City Oil and Gas Co. at West 100th Street near Bertha Avenue, drilled from 2,737 to 2,765 in the Clinton, a distance of 28 ft., but reported that "The Clinton here is 50 ft. deep, which is about double the normal thickness." The thickness most commonly given is about 20 ft. J. C. Gillette, of the National Carbon Co., says that one of the reasons for the

varying reports is that some drillers include in the Clinton certain rocks which do not belong there. In his experience he finds, on approaching the Clinton, first a layer of reddish, clayey sand, ranging from 0 to 12 ft. in thickness, which contains some gas. This is followed by 3 ft. of shale, and then comes about 20 ft. of gray, faint pink, or whitish sand, which is the chief gas reservoir. If these three strata were all included as Clinton, it would give a thickness approaching 35 ft.

Two specimens of pink, and gray or white Clinton sands have been investigated. They consist of quartz sand stained more or less with iron oxide, which even in the white sand dissolves in cold dilute hydrochloric acid, and stains the solution yellow with ferric chloride. Neither sample showed any traces of lime, and the white sand originated from a grayish, porous, friable, quartzose sandstone, particles of which, hard enough to scratch glass, could be seen in the sand. Two other specimens of grayish color were also examined, but these contained limestone fragments along with the sandstone and, consequently, effervesced easily with acid. The limestone particles undoubtedly came from horizons above. Although the Clinton sandstone is generally porous, it is sometimes reported as very dense and hard, so as to require shooting in order to open up the flow of gas. Two analyses were furnished by the East Ohio Gas Co. Under 1, is given an analysis of Clinton gas from Cleveland, while under 2, is another of West Virginia gas, used in Cleveland, for comparison.

TABLE 3.—*Analyses of Natural Gas Furnished by the East Ohio Gas Co.*

	1 Cleveland	2 West Virginia
Carbon dioxide.....	0.0	0.20
Illuminant (heavy hydrocarbons).....	0.2	0.30
Oxygen.....	0.1	0.15
Carbon monoxide.....	0.6	0.50
Methane.....	95.5	93.60
Hydrogen.....	1.6	1.65
Nitrogen.....	2.0	3.60
Total.....	100.0	100.00
Average B.t.u.'s.....	1,105	1,095

Approximately 925 wells have been drilled to the Clinton sand, 200 of which have been dry or so small that their production was not utilized. At least eight wells have yielded an initial flow of 10,000,000 or more cu. ft. each, and two of these started, according to reports, with about 14,000,000 and 13,000,000 cu. ft. daily. Like most wells in the Newburg sand, many Clinton wells have declined rapidly in production. It is a matter of interest that the original well of the National Carbon Co. which "came in" on Feb. 14, 1912, with about a million cubic feet at a pressure

of 1,100 lb., was still producing 30,000 ft. in December, 1915, at a pressure of 100 lb. This well kept a pressure of 200 lb. for about 2 years. The life of this well has been, therefore, much longer than the average in the Cleveland field. The record of a well in the Clinton sand, as given by the drillers, with interpretations of the data, is given in Table 4.

TABLE 4.—*Log of Winton Gas Engine Co. Well, West 106th Street, between Madison and Western Avenue, Lakewood District. Clinton Sand*

	Feet		Thickness, Feet	Depth, Feet
8¼-in. casing.....	60	Drift.....	60	60
Top of lime.....	1,140	Devonian shale....	1,080	1,140
Water.....	1,450	Devonian and Cay- ugan limestone....	510	1,650
6½-in. casing.....	1,650	Salina.....	200	1,850
Salt.....	1,750-1,850			
Lime.....	1,850-2,590			
Slate.....	2,590-2,660	Niagara.....	830	2,680
Top of little lime.....	2,660			
Through little lime.....	2,660-2,680			
Slate.....	2,680-2,704			
Top of Clinton sand.....	2,704			
Gas.....	2,704-2,706	Cataract and Queenston.....	69	2,749
Through sand.....	2,706-2,713			
Red sand.....	2,713-2,721			
Slate.....	2,721-2,749			

Trenton Limestone

Because of the importance of the Trenton limestone of the Ordovician period, in the western part of Ohio, as a source of gas and oil, two wells have been drilled to that formation. They were both put down on the East Side of Cleveland, and both attempts were unsuccessful. One was drilled by the Park Drop Forge Co. at East 79th Street and the New York Central Railroad, and the other by the Cleveland Twist Drill Co. at Lakeside Avenue and East 49th Street. In the latter well a small flow of gas was found in the Trenton, and the highest rock pressure observed was only 37 lb. per square inch. An analysis of this Trenton gas, furnished by J. V. Emmons, follows:

Carbon dioxide, 1.4; heavy hydrocarbons (illuminant), 0.8; oxygen, 0.4; carbon monoxide, 0.6; methane, 95.7; hydrogen, 0.0; nitrogen, 1.1.

A condensed record from this well, in which the top of the Trenton was reached at 4,445 ft., is given in Table 5.

TABLE 5.—*Partial Well Record, Cleveland Twist Drill Co., Lakeside Avenue and East 49th Street, Cleveland*

	Thickness, Feet	Depth, Feet
Drift.....	200	200
Devonian shale.....	760	960
Devonian and Silurian limestone "Big Lime".....	1,722	2,682
Medinian limestone and shale.....	102	2,784
"Clinton" sand (no gas).....	36	2,820
Queenston shale and sandstone.....	360	3,180
Cincinnatian shale and limestone.....	1,265	4,445
Trenton limestone.....	132+	4,577

Pressures

The largest authentic rock pressure reported in the Cleveland field was a well of the National Carbon Co. which registered 1,120 lb. per square inch. Another of 1,250 lb. was reported, but it was probably not authentic. The usual initial pressure for most wells in the Lakewood region was about 1,100 lb., but in August, 1915, with few exceptions, it had dropped to 125 lb. In the West Park district the initial pressure of about 1,000 lb. had decreased to about 100 lb. in August, 1915. The Stadler well decreased from 950 to 100 lb. during its period of service of about a year and a half.

OIL

Although there are vast amounts of petroleum in the Devonian Ohio shales, it is disseminated through them in such small quantities that it is not obtainable economically. At many places in the district are so-called oil springs. The Euclid bluestone at some localities contains small amounts of oil which come to the surface along cracks and bedding planes. The oil, however, probably originates in the shales immediately underneath. The Mills gas well showed oil at 2,200 ft., at a position in the Niagara Limestone of the Big Lime. This is probably the horizon of the Newburg sand. Several of the wells drilled and abandoned by the East Ohio Gas Co. in 1907 and 1908 were left uncapped, and oil either flows out at the top, or can be obtained at a depth of a few feet where it probably floats on water. During the period of the gas boom, during 1914 and 1915, about 30 wells are reported to have found showings of oil. Of these, two were found in the "Stray" sand at a depth of about 1,400 ft.; four occurrences were in the Newburg, and the remainder in the Clinton. Several wells were reported to flow 30 to 40 bbl. daily, either before or after shooting, but the majority came in at 10 to 20 bbl., and practically all were short-lived. Most of the wells of the Cleveland field, after producing a few barrels for a month or so, have ceased pumping. It is,

therefore, evident that any who had hopes of an oil boom in the Cleveland area were disappointed. Probably the best oil well is the one which was the real pioneer of the district, that of the Newburg Brick & Clay Co. This gas and oil well came in on Oct. 17, 1911, with a natural flow of 35 to 40 bbl. After shooting, it produced 100 bbl. daily for a time, and in December, 1915, it was still making 2 to 3 bbl. a day. The oil was struck at a depth of 2,520 ft. in the Newburg sand. It is sold to the Standard Oil Co., and is said to contain little gasoline.

Table 6 gives three distillation tests made by the National Refining Co., of crude oil from the Cleveland district, all of which probably came from the Clinton sand.

TABLE 6.—*Three Distillation Tests of Crude Oil from Wells in the Cleveland District*

	Percentage	Gravity Beaumé	Flash point
Lakewood Crude*			
Gasoline fraction.....	17.185	From 71.4 to 56.8	Degrees Foster
Kerosene fraction.....	34.375	From 55.1 to 41.0	120
Gas oil fraction.....	12.500	From 40.6 to 36.7	260
Wax oil fraction.....	15.625	From 36.1 to 30.9	330
Residuum.....	20.315		
	100.000		
West Park Crude†			
Gasoline fraction.....	18.18	From 80.5 to 57.2	
Kerosene fraction.....	31.815	From 53.7 to 40.9	130
Gas oil fraction.....	13.635	From 38.6 to 35.7	345
Wax oil fraction.....	13.635	From 38.6 to 30.4	365
Residuum.....	22.735		
	100.000		
Cleveland Crude‡			
Gasoline fraction.....	20.310	From 79.4 to 55.8	
Kerosene fraction.....	31.250	From 54.6 to 41.2	122
Gas oil fraction.....	12.500	From 40.3 to 37.7	280
Wax oil fraction.....	15.625	From 35.8 to 31.8	355
Residuum.....	20.315		
	100.000		

* 16,000 c.c. of crude oil with a gravity of 41.2 Bé., from Lakewood, Ohio. This crude contains a large percentage of high-grade water-white oil which can be refined for market without the use of lead. The percentage of sulphur is very low.

† 11,000 c.c. from a 3-gal. sample of crude oil having a gravity of 41.1 Bé., from West Park, Ohio.

‡ 16,000 c.c. crude oil with a gravity of 42.2 Bé., from Cleveland, Ohio.

RELATION OF GAS AND OIL TO GEOLOGICAL STRUCTURE

With regard to reservoir gas, too few accurate data have been obtained to warrant any very definite conclusion about structural relations. In general, the southeast dip of the gas-bearing rocks produces a monoclinal structure, along which the gas tends to rise to the highest levels. Compared with central Ohio, these higher levels would be located in northern Ohio, and in the Cleveland district. In a discussion of the structure of this region, H. P. Cushing has pointed out that certain warpings have materially increased the southwest dip of the Berea sandstone across the Berea quadrangle, and that this should cause a corresponding flattening in the dip of the Silurian rocks which contain the gas. The occurrence of such flats between pitching areas would probably tend to cause the accumulation of gas or oil, because at such places they would have little tendency to move farther. The chief gas-producing territory, as outlined by present development, lies in the Berea quadrangle, and the most successful drilling is being done to the southeast of Cleveland across that region, which, according to Cushing, should be characterized by gentler dips. Recently in the vicinity of Wooster, about 60 miles southeast of Cleveland, anticlinal structures have been proven to exist to a small extent in the Clinton sand, but in the Cleveland field the few measurements that have been made have not shown the presence of such structures.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should be in the form of a new paper.

Report of the Secretary of the Committee on Safety and Sanitation
BEING A CLASSIFIED SYNOPSIS OF THE DATA COLLECTED BY THE
COMMITTEE

Your committee's secretary submits the following report, or summary, to the members of the committee, in an endeavor to lay before them a general review of the information so far received and also his own knowledge of the work as carried on at a number of plants, this being done in part to secure further information and suggestions from members and lead to a more effective means of furthering the work of the committee. It is suggested that more detailed data be compiled regarding engineering features involved, and that papers be contributed which will deal with the safeguarding of underground electrical installations.

There are few operating companies which have not started a more or less systematic campaign to safeguard their employees from accidental injuries and unhealthful surroundings, but a great deal is to be gained by experience. All may welcome the advice and coöperation of those who have been actively engaged in the work and have demonstrated the results obtainable by organized effort, and it is, therefore, exceedingly gratifying to note the ready assistance and hearty coöperation rendered by the companies that may be truly termed pioneers in the movement.

No solution of the safety problem is to be found in a strict standardization of methods, rules or devices, as the various mining, milling and metallurgical practices present too many conditions differing broadly in general principles, but some standard system embodying principles already proven to be efficient should be created, which may be modified or elaborated to meet specific requirements of each individual company, or to fit working conditions as they exist in the various mining camps throughout the country.

An industrial plant is primarily a business undertaking, therefore the economic side of a safety or welfare campaign is of importance. Many small operators have been deterred from undertaking the work by their belief that it involved an expense beyond their means, and one that could not be counterbalanced by greater efficiency or saving along other lines. This idea is generally due to a lack of knowledge of the actual costs of such work, and of the results secured by judicious selection of a method commensurate with the size and nature of the opera-

tion. The widespread adoption of safety measures by both large and small operators is conclusive argument in favor of economic results.

The cost need not be great, and should be, in a measure, proportional to the pay-roll. There are many ways in which a company will benefit both directly or indirectly by the work. The efficiency of the men is increased by their working under conditions where there is little danger of accident. Safety work with the proper enforcement of rules creates a higher degree of order, neatness and attentive application to work, thereby increasing the productive capacity of every man on the property, and lowering the operative costs. Legal fees, damage settlements, care of dependents, insurance premium rates, loss of time, and loss of experienced labor are all materially decreased, while a considerable burden is lifted from the local community in the number of dependents, cripples, widows and orphans. Improved sanitary conditions and a higher percentage of health among the men creates better artisans and more efficient labor.

Whatever means one may adopt to introduce safety measures, it is essential that certain fundamental principles be followed and the following suggestions may be helpful:

Secure the coöperation of every man and woman on the property.

Let all employees participate in the movement, thereby securing their interest and ready acceptance of safety rules.

Make it clear to all that the management is doing this work for the good of the workmen and their families.

From the outset, indicate plainly that the movement is to be made a success, and that it has the full support of the management.

Simplify rules, but create all that are necessary to guard against every hazardous practice.

Have no rules that are not enforced.

Whenever the judgment of the men has to be relied upon and strict ruling is impossible, have safety suggestions that will guide them to act properly and promptly.

Do not depend too much upon rules and regulations, but remove the danger wherever it is practical to do so.

Those administering safety orders must see that they are carried out. A boss ordering a dangerous roof taken down, or merely a nail driven flush, should not leave until his instructions are executed.

No possibility of conflict of authority should exist.

Constant vigilance must be maintained.

An educational campaign is absolutely essential.

No intoxication should be tolerated, and should be condemned by fellow workmen, as well as by the management.

Secure the interest of the men during their leisure hours.

Accurate records embodying classifications of injuries must be kept.

Every injury, no matter how slight, should be recorded, and medical attention or first-aid given by competent persons.

Any boss discovering an injury to any of his men which has not been reported, should inquire as to its cause, and reprimand failure to report such injury.

Medical examination of all men employed should be made.

Proper classification of injuries soon reveals a danger spot and the departments needing the greatest amount of attention.

Excessively long shifts cause fatigue, which leads to many accidents.

Eight-hour instead of 10-hr. shifts have been found by several companies to reduce the accidents occurring toward the end of a day's work.

Fooing and wrestling should be strictly prohibited, as many accidents are caused in this manner.

It is important to select the best grade of tools and general equipment.

Engage only sober, careful and reliable men.

Periodical bulletins, or booklets, published as a means of furthering the "safety-first" movement, arouse the interest of the men. Excellent examples of these are, the *Ingot*, published by the Raritan Copper Works; the *Anode*, published by the Anaconda Copper Mining Co.; *Employers' Magazine*, published by The Lehigh Valley Coal Co.

ORGANIZATION

It is impossible to suggest any fixed plan of safety organization, but a review of those created by some of the larger companies will illustrate how the executive end of the work may be handled and an appropriate system adopted.

Edwin Higgins has furnished us with a description of several examples of safety organization existing among the mining companies operating on the Lake Superior Iron Ranges.

The first form of organization is illustrated by that created by a company which operates a number of large mines on the same range; briefly it is as follows:

An inspector is in charge of the safety department. His duty is to inspect all mines as frequently as possible, and submit reports and recommendations to the manager.

A committee composed of mine superintendents, head mining captain, master mechanic, assistant auditor, secretary of pension department, safety inspector and the manager, who acts as an ex-officio member, meets once a month; it confirms or rejects safety recommendations, and considers all accident reports. A committee of mine foremen, consisting of three members, each selected from a different mine, makes periodical trips in and about the mines. The inspector accompanies this committee and incorporates its recommendations in his report. At each mine there is a committee made up of workmen, with duties similar to those of the committee last-mentioned, except that its activities are confined to the mine from which the members are selected. The personnel of this committee is constantly changed, so that in time all employees are given a chance to criticize conditions.

A committee, acting more or less independently of the others, consists of three mine superintendents. It is the duty of this committee to investigate all fatal accidents.

The second form of organization is illustrated by:

A company operating large and small mines. The safety department is in charge of an inspector who, with the assistance of three experienced miners, inspects each mine at least twice a week. These committee men report to the inspector, who, in turn, reports to the superintendent.

Inspector makes monthly report in triplicate; one copy is returned to the inspector,

one to the head of the department, and the third is retained by the superintendent. The department copy is returned with improvement report, when improvements are made. All company bosses and first-aid men meet every 2 months to discuss accidents.

Another form of organization is illustrated by:

A company operating scattered mines which has an organization similar to the one just described. The safety committees consist of all foremen from a district, and act under the supervision of a general safety inspector.

The fourth operates in connection with one large mine:

An engineer has charge of the department of "efficiency and safety." Under him are three assistants. These assistants are given a certain feature of the work to oversee, and are required to report to the head of the department. At first daily inspections were made, but as conditions have improved, only two or three trips through the mine each week are now found necessary. There are daily meetings, at which all matters pertaining to efficiency and safety are discussed; these meetings are attended by the manager, superintendent, head of the efficiency and safety department, and the mine captains. By holding meetings in the morning it is possible to consider the reports of the shift bosses to the mine captain, and prompt attention can thus be given to all urgent matters.

The Copper Queen Consolidated Mining Co. was the first to undertake this work in the Southwest, and their organization and methods have been widely copied throughout Arizona and adjoining States. They began their organization by the appointment of a committee composed of three officials and six workmen, selected from the different mining divisions and departments. This committee formulated plans and rules, and developed the permanent organization, namely:

A central committee, composed of the superintendent of the mining department as chairman, the superintendent of mines, the master mechanic, a hoisting engineer, an electrician, the chairmen of each of the four workmen's committees, and the safety inspector, who is also the secretary. This committee acts as a legislative body, all matters which cannot be handled by members of the other committees being referred to the central committee for decision. Where heavy expenditures are necessary, the matter is referred to the general manager, with central committee recommendations.

There are four workmen's committees, three being from the mines, and one from the shops, power plants, yards, etc. These committees have from four to six members, whose duties are to investigate all accidents of a serious nature occurring within their jurisdiction. Their report and recommendation, if passed upon by a majority of the members, are submitted to the central committee; they also pass upon suggestions sent in by workmen in their departments. The members are appointed by the inspector, and are supposed to serve for 6 months, appointments being made 1 or 2 months apart, so as to retain men who are acquainted with its duties. The inspector coöperates with all committees, and renders as much assistance to them as possible.

In contrast to these well-organized departments, we find one large corporation, employing between 12,000 and 13,000 men, entrusting all its safety work to one man, who, unassisted by committees, has made a great reduction in the casualty list during 1914. This is an excellent example of what can be accomplished by individual responsibility, but only in exceptional circumstances is it recommended.

METHODS

To secure results, the confidence and enthusiasm of every employee must be sought and maintained. It is necessary to adopt measures that

will continually call to the attention of all employees the importance of strictly complying with the rules and regulations, and being always vigilant of danger, and of the safety of others. Francis P. Sinn emphasizes the necessity of generalizing committee inspection work, as follows:

"Those most accustomed to certain dangers are, as a rule, the last to recognize those dangers. Any scheme for accident prevention which confines the inspection for safety of any department to men in that department will be defective on this account. It is usually the man unaccustomed to a particular line of work being done, and unfamiliar with the surroundings, who first sees the dangers at hand, and it is often he who has the best suggestions to offer to remedy the dangerous conditions. Whatever remedy is decided on, however, must fit into the particular conditions at hand, and must be entirely satisfactory to the man in charge of the work in question.

"It is at times hard to distinguish between a disregard of danger due to ignorance of its existence, and that due to a braggadocioal spirit; the latter is more far reaching in effect, as it sets a bad example to others and is less easily remedied. An educational campaign must be followed, which will emphasize the safety-first idea and be beneficial to those at the top of the organization as well as those at the bottom.

"To assure one of the coöperation of the men, it is important to demonstrate plainly that the safety campaign is entirely for their good, and it is well to have them participate in the movement from the very beginning; at the same time, having it distinctly understood that the management inaugurated the idea, and will do everything in its power to make the campaign a successful one. Before commencing any work, or giving any publicity to the plan, have a careful survey made of the existing conditions, and avoidable dangers; then formulate the plans accordingly."

The organization created or the person detailed to attend to the safety work must fulfill all duties involved in a systematic and business-like manner. All suggestions should be acted upon, and those presenting them advised as to whether or not they are of a practical nature and are to be followed. Suggestions will thus be encouraged. Prizes sometimes induce men to take an interest in the work.

It is unjust to criticize or blame individuals for accidents due to methods or conditions which they have not been taught to regard as dangerous. On the other hand, publicity should be given to all accidents, as a matter of education. Illustrations showing how such accidents occurred, and how they could have been avoided, should be placed on bulletin boards and shown at public gatherings. Any violation of rules must be severely dealt with, and either carelessness or recklessness duly censured.

The Bethlehem Steel Co. has very successfully secured the interest and coöperation of its thousands of employees, in safety work, without any organization. The work is entirely in the hands of their safety engineer, Mr. Funda. They realized that there were many dangers due to unguarded machinery and the general layout of their plant, which it would be impossible to immediately alter or safeguard, and they feared that if suggestions were made by workmen's committees and not at once followed up, a bad impression would be created.

Mr. Funda has carefully studied the matter, he fully realizes where guards and changes are necessary, and gradually the plant is being safeguarded throughout. They claim that 90 per cent. of the accidents are due to careless practices, and the results of their campaign have more or less demonstrated the truth of this theory. They had but four fatal accidents in 1914, and a notable decrease in serious ones. No fair comparison can be made between minor accidents in 1913 and 1914 because the more accurate record kept during the latter year would give apparent increase, where in reality there is a decrease. The company expects soon to create an organization including workingmen's committees, believing that sufficient headway has now been gained.

Any safety campaign should be entered into enthusiastically, but not with an idea of immediate perfection. The strongest elements in such work are an accelerated interest and the natural development of a working system, involving safer and more efficient operative methods. Reckless expenditure of money for safety appliances and prizes, or a general hurrah campaign at the outset will result in a reaction; men's enthusiasm will soon lag, and the movement be defeated by its own explosiveness. However, conservatism should not be carried to excess, as general enthusiasm must be created as quickly as conditions will allow.

The human element is an important one, and the psychology of the case must be carefully considered. The point of view of the men may be entirely different from that of the management, and it is essential to realize this fact. In this regard, Mr. Sinn says:

"There is a tendency, however, on the part of the workmen, and a perfectly natural one, to believe that they do not have to be told not to get hurt, as they are just as much interested in keeping alive and well as we are in having them so."

Every move in the educational work should be made with this point of view in mind, special emphasis should be given to the fact that we are all exposed to hundreds of dangers which, as busy men, we fail to remember, unless we are expressly reminded of them, and have the contributory causes and thoughtless practices clearly pointed out to us.

Where skilled labor exclusively is employed underground, the problem of education is not exceedingly difficult, yet even then the "old-timer" is likely to scoff at familiar dangers and disregard safety innovations. When foreign labor is employed, perplexing difficulties confront one; even requiring the men to sign receipts for rule books have in many cases nearly led to riots. The babel of tongues not only complicates the educational feature but it adds to the general confusion underground, and creates a natural nervousness among the workers, increasing the number of otherwise avoidable accidents. It is well to segregate the men according to race, as far as possible, and never allow a boss to have under him men to whom he cannot readily talk.

The number of accidents may be greatly reduced by having a central

employment agency; a physical examination of every man employed, and, at stated intervals while in the employ of the company, a rating based on such examination whereby men are given work for which they are better fitted; and a complete card index record of all employees, and of all accidents.

The Copper Queen Consolidated Mining Co. has adopted the following educational methods, in addition to its splendid system of first-aid exercises and the training furnished by committee work. Suggestion cards and letterheads are placed at all mines and departments, and may be used by any employee who wishes to send in a safety suggestion. The suggestions may be dropped in a box provided for that purpose or mailed to the inspector. They are collected from the box once a month. All suggestions are considered carefully by the central committee or the workmen's committee, and the sender advised of the action taken.

Whenever a fatal or serious accident occurs the inspector must be notified at once, and a report made by him of the accident. If the nature of the accident is such that photographs will prove of value, these are taken and filed with the report. Bulletins are published, telling where the blame should be fixed, and how the accident might have been prevented. Great care is used to secure all possible information, and to place the blame where it belongs, neither company, officials, nor workmen being spared.

A rescue station has been centrally located and is fully supplied with everything needed for safety, first-aid and rescue work. First-aid supplies are also placed in round, air-tight cans, 10 in. in diameter, made at the company's shops at a cost of \$4 each; these are distributed through the workings and, except during a short time at first, little trouble from meddlers has been experienced.

Competitive field meets between first-aid teams are held under the auspices of the Red Cross, and according to the rules of the U. S. Bureau of Mines. Bulletin boards are at such places as the public library, reading room, Y. M. C. A., dispensary and outside the rescue station and employment office; on these are posted monthly accident reports, photographs illustrating the right and wrong way of doing certain work, for which men posed as if caught by falling of ground, etc. Safety mottoes printed on large cards have been posted throughout the mine workings.

Smokers are given, where the men are furnished with "safety-first" cigars; at these smokers lantern slides are shown, and moving pictures are given in the Y. M. C. A. by the committee. All employees and their families are invited to safety rallies. The lantern slides of underground scenes have proven of great interest to the ladies. The men are constantly inquiring when the next meeting is to be held, showing their interest in these rallies.

Upon the Michigan Iron Ranges, the U. S. Government Rescue and First-Aid Car, and first-aid instructions, have been of great assistance in securing the attention and coöperation of the men; this is true not only in Michigan, but wherever the Federal demonstrations have been given.

Some companies pay for all safety suggestions, and give buttons to employees who have served on the committee. Cash bonuses have been tried in various parts of the country, but probably nowhere has a better method been adopted, or greater results achieved, than at the United States Coal & Coke Co.'s operations at Gary, W. Va. Howard N. Eavenson wrote the committee a letter, which was read at the Pittsburgh meeting as a part of the discussion of Mr. Higgin's paper, and which sets forth in the following words the premium system as practiced at Gary:

"For the past 4 years, the United States Coal & Coke Co. at Gary, W. Va., a subsidiary of the U. S. Steel Corporation, has been awarding premiums to mine foremen and assistant foremen for the prevention of accidents to its employees, which is arranged on a merit and demerit basis. The system adopted is as follows:

Qualifications

1. "No man shall be eligible for a premium for any month, in any position, who has not worked in that position every working day during the month excepting one, unless he shall have been promoted during the month from one position to another, and is eligible in both positions.

"*Explanation.*—It has been a custom for men in this section of the country not to work regularly. A number of accidents have occurred because of the regular foreman not working, and new men substituting. This qualification has therefore been inserted with the view of getting men to work regularly, and thereby assist in the prevention of accident.

2. "A man's work must be satisfactory to his immediate superior, and, if it is not satisfactory, his superior has the right to charge him with demerits to the extent of ten per month.

"*Explanation.*—This qualification is inserted as a means of discipline, as in a number of instances some of the assistant foremen do not take sufficient interest in the prevention of accidents to attend the weekly meetings of the officials for discussion and investigation of accidents which occur.

3. "This premium is not considered a part of the assistant foremen or foremen's wages, but is strictly in the nature of an award or a gratuity for faithful services, rendered to the company.

Distribution

1. "Each foreman or assistant foreman is charged with demerits for each man who is injured under his charge, each month, at the rate of 10 demerits for each minor, 20 demerits for each serious and 40 demerits for each fatal accident.

2. "Any foreman or assistant foreman who does not have any accidents under him during any month is given a credit of 5 merits, which will go toward reducing the number of demerits standing against him until all the demerits are wiped out, when he will not be given any further merits until he again receives demerits. No accident in which the victim loses less than 7 days will be considered.

"Explanation.—It is not considered advisable to allow a man to accumulate merits, as it would have a tendency after he had accumulated a large number of merits to cause him to be less careful.

3. "Any assistant foreman in whose section the company's mine inspector finds any dangerous practices or dangerous conditions which might cause accidents will be charged 5 demerits each visit he makes and finds such conditions. If he finds a section to be O. K., and no dangerous practices or conditions the assistant mine foreman will be given a credit of 5 merits.

"Explanation.—This provision is made as it is often the case that accidents occur for which the assistant foreman is not directly responsible; his place might be as safe as it is possible to make it, but through carelessness on the part of one of his workmen an accident might occur over which he would have no control. In order to aid an assistant foreman who has been so unfortunate as to have an accident of this kind to get back into good standing, it has been provided that 5 merits be given him if his place be kept in safe condition.

"On the other hand, an assistant foreman might permit dangerous conditions and practices in his section and still be fortunate enough not to have an accident, though not due to any special care or attention on his part. It is, therefore, provided that such assistant foreman be given 5 demerits for the condition of his section.

4. "The foreman's account will be charged with all demerits and credited with all merits of the assistant foreman under him, excepting when demerits are given for neglect of duty or causes other than accidents.

5. "No person who has 10 or more demerits to his credit at the end of the month shall be entitled to any premium, but if he has less than 10 demerits, he shall receive a premium of \$5, if an assistant foreman, and \$10, if a foreman.

6. "Any mine foreman or assistant foreman, who for 6 consecutive months is entitled to the monthly premium of \$10 or \$5 under the present rules, will at the end of the sixth month receive a special premium of \$15 or \$10, and for each month thereafter so long as his record is up to the requirements under the present rules, but when his record does not come up to the requirements under the present rules, he will have to again make a clear record for another 6 months before he is again entitled to a special premium.

7. "The foremen and assistant foremen have it distinctly pointed out to them by their immediate superior what men or jobs are under their supervision.

8. "If a foreman or assistant foreman leaves the employ of the company and later reenters it, he assumes all demerits charged against him when he left the company."

Since the adoption of the premium system in May, 1909, the company has paid 2,275 premiums, amounting to \$15,335. During this time they have increased the number of tons of coal produced per fatal accident inside over 400 per cent. Of course, this large increase cannot all be attributed directly to the premium system, but there is no doubt that it materially aided in accomplishing this result.

No assistant mine foreman is allowed to have under his charge more than 25 to 30 men, depending upon the way the places are scattered, in order to enable him to visit each working place at least every 3 hr. during the day. This gives the assistant mine foreman ample time, in case he sees any dangerous slate, a post that needs setting, or anything of a dangerous nature, not only to instruct the workmen to do it, but to stay with them until the post is set, or the dangerous conditions

are removed. Upon two particular occasions the company increased the number of men under the charge of each assistant mine foreman, with the result that in each instance an increase of accidents occurred.

The New Jersey Zinc Co. tried giving cash prizes at the end of 6 months to the shift boss who had the best record for freedom from serious accidents during that period, having a numerical rating for all accidents, closely following that used in reference to compensation insurance in the State of New Jersey. The element of chance entered into this plan to such a degree that they soon decided to distribute \$10 monthly to every boss who attained a record better than an arbitrary standard, fixed at a rating of 1.2 disabilities per 1,000 shifts of labor worked in his gang during the month. It was further decided to distribute safety cigars at the end of each month to the members of the gang having the best record.

A modification of the bonus system has met with some degree of success at one or two iron mines, which require all shift bosses to be safety enthusiasts, and well versed in the most modern methods. For this service they have increased the wages given to shift bosses. However, this method lacks the incentive of a cash reward for a good record, and introduces a disagreeable element in the possibility of having to dismiss relatively good men. It is doubtful if such method would meet with any success, unless adopted by a company operating in a district where there were a large number of companies which had not adopted the plan and which paid a smaller wage to their own shift bosses.

Precautions that are of universal interest to coal operators may be revealed by studying the rule books printed by companies operating in different parts of the country, and under the laws of the various States.

Many of the dangers in mining cannot be prevented by mechanical devices, therefore it is of interest to note some of the causes of underground accidents, and the means sometimes adopted to lessen these dangers. The following summary of safety devices and regulations adopted by the H. C. Frick Coke Co. will illustrate a few of these points:

- Rescue stations, and 33 rescue teams which practice quarterly.

- General rules and regulations providing increased air at working faces and at intake above that required by the State mining law.

- One hundred and eighty first-aid teams below and above ground.

- Bore holes from surface to release any dangerous accumulations of explosive gas in gobs.

- Shot firers and inspection of all places where shots have been fired to see that there is no fire or other danger.

- Permissible explosives.

- Blasting by battery.

- A system of pipes for sprinkling dust in dry places.

- Employment of steady, reliable and sober men only, for responsible positions.

- The best and safest oils that can be procured for illuminating purposes in the mines and mine buildings, etc.

All mine buildings to be well ventilated and kept clean and neat.

Cans for storing oily waste, etc.

Open lights in all buildings prohibited.

Electric wiring to be well done and examined at least twice a year.

Proper fire apparatus and fire brigades.

Finger boards and arrows pointing way out of mine.

Mines visited and thoroughly inspected by company's mine inspector at least once in 60 days.

Frequent meetings at the plant of superintendents and bosses to exchange views and discuss conditions, etc.

The examination of safety-lamp mines on Sundays, holidays and lay-off days. The mines which have been idle for more than 2 consecutive days are examined before operations are resumed.

Religious and political opinions of workmen not to be interfered with.

Employees must have permission to be absent from duty.

Mine officials are to see that "turns" are properly and equally distributed.

Care to be exercised and economy in use of materials and supplies.

No discriminations on account of nationality or creed.

Bristol recording and "U" gages on fans to be kept in good condition.

Safety switches at bottom of all steep butts.

Masonry stoppings between all air courses.

Fencing off all abandoned places in mines.

Systematic timbering.

Use of post extractors.

Automatic uncoupling device on rope haulages where they can be used.

All shafts where men are hoisted are equipped with device to prevent overwinds.

All machinery fenced off and shop machine belts, etc., guarded.

Stables, pump rooms, haulage engine rooms and shaft bottoms underground are all of fireproof construction.

Printed pamphlets "Safety Regulations—Machinery" provided for the guarding of all machinery, and posting of danger signs, etc.

Additional shafts to aid ventilation in mines and provide escape way.

Connections between mines as an additional escape way.

Gates at surface which cannot be opened until cage is there.

Safety catches at cage landings.

Safety bars on cages. Introduction of concrete-lined shafts.

Safety cabs on electric larries. Steel rail throughout all mines.

Ascensional ventilation for mines. Testing for gas above roof in gobs.

Prohibiting work in any place in which gas is discovered.

Printed pamphlets in five different languages showing how to do work in safety and guard against accidents.

As mentioned above, this company has issued a pamphlet entitled "Safety First," which is printed in the different languages spoken by its employees and illustrated by reproductions of photographs, which clearly point out the dangers of not following the instructions as laid down. The proper way of performing various duties and the outcome of carelessness are strongly impressed upon one's mind by these underground photographs, for which men have posed as if performing their

work in a careless manner with injury or death resulting. Footnotes describe these pictures and important instructions are printed in large type.

Photographs especially illustrate the danger of men walking along haulage-ways and being crushed by trips; of failing to put clevis-blocks on rails when loading cars in dips; of bad joints in tracks; of riding in front of trip, or between cars; of not standing in a safe place when cars which have jumped the track are shunted on again; of pushing cars with hands on the corner; of not obeying signals and waiting for trip coming in an opposite direction; of coupling cars on the inside of a curve where one would be caught and crushed if trip began to move; of the dangers of shock from trolley wires, if men carry augers, picks or shovels over their shoulders, should trolley wire guard be broken.

One company operating in the West Virginia section requires a systematic standard amount of timbering, whether the top is considered dangerous or not, and extra timbering where unusually bad conditions prevail. By this requirement, they do not have to rely so much upon the judgment of the men as to whether or not the top is dangerous. This same company requires a clearance of not less than $2\frac{1}{2}$ ft. on each side of the mine cars along all haulage-ways. In some cases, water is piped to all workings to dampen the roof and side and wet down the dust.

SAFETY SIGNS AND SIGNALS

It is necessary to have signs to warn men of dangers, and to point out places of safety, and means of exit. In view of the fact that many languages are spoken by the laborers of this country, and that the miner is naturally a great wanderer, it is highly important that some universal danger sign that needs no verbal explanation be adopted. Mr. Higgins suggests that the solid red circle be accepted as a universal sign, as it appears to be in every way adapted to underground use. He further suggests an arrow as a direction sign to exit along manway, etc., and a symbol of a ladder, to indicate a ladder-way or stairway. It is, however, impossible to cover all the requirements of danger signals by symbolic signs, but whatever wording is used should be as brief and explicit as possible. Too much variation in signboard instructions leads to confusion. It seems as though some distinction should be made between a danger sign and those denoting warnings or bearing instructions, as the red ball should instantly impress upon the mind the necessity for immediate caution. A few of the signs observed by Mr. Higgins in various parts of the country are described by him as follows:

Signboards for Points on the Surface

"Danger: Keep Away" (around the boundary of surface caves).

"Do Not Walk On These Tracks."

"Danger; Keep Away From Shaft Collar."

- "No Smoking Allowed Around Shaft Collar."
- "Lighted Candles Or Lamps Not Allowed On Main Cage."
- "Look Up" (under trestle or tippie where rock, ore or coal is handled).
- "Keep Out."
- "Careless Workmen Not Wanted Here" (on office).
- "No Admittance Without Permission From Office."
- "Live Wire."
- "High Voltage. Do Not Touch."
- "Safety First."
- "Riding On Ore Skip Positively Prohibited."
- "Riding On Incline Forbidden."

Signboards for Shaft Stations

A placard showing complete code of hoisting signals.

- "Do Not Ring To Move Cage Or Skip When Men Are Working In Shaft Or Sump."
- "Not More Than — Men Permitted To Ride In Cage."
- "Cage Doors Must Be Closed When Men Are Being Handled."
- "No Loafing Around This Station."
- "Throw All Rubbish Here" (at entrance to small opening off station).
- "Miners Not Permitted To Ride On Motor."
- "Look Where You Are Going—This Means You."
- "Wrestling, Pushing Or Crowding Positively Prohibited."
- "Safety First."
- "2,000 Level" or "2,500 Level."
- "Look Out For Motors" (Locomotives).

Signboards for Pump Stations

- "Keep Oily Waste In Receptacle Provided."
- "Do Not Oil The Timber—Oil The Machinery."
- "Do Not Hang Candles Or Lamps On Timber."
- "Keep This Place Clear Of All Rubbish."
- "Beware—Live Wire."
- "High Voltage—Do Not Touch."

Signboards for Mine Workings

- "To 2nd Outlet Shaft" (with arrow or hand pointing).
- "Ladder-way To 12th Level."
- "To Main Shaft" (with arrow pointing).
- "To Timber Shaft" (with arrow pointing).
- "Dangerous Ground—Keep Away."
- "Danger Overhead."
- "Look Out For Trolley Wire" (at points where timber carrying the trolley wire is sagging, it is assumed that the wire has guard boards wherever men are at all likely to come in contact with it).
- "Look Out For Motors."
- "Danger—Dynamite Stored Here."
- "Powder Magazine."
- "Danger—Blasting Here."
- "Danger Ahead—Old Workings."

"Ladder-way To Surface."

"No Smoking Allowed."

"Danger From Ore Trains—No Traveling In This Tunnel" (or Drift).

"No Candles Or Lamps Allowed In Magazine."

"No Candles Or Lamps Allowed In Stables."

At entrance to a gaseous mine, or district: "Open Lamps, Pipes, Cigars, Cigarettes Or Matches Must Not Be Taken Into This Mine" (or District).

The Tennessee Coal & Iron Co.'s committee on the wording of "safety signs" is as follows:

The wording of signs should be as brief and as much to the point as possible. The word "Danger" should be prominent on all signs put up at dangerous points. The word "Warning" should be prominent on all signs as to how an employee can best do work with safety.

The following signs are considered applicable to all works:

1. "Notice—Stop Machinery Before Oiling, Wiping Or Repairing."
2. "Elektrika."
3. "Danger" (a tag).
4. "Danger—Look Out For Cars."
5. "Danger—Keep Out."
6. "Danger—Will Not Clear Man."
7. "Danger—Keep From Under Loads Carried By Crane."
8. "Commit No Nuisance. No Loafing Allowed" (six different languages).
9. "Danger—You are warned against working without proper eye protectors, or with battered tools. Get proper tools and eye shields from your foreman."
10. "Warning—Employees working around engines, moving or revolving machinery shafting, etc., are warned of the danger, and are prohibited from wearing torn clothing, loose or unbuttoned jackets, blouses, shirts, long neckties, and loose sleeves. Always wear jackets tucked in the trousers or under the overall bib. Never forget to examine your clothing before commencing your work."
11. "All persons are forbidden to enter any cylinder of an engine until flywheel has been securely blocked, to prevent possibility of engine turning over."
12. "Danger—No Smoking Allowed."
13. "Danger—Do Not Move This Valve."
14. "Danger—Man in Boiler."
15. "Danger—Keep Off."
16. "Danger—Do Not Walk On Track Or Trestle."
17. "Engine Room Rules" (printed on cardboard and framed—copy attached).
18. "Boiler House Rules" (printed on cardboard—copy attached).
19. "What To Do In Case Of Electric Shock" (printed on cardboard and framed—copy attached).
20. "Danger—This is electrical apparatus and carries dangerous electrical current. All persons not especially authorized to work on the same, are hereby prohibited from doing so. Failure to comply with this warning may result in death" (five languages).

The following signs are applicable to all mines:

1. "Notice!—All persons are hereby warned not to travel on the slope, or ride on loaded or empty cars. Any one failing to comply with this notice will be discharged. You must travel the manway."

2. "Danger—Magazine."

3. "Notice! No Powder, Fuse, Caps Or Oil Allowed In The Wash House."

The following signs are applicable to the different works:

Ensley Works

1. "Notice! Men Must Take Fuses Out Before Beginning Work On Or Around Crane, And Return Same When Through."

2. "Danger! 6,600 Volts. Do Not Climb This Tower."

3. "Danger! This Platform Is Dangerous When Pouring Heat."

4. "Danger! Do Not Pass Between Wall And Track."

5. "Danger! Look Out For Larry Car."

6. "Danger! Employees Are Warned Against Crossing The Run-out Tables. Always Use Stairs."

7. "Danger! No Smoking Or Bare Lights Allowed In This Room."

Ore Mines

1. "Warning! No Traveling On This Slope. Must Use Manway."

2. "Danger! Do Not Go Inside Or Beyond This Notice Until Engine Is Stopped."

3. "Bell Signals" (copy attached).

Coal Mines

1. "Danger! Do Not Open Gate."

2. "Danger! Do Not Get On Cage Without O. K. From Cager. Ten Men Are A Load."

3. "Danger! Do Not Get On Cage From This Side."

4. "Escape-way To No. 6 Slope."

5. "Escape-way To No. 2 Slope."

6. "Escape-way To No. 1 Shaft."

7. "This Way To No. 1 Shaft."

8. "Escape-way To McArdle Slope."

9. "This Way Leading Along Fault To No. 6 Boilers."

10. "Warning! No Open Lights Allowed In This Stable."

11. "Danger! Keep Out! Gas!"

12. "Danger! Do Not Exceed 4 Miles Per Hour."

13. "Notice! No Loafing. For Signal Man Only."

Foundry Furnaces

1. "Do Not Pass Between Column And Track."

2. "Danger! Keep Out! 2,300 Volts."

3. "Notice! Danger! Do Not Touch."

4. "Danger! Keep Off Coal Bin."

5. "Danger! Keep Off Of Track."

Bessemer Rolling Mill

All signs reported in use at Bessemer Rolling Mill are general signs.

All signs should have the sanction of the department superintendent before being placed. Care should be exercised to see that all signs can be complied with. None should be placed unless they are enforced.

All signs should be made of enameled 18-gage steel. The groundwork should be white, with blue block letters and blue margin.

The size of letters in the words "DANGER" and "WARNING" should be 2 in. in height; balance of the letters 1 in.

We would suggest that all signs be made $8\frac{1}{2}$ by 11 in. or some multiple of that size.

The minimum space between the letters shall be three-fourths of the width of the lines of the letters. The minimum space between words shall be three and one-half times the width of the lines forming the letters.

SAFETY GUARDS

Many of the machine guards and other safety devices used in power plants, and in factories of all kinds, are likewise serviceable throughout the mechanical and metallurgical departments of a mine. Wherever exposed gears, cutting tools, fly-wheels, revolving shafts, belts and grinding or emery wheels, are within the reach of those working nearby, there is danger of serious injury, unless proper guards are installed. The general standards recommended by the committee of safety engineers of the Workmen's Compensation Service Bureau describe the equipment approved by the insurance companies and State inspectors. The book published by this bureau will be found most helpful, when providing guards for machinery already installed. If new machinery is to be purchased, every effort should be made to eliminate dangerous construction, and the specifications should embody safety requirements. In most cases it is cheaper to design a machine for permanent safety than to annex guards after it is in place.

According to recommended standards, the following danger points are the usual ones needing protection:

All gears and all friction clutches within 7 ft. of floor or platform should be completely encased by metal or close-meshed wire screen.

Vertical shafts should be encased to height of 6 ft. from floor or platform.

Horizontal shafting and dead ends of shafts less than 6 ft. from the floor or platform should be encased or protected by railing.

Horizontal belts within 6 ft. of floor to be guarded on top and sides. Overhead belts within 7 ft. of floor to be guarded beneath and on both sides; those more than 7 ft. from floor to be guarded beneath.

Vertical and inclined belts should be either completely inclosed, or with a railing $3\frac{1}{2}$ ft. high, and 5 in. from belt.

Belt shifter should be provided.

Belt pulleys within 7 ft. of floor should be encased or properly protected by railings.

Self-oiling bearings and shaftings provided.

All protruding or sunken parts on rapidly moving machinery are exceptionally dangerous.

Remove or guard revolving bolts or open keyways. Where keyways are essential, carefully fitted pieces of wood may be inserted.

Dead ends of shaft may be easily encased in pipe fittings. A cheap and novel means of encasing shafting is described in the "safety" supplement of the November issue of *The American Industries*. Ordinary three-ply mailing tubes the diameter of the shafting are cut to fit the length and split to slip over the shaft. These are fastened on by means of gummed paper, and should be shellaced to give added strength and durability. They may be wound spirally with tape, if greater strength is needed. These guards revolve with the shaft, but should a workman accidentally come in contact with the tube, it will cease revolving, thereby eliminating the danger of clothing being caught in the mechanism.

A personal knowledge of a number of large plants, and a set of photographs from The New Jersey Zinc Co., bring to mind the following appliances and brief details of their construction, viz.:

Place double-bar iron guardrails, $3\frac{1}{2}$ ft. high, and provide with toeboards wherever there is an opening to a working place below:

Around all flywheels.

Around motors and dynamos, or least on side where men have to pass.

Around belts near floors.

Around permanent floor openings.

Around all open tanks.

Along the outer edge of all platforms, stair landings, bridges, open mezzanine floors, and open pits.

In front of hot refuse piles or slag heaps (with danger sign).

In front of doors or pathways near railroad tracks or other places where accidents occur from men stepping directly in front of moving cars, trips or cranes. (The necessity of walking around the end of such railings allows time to observe approaching danger.)

In front of engine blowers and grinding wheels.

Movable rail guards or wire-mesh gates installed around open elevator shafts, trap doors, etc. These should be automatically placed in position when openings are otherwise left unguarded.

Handrails in stairways, on all open sides; on one side of closed stairways less than 4 ft. wide; on both sides between 4 and 8 ft. in width and on both sides and in the middle for over 8 ft. wide.

Permanent ladders to have handrails at the top, and platforms at every 10 ft. in height.

Portable ladders provided with safety shoes (spikes for wooden or dirt floors and automatically adjusting flat shoes for concrete or stone floors).

Vacuum exhaust system for shavings, sawdust, grinding dust at all saws, lathes and emery wheels.

Locked gates leading to space back of electrical switchboards, and rubber mats near all exposed parts.

Safety belts for men working in hoppers.

Safety wrenches for opening side-bottom dump cars.

Safety lid opener for ore cars.

Safety grab hook for handling hot pokers.

Safety coupling tools for brakemen.

Goggles worn by men exposed to eye accidents.

Eye protection furnished where high heats have to be observed.

Hand warnings on tracks in front of moving cranes and machine carriages.

Safety catches on elevators and cages, and overwind guards on all hoists used by men.

Good light and ventilation add greatly to the safety of all working places.

NOTE.—The Ingersoll-Rand plant at Phillipsburgh, N. J., is ideal in this regard. Mr. Zook, the construction engineer, has designed the buildings so that no working rooms are more than 60 ft. wide, except in the machine shop, where sawtooth roofs with northern exposure, have been adopted. Steel-sashed windows have added greatly to the lighting conditions in the new packing building.

ELECTRICAL EQUIPMENT UNDERGROUND

Electricity is today an important element in underground development, and owing to the conditions surrounding such installations, there are dangers which seldom enter

into other lines of electrical equipment. The presence of underground water, gas, explosive dust, and explosives used in mining, adds to the dangers of fire or shock. Owing to the constant changes made necessary, as underground workings are extended, it would be impossible to install installation that would be employed if the wiring was to be permanent.

At isolated properties, men with little knowledge of electricity are necessarily employed to equip the mine, and men with no knowledge or realization of the dangers associated with high potentials are in constant peril, and even experienced men are liable to accident due to poor insulation, low trolley wires, improper inspection and the cramped quarters in which they have to work.

Completely insulated circuits are far less dangerous than those that are grounded, as in the former case the current can be confined to its proper channel if adequately insulated. Trolley wires are especially dangerous, as there is always the possibility of men making a grounded circuit by either coming into contact with the wires themselves, or having tools accidentally touch them. Equipment may become charged if insulated from the ground. Improper ground connection is often caused by sanded tracks or excessive amounts of oil, in which case locomotives and cars may give to men standing on the ground as severe a shock as the trolley wires would.

Fires may be caused by short-circuits due to defective insulation, or combustible matter placed near fuses and other portions of the circuit where arcing may take place. Coal and wood may be ignited in this fashion. Explosions may be caused by the ignition of explosive gas and coal dust, if an electric spark of sufficient intensity passes through a dry atmosphere containing these elements. Explosives are frequently accidentally discharged either in the act of being transported upon electric haulage-ways, or by accidental detonation. It is always dangerous to use a grounded circuit for firing purposes, as the difference in potential between the explosive and the ground at a slight distance away may cause accidental detonation before battery connections are made, if an exposed surface of the wire should come into contact with the ground.

The underwriters' rules regarding electrical installations will be found of great assistance to any mine manager who has to deal with electricity for either power or lighting purposes.

RULES FOR EMPLOYEES

From the various rule books at hand, the following are submitted for approval, correction and additions. This list and grouping is the result of careful study, rearrangement, rewording and elimination, but those in charge of the safety work at the various plants will no doubt be able to make valuable suggestions owing to their daily familiarity with the practical conditions and unusual emergencies calling for special regulations.

It must be borne in mind that the rules are intended as a guide to others in formulating their own sets and consequently cannot provide for local dangers, while, on the other hand, it will include rules that can only be adopted by a few.

General Rules on Safety

1. Each and every employee is hereby instructed to report to his foreman any defective tools or appliances or any feature of his working conditions or surroundings which he may think affects the safety of himself or others. Do not work with tools or appliances of any kind that are defective.

2. If you are injured, no matter how little, you must report or get word to your foreman at once. A slight scratch often causes blood poison. (Report or have foreman report all injuries to the doctor.)

3. If you find anyone injured or sick on the property of the company, whether an employee or not, take him to the doctor.

4. Do not report for work when under the influence of liquor. Intoxicating liquors about the works are strictly prohibited.

5. Vigilance and watchfulness insure safety. To avoid danger adopt the safe course. An employee must not trust to the care exercised by another when his own safety is involved, but, on the other hand, always consider the safety of others.

6. When working above others great care must be taken not to drop any material without first giving warning to those below. When you are going to either work above or below other men, let those men know about it.

7. Never throw down any material without first looking to see that it will not hurt somebody.

8. Neatness and order are a safeguard against accident. Boards with nails protruding must not be left around the mine or premises.

Construction men must clean up all loose boards after them, but all men are requested to remove dangerous loose boards or planks they see and report to foreman.

9. Do not leave bars, tools, or material of any kind, leaning against a wall so that they are likely to fall and hurt some one.

10. All employees, except motormen, switchmen, trammers, are strictly forbidden to ride on moving cars.

(NOTE.—Designate those authorized to ride.)

11. Do not get on or off any elevator while it is in motion.

12. When working around machinery in motion do not wear gloves or clothing with loose ends or loose fitting coats or sleeves.

13. Do not reach or lean across moving belts or gears.

14. Do not set in motion any machinery without first seeing if anyone is in a position to be injured and whether all safety guards are in their proper place.

15. No one must undertake to put on or remove a belt while the wheels driving it are in motion, except men skilled in such work, and they must always use the proper appliances for the purpose.

16. Never undertake the removal of a belt which may have become wrapped around a shaft or pulley while such is in motion.

17. Keep from under loads suspended in the air.

18. Any workman removing a cover from any opening in floors, tunnels, ground or pit, or any man removing a brattice or stopping must see that it is properly replaced before leaving the place.

19. Do not start machinery or turn on electric connection without first seeing if anyone is in a position to be injured and whether all safety guards are in their proper place.

20. No one must handle explosives except men skilled in their use and properly authorized, and all precautions must be taken to prevent accidents from blasting. Explosives must not be left exposed or unguarded.

21. Do not use defective ladders, or stairways. Notify your foreman.

22. It is dangerous to permit molten matte or metal to come into contact with water, and you are warned to exercise great care to avoid such danger. Dangerous explosions may result from carelessness in this regard.

23. All employees shall use the regular roads and stairways for travel. They shall look out for moving cars on both the motor and railroad tracks. Keep off railway tracks except at regular crossings.

24. Any conduct which may be termed "fooling," such as wrestling, boxing or throwing pieces of loose material, is strictly forbidden.

25. Do not fool with compressed air and never blow compressed air on anyone.

26. When you complete a job never leave tools or materials overhead, and always replace safeguards.

27. Never rush from the light into a dark place or from the dark into a light place. Go slowly, use caution.

28. Do not touch electrical machinery, power or electric light wires, unless you are properly authorized to do so.

29. No person in a place of trust shall deputize another to do his work without the sanction of his superior officers.

30. No person in a place of trust shall absent himself from work without a legitimate excuse, or having first obtained permission from his superior officer, and when possible he must send notice.

Mining Department

1. Every working place shall be regularly inspected and employees who find anything in an unsafe condition shall report same to their foreman.

2. The general condition of the timbering in the mine shall be made safe and kept so.

3. A miner's first duty shall be to pick down the roof and walls, and not permit the shovelers or other employees to work under a place until it has been tested and made safe for them.

4. If an employee drops any material or tools down the shaft or deep working place, he shall immediately report same to the hoisting engineer, who will have the shaft inspected before continuing with the regular work.

5. In stopes worked by the square-set method, the working floors shall be securely lagged over, and the lagging shall be long enough to reach halfway across the caps or girts.

6. Every winze, raise or incline of steeper slope than 40° from the horizontal and deeper than 40 ft., through which men are obliged to travel, shall be provided with a ladder-way.

7. Suitable ladders or footways shall be provided to connect floors in stopes and other places requiring communication in the mine.

8. Ladder-ways shall be provided in all shafts in the course of sinking to within such distance from the bottom as will secure them from damage by blasting, and from the end of such ladder-way portable ladders shall be extended to the bottom of the shafts.

9. Broken or defective ladders shall be reported and repaired immediately.

10. Ladders shall not be removed from their usual places without orders from the shift boss.

11. Employees are forbidden to throw tools, steel or any other material down the manway, but shall lower them with a rope provided for that purpose.

12. Carmen shall exercise care when loading out of a chute, also when pushing cars, that their hands may not be crushed against the timber.

13. (Winzes or raises shall not be started in the direct line of a drift, but shall be offset from the drift.)

14. Every winze, or raise, now opening from below directly on any drifts or tunnels traveled by men shall be covered by a grizzly or by doors. The opening of such offset winzes shall be protected by a fence or guardrail not less than 3 ft. nor more than 4 ft. in height above the level of the drift.

15. Men shall learn the various exits and raises or winzes connecting the level on which they are employed with other levels.

16. No employees except the motormen and switchmen shall be allowed to ride on a motor or motor train.

17. When men other than those employed on the motor are using the motor track, they shall notify the motormen accordingly, for the motor has the right-of-way.

18. Employees are forbidden to carry tools, steel or other material upon their shoulders in any workings where there are electric light or trolley lines. This is dangerous and may result in death.

19. Stationary lights shall be provided at all stations in the working shafts, and at night at all working places on the surface.

20. Employees shall not enter a drift, stope or other workings, where powder smoke, gas or bad air is such as would be injurious.

21. Employees shall use great care in keeping all underground ventilating doors closed.

22. No candle shall be left burning in a mine or any part of a mine when the person using the candle departs for the day.

23. All compartments of shafts used for the hoisting and lowering of men and ore shall be inspected at the beginning of each shift to see if there are any broken guides or other defects. Any defects, if found, shall be reported and repaired before further use of said compartments.

24. Men working in the shafts shall have a suitable covering to protect themselves from material falling down the shaft.

25. They shall give full information concerning the place and nature of their work, to all hoisting engineers on duty during the time they are employed in the shaft, so that the cage will not be let down on them.

They shall have their working platforms of sufficient size and strength to safely carry on their work.

26. In no case shall a cage, skip or bucket be lowered to the bottom of a shaft when men are working there. It must be stopped at least 15 ft. above the bottom until the signal to lower further has been given by one of the men at the bottom of the shaft. (This rule shall not apply to shafts less than 50 ft. in depth.)

27. No open hook shall be used with a bucket in hoisting, but only some approved form of safety hook or shackle hook.

All shafts or winzes shall be cleaned down below the bulkhead after each blasting.

28. The top of every shaft shall be protected by a substantial guardrail or chain.

29. At all shaft stations a gate or a guardrail shall be provided and kept in place across the shaft except when a cage is being used.

30. All chutes, manways, winzes, raises and other openings shall be covered by a substantial hatch, planking or grizzly, or provided with guardrails or chains, and shall be kept in such a condition that men cannot walk or fall into them.

31. Employees shall not crowd or rush while getting on or off of cages.

32. When using the cage, all signals shall be given while standing on the cage and not upon the station.

33. To release the cage the signal shall be given while standing upon the station and not upon the cage.

34. Powder nippers shall be very cautious in the handling of explosives, both underground and on the surface. Before using the railroad or motor tracks they shall make sure that they have the right-of-way. They shall also use extra care in handling the powder on the cages and in all other places.

35. Employees shall be careful in the opening of boxes of powder.

36. All powder used in stopes, raises and other places where it is necessary to hoist or lower it with a rope shall be carried in sacks provided for that purpose. A box may slip out of a rope and its use is, therefore, forbidden.

37. All blasting powder used shall be properly marked with the date of manu-

facture on each stick, and no powder shall be used after 12 months from the date of manufacture.

38. All explosives shall be stored in a magazine provided for that purpose alone; the magazine shall be electrically lighted and shall be placed far enough from the working shaft, tunnel or incline to insure its remaining intact in the event of the entire stock of explosives in the magazine being exploded. No powder or other explosives shall be stored in underground workings where men are employed. All explosives in excess of the amount required for 24 hr. work must be kept in said magazine, and such temporary supply shall not be kept at any place within the mine where its accidental discharge would cut off the escape of miners working therein.

39. Smoking in the magazine and while carrying powder is forbidden. The carrying of lighted candles or lamps in the magazine is also forbidden.

40. Oils or other combustible substances, or blasting caps, shall not be kept or stored in the same magazine with explosives.

41. Cap crimpers shall be furnished and must be used. Crimping with the teeth is dangerous and is forbidden.

42. Skewers shall be furnished for the use of all miners handling explosives and shall be used for making holes in the powder for the fuse. No candlesticks or other instruments shall be used for this purpose.

43. All employees are forbidden to use anything other than a wooden tamping stick in the loading or charging of a hole with blasting powder or other high explosive.

44. Tamping shall be done by pressure and not by strokes.

45. Miners shall never be alone when blasting, but must keep at least one man with them. They shall have a reserve light near a manway or some other convenient place in case their lights go out after "spitting" the fuse.

46. Before firing charges, warning must be given in every direction from which access may be had to the place where blasting is going on.

47. The number of reports shall be counted by the miners firing same, and misfire holes shall be reported to the timekeeper and posted on a blackboard made for that purpose. The shift bosses shall also notify their men working in places where misfires have occurred.

48. No employee shall be permitted to enter a place where a misfire has occurred until 30 min. after the time when the fuse was lighted.

49. Employees shall not attempt to extract explosives from a hole where a misfire has occurred, or in which all of the powder has not exploded. In such a case, a new charge shall be added before detonation.

50. Where boulders are being blasted by placing explosives on them and not in holes drilled for that purpose, employees shall see that the powder is well covered with clay, or some other suitable material, so that it cannot take fire.

51. Where boulders at the base or on the slope of a muck pile are so charged, the employees in charge of the work shall not permit any person to disturb or knock down any loose rock from above, as it might cause an explosion.

52. No employee other than the motormen and switchmen shall be allowed to ride on a motor or motor train.

53. Motormen shall ring their bell and slow up when coming to curves, switches and doors.

54. They shall have an electric light on each end of their motor at all times.

55. They shall report any defects in their equipment to the electrician or to the foreman and have it repaired.

56. All cables used for hoisting purposes shall be of approved quality and manufacture.

57. All cables shall be daily inspected by some competent person appointed by

the mine superintendent, and any defects shall be immediately reported and the ropes changed if necessary.

58. Ropes shall be kept plainly marked for the benefit of the hoisting engineers.

59. All cages used for the hoisting or lowering of men in shafts deeper than 300 ft. shall be equipped with iron bonnets, with gates at least 5 ft. in height, with overhead bars of such arrangement as to give every man on the cage an easy and secure handhold. They shall also be equipped with safety catches of sufficient strength to hold the cage at its maximum load at any point in the shaft in case of the breaking of the hoisting cable.

60. Emergency chains shall be used from the cable to the cage in case of a breakage in the king bolt or clevis pin; and also between the upper and lower decks in case of a breakage of the connecting pins for these decks.

61. All cages, together with the king bolt and clevis bolt, shall be daily inspected by some competent person appointed by the mine superintendent. They shall be kept in first-class condition and all defects shall be reported and repaired before further use.

62. The safety catches on cages and crossheads shall be kept well oiled and in good working condition.

63. The cage men and top men shall keep a careful watch over the cages during their shift's work and immediately report all defects and have them repaired before continuing their work.

64. At no time shall more than (the number allowed depending upon size of cage) men be permitted to ride on the deck of a cage.

65. No person shall ride upon a cage loaded with tools, timber, powder or other material except for the purpose of assisting in passing these through the shaft.

66. No person shall ride on a cage loaded with rock or ore.

67. When hoisting or lowering tools, timbers, or other material in the shaft, the ends, if projecting above the top of the cage or bucket, shall be securely lashed to the cable or to the upper part of the cage; and tools, timbers or other material loaded erectly upon a cage shall be securely lashed before being hoisted or lowered.

Surface Department

1. If you remove the cover from a valve pit, sewer, or any manhole, see that the opening is guarded so that no one can fall into it. Holes caused by excavation, or any other cause, must be similarly guarded.

2. Always use cross bracing in excavating where there is any danger of a cave-in.

3. Do not remain near where blasting is being done.

4. Use care in placing ladders before using them. If there is any danger of the ladder slipping have someone hold it.

5. Never try to climb a ladder without the free use of both hands. If material is to be handled use a rope.

6. Never rush from the light into a dark place or from the dark into a light place. Go slow; use caution.

7. Do not attempt to jump on a moving car.

8. Always see that the brakes are set before starting to unload a car, and never leave a car set out without setting the brakes.

9. When opening dump cars, exercise great care so that the dumping lever will not hit you.

10. Do not walk alongside of cars when material is being thrown from them. It may strike you.

11. In unloading cars, material should be piled at least $4\frac{1}{2}$ ft. from the rail. If it is impossible to do this, place a warning sign at each end of the pile. Any material that may fall upon or alongside the track must be cleaned up immediately.

12. Do not pile any material so high that it is liable to fall or cause another pile to fall.
13. Care should be taken in loading material on cars so that no portion will project over the sides, or fall off, while in transit.
14. Do not take hold of rope or cable above a block, as your fingers may be drawn into the block and injured.
15. Never leave a derrick boom so that it can be swayed by the wind.

Boiler-House Department

1. Report all defective conditions to superintendent of plant.
2. Two men should always work together when cleaning out boilers, it being the duty of one man to remain outside of the boiler all the time in a position to see the man working in the boiler and give assistance in case it is needed.
3. Never go into a boiler until you have locked the steam valve and closed all other valves.
4. Before entering a boiler put a torch or candle inside to determine the presence of gas or bad air.
5. A sign "DANGER, DO NOT MOVE," should be hung on the steam valve of the boiler when it is shut down.
6. Safety valves should be tested every shift.
7. Never tighten any bolts or nipples or do any calking while steam pressure is on the boiler.
8. In turning a boiler into the main steam line after being out of service, the steam pressure of the boiler should be brought to within 5 lb. of the main-line pressure and the valve next to the boiler should be opened first to allow any water in the connecting pipe to flow back into the boiler, after which the valve next to the main should be opened very slowly until the pressure is equal in boiler and steam line. When boilers are off for cleaning, the stop valves next to the main must be closed. No dependence is to be placed on any automatic valve alone. These latter valves must be examined each time boiler is out of service, so as to insure their working in emergencies, such as the bursting of a tube.
9. No one should enter a boiler without first making sure that all valves are provided with signs "DO NOT OPEN," or locked shut before any one is allowed to go inside boiler.
10. Where more than one boiler is connected solidly into one blow-off main, the boiler cleaned should be locked shut before any one is allowed to go inside boiler.
11. To prevent water tenders being injured by a glass bursting, water glasses should be provided with a No. 16 brass coil spring, about $\frac{1}{8}$ in. greater in diameter than glass and with about $\frac{1}{8}$ in. between separate turns so that the water level can be easily observed.
12. Combustion chambers must be kept free from accumulation of clinkers, ash and dust.
13. If the water forms a hard scale, the use of soda ash may be considered, or the use of good mineral oil, such as kerosene, may prove of benefit. About a $\frac{1}{2}$ gal. of each 100 H. P., should be placed in the boiler when empty; the addition of water causing the oil to follow the surface of the shell and loosen scale.
14. Each boiler plant should be equipped with the proper tools for firing, such as rakes, slice bars, steel poker, and a proper size of shovels (about No. 3). A proper place must be provided in which to keep all tools, and they must not be allowed to lie around promiscuously.
15. The boiler, when being taken out of service, should be cooled down as much as possible before being blown out; as the practice of blowing out hot water and filling

with cold water and blowing down, will cause the scale to bake on the tubes and shell plates from the heat remaining in the setting, as well as cause serious strains on shells.

16. Boilers should be blown at least twice every 24 hr. (more often with bad water) while in operation, and special care should be taken to see that the blow-off valves are kept from leaking. Feed valves must not be allowed to leak to such an extent that boilers have to be blown down to prevent high water.

17. Gages must be tested sufficiently often to insure corrections.

18. All boilers must be equipped with three gage cocks, and a water glass; except locomotive boilers, which only require gage cocks.

19. Bottom water valves must be provided for blowing out gage glasses.

20. Thermometers should be installed in feed lines of all plants and an effort made to attain the highest possible feed temperature. It might be stated here that each 10° heat added to the feed temperature means approximately 1 per cent. saving in coal, as well as increasing the capacity and reducing cost of cleaning and repairs to boiler.

21. In all horizontal return-tubular boilers the tubes must be headed at both ends.

22. In all water-tube boilers the tubes must extend through sheet about $\frac{5}{16}$ in. and flared to an angle of about 35° with the straight portion of tube well down against the tube sheet. This work should be done with a regular expanding tool.

Blast-Furnace Department

1. Men working in limestone, ore, or coke bins must notify the foreman of the unloading department as to the bin in which they will be working. When working in a bin always use a safety rope, fastened above so that it will not be cut in two by a passing train.

2. In loading cars, be careful not to get struck by pieces of ore or limestone bouncing off the car.

3. Do not walk out from behind a furnace on the charge floor without making sure that the track is clear.

4. When handling a hook or bar in the furnace, be careful not to strike the trolley box.

5. When pulling a short bar with the chain along the length of the furnace, get far enough away from the bar so as not to be dragged in, in case the crust breaks and the bar drops into the furnace.

6. When breaking blast-furnace crusts do not attach the chain to the locomotive.

7. Do not use too short a bar or too short a hook.

8. Do not use wet or moist bars around tap holes, spouts, or nose pieces.

9. Protect your eyes when digging away crusts from spouts and nose pieces.

10. Don't walk over settler roofs without first seeing that they are covered with sheet iron.

11. In tapping out a furnace at the back, see that the ground between the dams is dry.

12. Never use a key, gad, or bar, which is battered or burred on the striking end. Have the burr dressed off before using.

13. Don't throw moist "clean up" into matte ladles.

14. When helping on the slag machine, never stand near the molds where they enter the trip. Always stand behind the center of the trip.

15. Don't stand at the edge of the casting machine platform over the skip pit, or go into the pit when the skip is taking up a load of slag.

16. Whenever an arc light, or incandescent light, is broken or goes out, report it at once.

17. Motors must not be driven by any one except the motor driver or the foreman.

18. Motor drivers must not enter the concentrator tunnel with the charge train when the red-light danger signal is illuminated.

19. Motor drivers and helpers must stay with their trains until their relief comes.

20. Loud shouting is forbidden except for the purpose of preventing accidents.

Special Rules to be Observed by Employees Handling Explosives

1. Do not smoke.

2. Light no matches.

3. Handling and firing of dynamite is very dangerous both for you and others around you. Be careful.

4. Boxes of dynamite should be opened outside the powder house with wooden mallet and wedges.

Remove scale from powder cans very gently; do not tear open the can in any way.

5. Keep powder away from electric machinery.

6. Keep powder dry.

7. Don't allow dynamite, powder, electric caps or fuses to lie around loose in places where you are working.

8. Don't try to thaw dynamite. The dynamite used at this plant will not freeze.

9. Caps are very dangerous.

Do not bite them.

Do not drop them.

Do not step on them.

Do not carry them in your pocket.

Do not put them near dynamite.

10. How to make a dynamite cartridge.

Keep away from a fire, windward.

Do not smoke.

Cut off square the proper length of fuse (never shorter than 2 ft.).

Stick end of fuse into cap.

Crimp cap with crimper. (Never use a knife nor bite the cap.)

Seal joint with soap.

Unfold end of paper covering stick of dynamite.

Make hole in dynamite with round sliver of wood.

Lower cap into hole by fuse.

Fold paper around fuse and tie with a string.

11. Rules for blasting.

Tamp holes with wooden rammers only.

Ram powder down gently; do not pound.

Shake, or spring, or chamber, only one hole at a time.

Watch your shots.

Allow plenty of time for the rock to cool before putting in another charge.

In charging holes with black powder put in two cartridges, one with a fuse and one with an electric exploder.

Send warnings in all directions to get under shelter before blasting.

Do not spit more than five fuses at one time.

Count your shots. If a charge misses fire, do not touch it except under personal direction of the general foreman. A well-made cartridge never misses fire.

12. Rules for blasting machine.

Do not put a sharp bend in the wires.

Clean ends of wires before connecting together.

Twist tightly and wrap connection with tape.

Connect holes in series; one wire of the first hole is connected to one from the third hole, etc. This leaves a free wire from the first and last bore holes.

Connect these to the wires from the blasting machine immediately before blasting.

Rules for Coal Mining in General

1. Every person shall at all times obey the lawful demands or orders of his superior officer, or the person under whose charge he shall be.

2. When the boss tells you how to keep yourself safe, do as you are told.

3. No person in a place of trust shall deputize another to do his work without the sanction of his superior officers.

4. No person in a place of trust shall absent himself from work without a legitimate excuse, or having first obtained permission from his superior officer, and when possible he must send notice.

5. Every person who shall observe, or who shall come to the knowledge of any danger or dangerous condition in any mine, traveling way, air course, or defective door, brattice chuteguard or mine appliance, which might cause a dangerous condition, shall immediately report it to the mine foreman, fire boss, or other person in charge, so that it may be repaired, but before leaving his place he shall put some plain warning across the entrance thereto to warn others against entering into danger, or tell his foreman or other official of any bad condition or dangers as soon as he knows of them.

6. Fire boss must examine all parts of the mine, especially where falls have occurred; he marks the date upon the rock, of the fall and upon the working face, also at a third point such as the side of a post.

Men must look for fire boss marks and also report the absence thereof immediately.

7. When you enter the mine, ascertain the conditions of your working place from the fire boss as soon as you arrive at the fire boss's station.

8. When you get to your working face, the first thing to do is to examine the roof of your place, and see that all loose rock, slate or coal is taken down and timbers set where needed.

9. Test the safety of the top and frequently as you mine the coal away beneath look for "slips."

10. Miner tests top by holding his hand against the top and striking with his pick at the same time—trust only to this method of feeling a movement; do not depend upon a "sound test."

11. If top cannot be reached by hand, miner uses two picks, one to feel for motion and the other with which to strike.

12. When top is found to be unsafe, post must be set at once.

13. Dangerous to go under unsupported drawslate, also to try to take it down by forcing pick handle between it and the top; if handle slips, man will fall under slate and probably be killed.

14. Thin coal under drawslate, which will not support it, but hides the slits, is exceedingly treacherous. Be watchful.

15. Set post and crosspieces to drawslate if coal is broken.

16. Set crossbars in working places that have been, or are likely soon to be, under great weight before you have mined it out.

17. Set plenty of timber in your working places.

18. Set the timber IN TIME—delay is dangerous.

19. Where you are not sure the top is safe, set "temporary" timber for safety while putting up "permanent" timber.

20. If the top is crushed or broken, do not leave it up—it is not safe.

21. It is not always safe to knock out posts, but generally safe to pull them out. When top is badly broken or has big slits in it, do not attempt to take out any props or crossbars. Rib bosses are employed to superintend this work, and see that a safe exit is provided.

22. Extra posts are set and rib boss tells miner which post to remove.

23. Timber men shall order all props, cap pieces, and all other timbers necessary, at least one day in advance of needing them. If he fails to receive such timbers, and finds the place unsafe, he shall vacate it until the necessary timbers have arrived.

24. Company's rules for timbering rooms are that posts should be in rows, 4 ft. apart and 4 ft. between posts; they should all be in line. A miner shall set no less but shall set more posts if top is not good.

25. While working on entries they must keep lagging ahead of crossbars. Crossbars should not be set close to top but room left for lagging.

26. Timber men must test entries and set posts where top is found to be unsafe.

27. Unsafe places must be fenced off, and standard danger signals set; both fence and signal are required.

28. Signs with pointing finger and the words, "This way out," show proper manways, and course for men to follow.

29. After setting timbers, bury all chips or small sticks, as they may become ignited and start a fire.

30. Do not put your hands on corners of cars to push or place them.

31. Keep YOUR track in good condition.

32. Travel on roads made for that purpose.

33. Do not pass DANGER signs.

34. Put a "drag" on rear of a car being hauled out of "dip" places.

35. Use a clevis-block in front of car wheel while loading, in all places going to the dip or to the rise, where cars would run away.

36. To make it more safe, use a tie-and-block in front of the car in all places going to the rise.

37. All forward motion of cars should be controlled by the use of sprags, and no one should place himself in front of cars or trip.

38. Do not carry tools, like augers or iron bars, on the shoulder on roads in which a trolley wire is hung.

39. Be careful of the passing trips in going back and forth to work in the mine. Should there be no manway, as is the case in a great many places, look out for the manholes in the slopes or motor roads. The manholes are white-washed.

40. Make yourself familiar with the manholes, so that you will know where every hole is. Do not use hauling roads if there is a manway.

41. Acquaint yourself with all the inlets and outlets of the mine in which you work. By doing this, you will be better able to find your way out if a fire or other accident should happen.

42. No miner or other working man shall be permitted to introduce into the mine any stranger or person, on any pretense, without the consent of the superintendent.

43. Always use the safety devices provided.

44. Ventilation at all times is one of the most important considerations. Air courses must be kept free of any impediment.

45. No miner shall enter any working room or place in which the air is taken to the face by means of a brattice, if this brattice is down or in defective condition, so as not to take the full current of air to the farthest portion of the room or place. If any brattice is found to be in defective condition, it must be reported to the mine foreman, or fire boss, at once, and the room or place shall not be worked until the brattice has been repaired.

46. All machinery, tools and other equipment should be examined at least once daily by duly authorized persons. Generally this duty is entrusted to foremen or to the highest man in charge of the work directly affected by the equipment to be examined.

47. All broken, damaged or unsafe cars must be sent immediately to the shop,

and plainly marked "shop," so that there is no danger of their being shunted into operative use on the way.

48. Tracks, trolley wires and switches must be kept in first-class condition at all times.

49. Coal should be properly undermined or overcut before blasting, according to the special requirements of each mine.

50. Old workings must not be entered unless due authority is given.

51. Any person who opens a door, removes a brattice or stopping, must see that it is properly closed or replaced before leaving the place.

52. Miners and all others are forbidden to knowingly or willingly deface or remove marks which may be made in any part of the workings for the guidance of the working operations. All employees are forbidden to displace, injure or damage in any way the stoppings, props, tracks, machinery or any other apparatus used in and about the mines.

53. Persons ascending or descending a slope, or riding in a car, will not be allowed to enter or leave the car while it is in motion.

54. The mine gives off explosive gas, and safety lamps only are used, or

In every working of a coal mine approaching any place where there is likely to be an accumulation of explosive gas, or in any working where there is imminent danger from explosive gas, no light, lamp, or fire other than a locked safety lamp shall be allowed or used.

55. Men are searched for matches and pipes upon entering and have their lamps examined, which is in compliance with the State law, offenders being arrested.

56. When safety lamps are in use they shall be used with the greatest care. Every person on receiving his lamp must examine it to see that it is securely locked; while at work he shall pay frequent attention to his lamp, and if it becomes unsafe from fire damp, oil, or gasoline spilled upon the glass or gauze, the gauze punctured, even with a small pin, the cracking of the glass, or any other cause whatever, he shall at once extinguish the light and immediately take it to the lamp house or relighting station and report it to the foreman or fire boss, or

Be careful not to injure your safety lamp.

57. A safety lamp shall at all times be kept in a perpendicular position and must be kept hung up when practicable, and at a safe distance from the swing of the pick, hammer, or other tools, or

Safety lamps must be locked, inspected and kept in upright position, and no matches shall be allowed in workings where lamps are used.

58. No person shall improperly use or damage any safety lamp, or attempt to blow out flame in any lamp except for purpose of testing a lamp before entering the mine; and no person, unless duly authorized, shall unlock or attempt to unlock or open any safety lamp, or have in his possession any key or contrivance for opening any safety lamp.

59. Employees will be charged with all breakage of safety lamps and for the loss of a lamp.

60. Do not drill holes to within 6 in. of the back of the mining. Do not drill out missed shots.

61. You should go to the powder house in the morning for your powder, and under no circumstances take out more powder than is required for one day's work.

62. Provide yourself with a safe container for carrying powder. Powder should not be carried in your pockets. A fiber powder case can be secured from the company's office. Such a powder case is safe, and has been used with great satisfaction to miners.

63. Do not mix giant and other powders in the same cartridge; it is dangerous.

64. Leave dummies enough in readiness to charge the hole, or holes, to the mouth.

65. Be very careful when filling the dummies that no coal is mixed in with the adobe clay or slate tamping.

66. Under no conditions should miners be allowed to use coal dust, or other combustible materials, for tamping.

67. When drilling rib shots, take care not to grip the holes too tight on the rib.

68. Do not get the holes too high, as they will get into the roof and make roof conditions bad.

69. The shot lighter shall see that all passages leading to the place where the shot is to be fired are properly guarded before lighting the shot; and when firing a shot in a place approaching a workings into which the shot might possibly hole through, it shall be his duty to give warning to the men in the workings.

70. Before firing a shot in a dusty place the floor, sides, face and roof must be thoroughly dampened for a distance of 20 ft. from the shot.

71. Do not fire shots with less than 100 ft. of "lead" wires.

72. Do not fire shots with damaged "lead" wires or bad connections.

73. Do not carry "electric caps," except in a locked case.

74. Do not fire a shot where there is any coal dust within 100 ft. of the shothole, or where the dust has not been thoroughly wetted for a distance of at least 100 ft. from the shothole.

75. Do not put more than $1\frac{1}{2}$ lb. of explosive in any shothole.

76. Do not fire shots where there is not enough shelter from the blast.

77. Misfires must not be withdrawn, or the holes reopened, nor should the miner return to the place for 8 or more hr. if fuse is used, and not until current is disconnected and 5 or 10 min. have elapsed, if battery detonation is employed, or

Do not go back to a misfire shot before the lead wires are disconnected and 5 min. have passed.

78. If you have charged a shot and cannot fire it, fence off the place and put up a danger signal before leaving it.

79. Any person firing a shot must observe the above rules.

80. When charging his hole for a blast or shot, if the cartridge sticks he must remove it carefully and reduce its size, or enlarge the hole so it will enter easily. He must not ram or force the cartridge with a drill. When black powder is used, he must use a tamping rod tipped with copper. For other powder he must use a wooden tamping rod. Care must be taken to tamp the cartridge with sufficient depth of material. Slate or clay should be used in all cases. Coal dust must not be used. If his place be dry and gives off inflammable gas, all fine dust must be removed from the immediate vicinity of the shot, or the working face, roof and floor thoroughly dampened for 20 ft.

81. After each blast he shall exercise care in examining the roof and coal, and shall secure them safely before beginning work.

Tools

82. The use of iron needles and iron tamping bars not tipped with 5 in. of copper is hereby declared unlawful. Where dynamite is used a wooden tamping bar only must be used.

Car

83. No person shall ride upon a loaded cage, or car used for hoisting purposes, in any shaft or slope, except rope rider or couplers; nor shall any coal be hoisted out of any mine while persons are descending into such mine, notice of which shall be kept posted at said mines.

Care and Handling of Cars

84. Broken, damaged, or unsafe mine cars must be marked "Shop" by employees handling cars. It is part of the duties of top and bottom cagers, brakemen, switchers, engineers, motormen, loaders, drivers, and tippie men to so mark any defective cars coming under their observation and shunt the shortest way to the shop.

85. The brakemen on the bunker run will see that all cars marked "shop," or that are known to him to be defective, are switched on the shop repair track and not taken out until repaired.

Shot Lighter

86. The shot lighter shall be subject to the mine foreman, and fire all properly placed shots after first examining the working place as to the state of inflammable gas, or coal dust, and if any be found the shots shall not be fired until such danger is removed.

Switchmen

87. It is the duty of the switchmen to attend to switches and see that they are properly set, couple and uncouple the trip, brake or sprag the cars, keep a lookout ahead for danger, and signal the motorman or engineer.

88. Switchmen must not allow any person to ride on cars without permission of the manager or foreman.

Cagers

89. It shall be the duty of the top and bottom cager to take charge of the slope during working hours and to report promptly to the mine foreman any defect in track, rollers or signal wires; shop all defective cars, and in case of a wreck on the slope to send word to the mine foreman.

90. The top cager will make an inspection of the slope after any wreck, or at any time he has reason to believe there is anything wrong in the slope.

91. The cager will not allow anyone to ride on a loaded car, or when coal is being hoisted on the opposite side.

92. The cager will not allow anyone to get off or on a car while in motion, or after the signal has been given the engineer to hoist or lower.

Drivers

93. Be careful in getting on and off cars when they are in motion. Avoid getting off moving cars, if possible, near a prop or close rib.

94. Do not get between cars to uncouple them or release brakes. Keep your legs from between bumpers.

95. Do not ride between cars unless your work cannot be done any other way.

96. Be very careful in spragging cars, as drivers have been known to lose fingers by doing this carelessly.

97. In riding on front end of empty or loaded cars, keep your head low enough so that it will not be caught between the car and cross-timbers. Lives have been lost by not keeping the head low enough.

98. Do not make "flying switches."

99. Do not ride in front of loaded trip unless your work cannot be done any other way.

100. Keep your trips under control.

101. Promptly notify the mine officials of:

(a) Bad track.

(b) Slate, dirt, or posts piled along the side so that you cannot pass the trip in safety or keep it under control.

(c) Bad roof, when known to you.

102. Do not excite your horse or mule by rough or bad treatment.

103. The driver must properly care for mule while in his charge, feed and water it at dinner hour, and must not abuse it at any time, nor allow anyone else to do so.

104. The driver must report the condition of the mule and harness to the mine foreman in case of injury, sickness, loss of shoe, or defective harness.

105. The driver must stop all defective cars that come to his attention and report all defective track, chutes, and dangerous conditions on the entry or gangway.

Motormen and Locomotive Engineers

106. They shall at all times keep a lookout ahead and act promptly on seeing or receiving any signals.

107. They will be held responsible for the care and cleanliness of their machines and shall report any defects at once to their foreman.

108. Motormen must follow instructions received from the electrician in regard to the care of their machines.

Trackmen

109. Examine the roof carefully before commencing to lay switch or repair track, as you may be working at that place for several hours.

110. Do not carry spike or crowbars, picks or shovels on your shoulders when traveling in entries where trolley or naked wires are; you might get a shock.

111. Always be careful in repairing track near trolley or naked wires. Men have lost their lives by being careless in such places.

112. Always report on unsafe conditions that you may find in the mine to mine foreman; or, if possible, remedy the defects yourself.

113. Report to the foreman all defective track, switches, trolley wire, or appliances, and dangerous places.

114. Do not allow any person, except your switchman, mine foreman, trackman and inspector, to ride on the motor or locomotive, or on the cars, without the permission of the manager or foreman.

Trappers

115. Do not keep your door open any longer than to allow trips, mules or men to pass through it.

116. Your door was built to conduct the air to other parts of the mine, and should not be kept open longer than is absolutely necessary.

117. Do not get in the dark if you can avoid it, and do not go to sleep behind your door. Trappers have been seriously injured by this carelessness.

118. Do not change the cotton in your lamp while in the mine, not throw the unburned wick away. This has caused fires and serious loss of life in the past.

Pumpmen

119. It shall be their duty to keep the pump in their charge in good condition for work. All ordinary repairs to pumps and pipe must be made by the pumpman.

120. Serious damage or breakage of pumps, or failure to keep the water within a safe limit, must be reported immediately to the master mechanic and mine foreman.

121. Reports of insufficient supply of steam or power must be made daily until remedied.

122. Tracklayers, gangway timbermen, pillarmen, roperiders, bottom cager, top cager, pumpmen, are subject to the mine foreman and must execute their work as ordered from time to time by the foreman.

123. The above employees before absenting themselves from work must properly arrange with the mine foreman.

Master Mechanic

124. The master mechanic will have full and complete charge of all machinery in and around the mine, and must see that it is properly used, kept in good repair, and clean.

125. The master mechanic must inspect, or cause to be inspected all ropes and machinery used for hoisting or lowering employees out of or into the mines in each 24 hr.

126. The master mechanic will be chief of the mine fire department and will see that fire hydrants are placed and kept in repair, and a supply of hose is on hand for the protection of all mine buildings and mine slopes and pump rooms.

127. The master mechanic must keep the ventilating fan running as directed in writing. Such notice in writing must be posted in the fan house. In case of accident that will necessitate shutting the fan down, the mine foreman shall be immediately notified and the fan must, if possible, be kept turning over until word is received from the mine foreman to shut down.

128. The master mechanic will examine, or cause to be examined, the fan at the beginning of each shift and see that it is kept in perfect order. If it becomes necessary to repair the fan he must give timely notice to the mine foreman.

129. In case of a mine fire or explosion, he must report immediately at the fan and keep it running as usual and not allow anyone to stop or reverse it, or make any changes himself, without written instructions or direct telephone communication from the mine foreman or manager.

130. All men connected with the running and care of all engines, boilers, and other machinery, and employees in the machine and blacksmith shop, are subject to his supervision and orders.

Hoisting Engineers

131. Hoisting engineers must at all times pay strict attention to signals, and to the trip. When hoisting or holding a trip on the slope, under no circumstances shall an engineer leave his reverse and brake levers; and particular care must be exercised in hoisting and lowering man trips, which must never be run at a speed exceeding 600 ft. per minute.

132. Any defect in the engine or rope must be immediately reported to the master mechanic.

133. The engineer will consult with the mine foreman as to the speed trips are to be run, and report any defective track to the foreman.

134. A copy of the "Code Signals" must be hung up conspicuously in the engine house.

135. It shall be the duty of the engineer, subject to the control of the master mechanic, to see that the firemen perform their work properly, and that steam in sufficient quantity and pressure is furnished to do the work.

136. The engine house and engine must be kept neat and clean.

137. Firemen must at all times keep their boilers properly supplied with water and immediately report any accident to pump, injectors, or pipes furnishing same, and

any deficiency in the water supply. Should the water in the boiler get below the try cocks or gage the fires must be pulled immediately and reported.

GENERAL RULES AND SUGGESTIONS FOR FOREMEN AND SUPERINTENDENTS AS PER
JONES INTEREST IN WEST VIRGINIA COAL MINES

Coal Mines

1. All sights to be in center of entries and entries kept on sights.
2. Roadsmen laying track must fishplate every joint; and bond everywhere electric haulage is used or is expected to be used.
3. When a parting is laid, a bond of sufficient length to close the break caused by the switch must be put in.
4. When a parting is removed and a filler rail put in, it must be splice barred and bonded if electric haulage is used or expected to be used and the bond which was used when the parting was in, must be removed to a safe place for use again.
5. When roadsmen have finished a job on which they are working, they must pile up all extra ties and iron, gather up all spikes, splice bars and bolts, and put into kegs ready for removal to next job.
6. Insist on miners loading cars to their capacity.
7. Insist on miners loading all slack and machine dust.
8. Insist on miners keeping all slate, binders and other impurities out of the coal.
9. Insist on miners keeping road close to rib side of room; except in mines operating with road in the middle.
10. Insist on miners keeping themselves safe.
11. Insist on miners working full time when mine is in operation.
12. Allow no miners to load any unnecessary slate cars.
13. Insist on machine men cutting entries before rooms and cut close to bottom.
14. Insist on machine men removing exposed bits before allowing mule to haul the machine.
15. Insist on machine men and loaders having dull bits sent out, placing them on brake side of coal which check hangs.
16. Insist on the machine man keeping his section cut up or put somebody else in his place. This is for the purpose of keeping up the regular tonnage.
17. Any employee wishing to be absent from duty must first apply to and receive permission from his foreman.
18. Any workman offering money, liquor or valuables of any kind to a foreman, boss or clerk will be discharged; and any foreman, boss or clerk accepting money, liquor or valuables of any kind from workmen will be discharged.
19. Entries must absolutely be not over 10 ft. wide.
20. Entries must be kept on sights.
21. Slate must be trimmed off sides and loaded before yardage is taken up.
22. Entry men must not fill in road with bug dust.
23. Entries must get all the cars they can load; must be cut as soon as cleaned up.
24. One cut or more can be taken out of room necks as the entries advance, providing it does not delay the entry.
25. Air must be kept to the faces.
26. Rooms shall not be driven over their prescribed distance unless to recover some lost coal.
27. Great care must be taken in posting rooms, and if in danger of falling they should be reposted or road removed.
28. Allow no slate to be piled on rib or blind side of rooms.
29. Try to induce the miners to block off the coal in front before shooting and take less powder.

30. No room should be allowed to get behind as it prevents uniform shooting of ribs and receives all the weight from other rooms.
31. Posts should be set in break-throughs to support rooms.
32. No loaded or empty cars should be permitted to stand idle at any point.
33. All rooms to be measured up before rib drawing is begun.
34. All break-throughs to be finished before entries are driven more than three cuts ahead.
35. Have short slate posts in entries.
36. Room sights shall be carried in the middle of track and shall be sighted each cut.
37. Motormen and drivers upon receiving empty cars should always examine brakes to make sure they are in good order, and if there is any defect or the wagon is not safe to load, mark the wagon and return it empty to the outside.
38. Safety or derailing switches must be installed in such parts of the mine as in the opinion of the mine foreman may be necessary to prevent runaway cars from injuring employees.
39. Puncher runners cutting in sections shall have two safety lamps and keep them out of the dust as much as possible when cutting.
40. All safety lamps to be examined by a person designated by the mine foreman, other than the one cleaning and filling them.
41. Where water lines are used they shall be kept within 60 ft. of the face and not closer than 20 ft. and in extending them the valve shall always be carried on end of line next to the face.
42. It shall be the duty of the mine foreman to see that haulage ropes, sheaves, cages, etc., are examined according to law.
43. In all shaft mines when cage has not been running for some time, or after supplies have been lowered, an empty cage shall be run before men shall be allowed to ascend or descend.
44. In shaft mines men shall not be hauled on a cage when anything else is on the other cage.
45. In shooting entries about to hole through no shot shall be fired until the shooter himself warns the men on the other side.
46. Danger boards shall comply with State mine law.
47. No place shall be considered as fenced off unless a danger board is fastened to fence in such manner that a man cannot cross without seeing the board. Laying a board on the bottom does not comply with this rule.
48. All miners must use the kind of powder prescribed by mine foreman for this mine.
49. Loaders must not drill holes for the purpose of blasting further in any direction than the undercutting or mining.
50. Motormen must not run with trolley pole ahead except where it is impractical to turn it. Where it is absolutely necessary to run with pole ahead, trip must be kept under control so no injury can occur to motorman through breaking of pole in case it should fly from wire.
51. Motormen must report at end of each shift exact condition of their motors, an account of work performed during shift and all delays interfering with the progress of his work.
52. Motormen must keep sufficient supply of sand in sand boxes of motors for a round trip.
53. Motormen must examine their motors frequently and see that all connections, contacts and gears are in perfect working order.
54. Motormen must thoroughly understand the mechanism of their motors and keep them in good running order.

55. Motormen must oil their motors regularly.
56. When anything is out of order that motormen cannot readily fix, motor must be shopped.
57. Motormen must not overload their motors.
58. Motormen must not throw the reverse lever to change the direction of the current when motor is moving in the opposite direction, except in case of emergency.
59. No shooting of any kind must be done unless the explosive is placed in a hole properly drilled and tamped, except under the immediate supervision of the mine foreman.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Shot-firing in Bituminous Mines

BY M. D. COOPER,* E. M., CURTISVILLE, PA.

(New York Meeting, February, 1917)

FOR the purpose of obtaining some first-hand data in regard to the shooting down of coal in bituminous mines, it was the writer's good fortune to be employed as a shot-firer for almost one year. In all, 6,020 shots were fired by him during this period.

All of the work was done in what is known as the South Main section of the No. 2 Mine of the Ellsworth Collieries Co., Ellsworth, Pa., a subsidiary of the Lackawanna Steel Co.

Position of Shot-firing in the Cycle of Operations

The established practice of the Ellsworth Collieries Co. calls first for the undercutting of the coal by electric chain machines across the face of the entry or room. This is followed by replacing any posts removed by the machine-runners, or the setting of new posts or timber, as may be required. After this has been done the machine cuttings are loaded into cars. Then the coal is spragged, and a shot hole drilled to the depth of the undercut. Next in order, the shot-firer loads the hole and fires the charge. The working place is then cleaned up, timbered if necessary, the sides and face squared up, and loose material taken down. The face is again ready to be undercut.

Materials Used

Permissible explosives were used exclusively in blasting coal. These were Carbonite No. 2, a nitroglycerine explosive; and three different makes of ammonium nitrate explosives, Tunnelite, Red H, and Mine-ite 5-D. The last-named was used during the greater part of the period. It is interesting to note that even at the beginning of this work, the gases resulting from the explosions of the charges caused no especial distress on the shot-firer's part and this continued to be the case except on the occasions when a change was made from one class of explosive to another.

* Assistant Superintendent, Ford Collieries Co.

At these times the gases had an unpleasant, but not at all serious, effect for perhaps a day. The explosive was brought into the mine by the miners, each man being required to have a small wooden box for this purpose. The box was equipped with a grooved cover, and was of sufficient size to contain nine cartridges, each $1\frac{1}{4}$ in. in diameter and 8 in. long. Without a box, the miner was refused a supply of explosives.

For the breaking up of an occasional rock fall in an entry, straight 40 per cent. nitroglycerine dynamite was used.

All of the charges, both of permissible explosives and dynamite, were detonated by du Pont No. 6 electric blasting caps, equipped with 6-ft. iron wires. The uniformity of results obtained from these caps will be

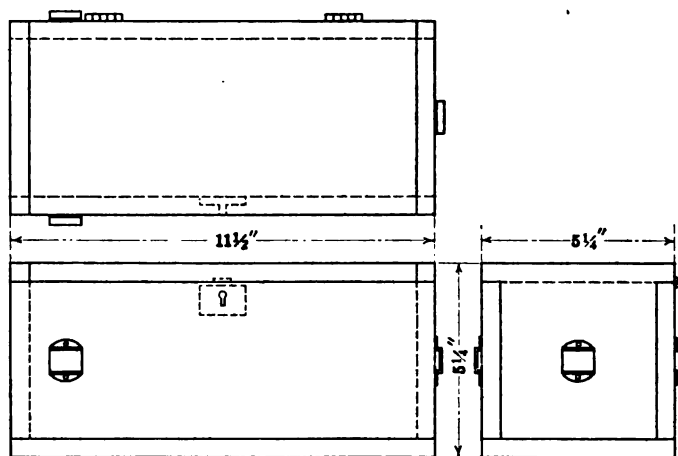


FIG. 1.—BOX FOR CARRYING DETONATORS.

commented upon later in connection with the subject of misfires. Formerly, the miners were required to carry into the mine their own detonators, inclosed in small locked boxes that held five or six caps. This was an insufficient precaution, however, against danger from the acts of careless or inexperienced men. It was found that the new men sometimes stored explosives and detonators in the same box, and in some cases attempted to load their own shot holes before the arrival of the shot-firer. To eliminate all dangers that could result from these causes, it was decided to have all of the detonators handled by the shot-firers only. Consequently, provision was made for issuing checks to the miners in place of caps. When the miners called at the supply magazine to purchase their daily stock of caps, they were given checks, each good for one cap. One of the checks was in turn given to the shot-firer each time he fired a charge for the miner.

A convenient type of box for carrying electric blasting caps is shown in Fig. 1. This is the type used by the Ford Collieries Co., and is patterned after the Ellsworth box. It is constructed of $\frac{1}{2}$ -in. boards, is varnished inside and out, fitted with a hinged cover and lock and with small sheet-metal slots through which a shoulder strap is run. The dimensions are such that a du Pont paper box containing 50 exploders will fit snugly inside. Each shot-firer is supplied with two boxes, one being left with the supply clerk to be filled, while the other is carried into the mine for

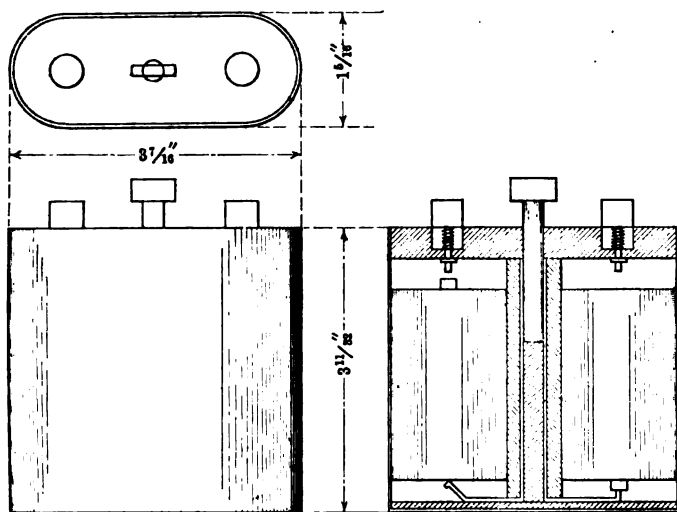


FIG. 2.—SHOT-FIRER'S BATTERY.

use. At the end of the shift, the shot-firer returns the box and a miner's cap check for each detonator taken out of it during the shift.

The lead wire used was about 110 ft. in length, of No. 14 B. & S. gage duplex copper wire. This length was found ample for safety. However, from time to time a short length is apt to be broken from the end nearer the charge and to replace this loss a corresponding length must be spliced on. This may be neglected in some cases, as there is a tendency on the part of shot-firers to carry no more wire than is absolutely necessary. Here is a point of danger, often overlooked, that may well be given attention by safety inspectors.

The battery used consisted of two dry cells inclosed in a suitable container, so arranged that the contacts were made inside and the danger of accidental contact avoided. At the time the writer took up this work, the battery in use consisted of two standard dry cells fitted into a wooden box made of $\frac{1}{2}$ -in. boards. This proved unnecessarily heavy and inefficient. Therefore, Andrew Boland, Chief Electrician of the Ellsworth Collieries

Co., was requested to have a new battery made, to hold two small cells of the type usually employed in flash lamps. He devised the battery illustrated in Fig. 2. This type of battery was found to be especially convenient, for it could be carried in the pocket. Moreover, it was doubly safe because before the circuit was completed it was necessary both to insert the key and press down the two contact buttons.

Clay was used for tamping all shot holes. The usual procedure required that the shot-firer observe the quantity on hand in his section of the mine. When this fell below a sufficient amount the mine foreman was notified. The latter had a mine carload sent in by the night shift to the point designated by the shot-firer. The clay was usually unloaded in a break-through closed by a stopping and conveniently located to a number of working places. Here the miner obtained his individual supply. If the clay proved too dry, enough water was added to make it plastic, in order that it might be worked by hand into roughly cylindrical masses about 1 in. in diameter and 6 or 7 in. in length.

In the case of rock falls, the explosive was as a rule placed on the upper surface of the piece to be broken, and then a shovelful of rather wet clay was packed over the charge before firing.

The miners were all required to provide a tamping rod. This was in all cases obtained by them from among the young trees in nearby woods. It was 6 ft. long and 1 in. in diameter.

In addition to the materials described above and used in the firing of shots, it was necessary, of course, to provide each shot-firer with an approved flame safety lamp, in order to enable him to comply with the law and the rules of the mine, requiring him to test for gas.

Method of Firing

The first step to be taken in an effort to safeguard the firing of shots is the selection of suitable shot-firers. The Bituminous Mine Law of Pennsylvania provides that in the portions of a mine where locked safety lamps are used, "the mine foreman shall employ a sufficient number of competent persons, who are able to speak the English language, to act as shot-firers, whose duty shall be to charge, tamp, and fire all holes properly placed by the miners, and to refuse to charge any holes not properly placed."

To meet these requirements fully, it is essential that the men selected for this work be of the sort who appreciate the responsibility placed upon them, who understand the proper use of explosives, and who are willing and able at all times to keep constantly in mind the fact that the reason for the creation of their positions is the prevention of accidents. It has been found desirable to have men with fire-boss certificates employed as shot-firers, if it is possible to obtain them. There are two good reasons

for this arrangement. First, a man with the certificate is generally competent to undertake the work. Second, if the regular fire boss is unable for any reason to make his run, a man is available who is thoroughly familiar with the regular fire boss's territory, and is able to go in and make the examinations as required by law.

The number of shots that may be fired in one shift depends upon the extent of the section covered, the proportion of narrow places to wide places, and the experience and ability of the shot-firer. Under average conditions, a shot-firer ought to be able to load and fire between 40 and 50

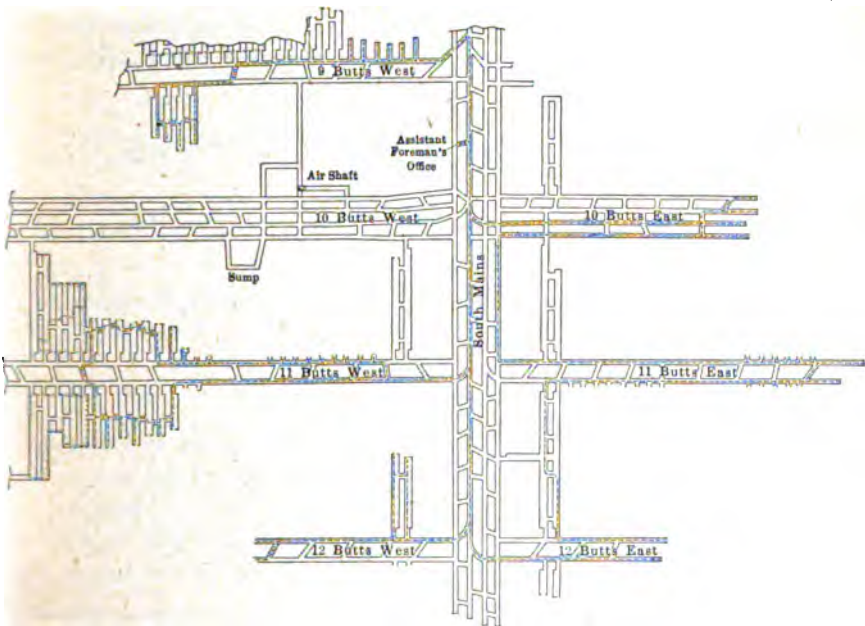


FIG. 3.—PART OF MAP OF ELLSWORTH NO. 2 MINE, SHOWING SOUTH MAIN SECTION.

shots during an 8-hr. shift. This limit may be exceeded where the working places are concentrated; or, on the other hand, it may not be reached by a shot-firer who is required to visit a widely scattered group of narrow working places. Fig. 3 is a portion of the map of Ellsworth No. 2 Mine showing the South Main section. The route of the shot-firer is shown by the dotted line.

Cycle of Operations

In making his rounds, the first act on the part of the shot-firer is to find whether the loader at whose place he has arrived is ready to have his

coal shot down. It is possible to determine this by standing in the entry and calling or whistling to the man at the face, or by going into the working place and making a personal examination. By far the best method is the latter, as it affords more frequent inspection of the working places during the shift, but when a shot-firer must cover a difficult section, it may be impossible for him to go into each place on each round.

Having found that a shot is to be fired, the shot-firer is required to make a thorough examination of the condition of the place. First, the safety lamp is used to make sure that there is no trace of gas present. The roof and sides are examined to see that the required method of posting or timbering has been lived up to and that no additional timbering is required. Furthermore, two 7-ft. posts must be placed as sprags at the face in a room, or one in an entry, as shown in Fig. 4. The miner

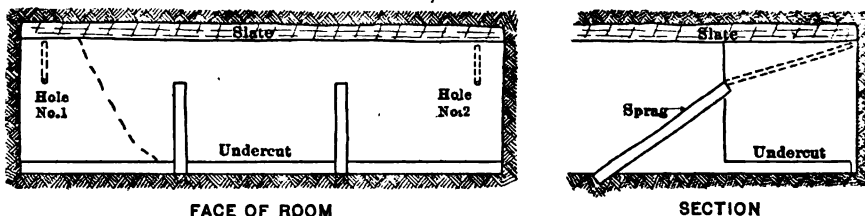


FIG. 4.—LOCATIONS OF SHOT HOLES AND SPRAGS.

must load out all of his slack. The position of the place with respect to other workings ahead or at the sides must be noted, and the men in these places notified that a shot is to be fired. In the case of breaking up a rock fall in an entry, the same precautions must be taken and, in addition, men must be posted at safe distances on both sides of the fall to warn persons that the shot-firer is about to shoot.

The condition of the place having been found satisfactory, the shot-firer next examines the hole that has been drilled into the coal by the miner. First, he inspects it to be sure that the location of the hole conforms to the rules of the mine, these locations being shown approximately in Fig. 4. Second, by using the tamping rod he measures the length of the hole to see that it does not exceed the depth of the undercutting. Third, he examines the hole to see that it is rid of the drill cuttings.

The shot hole is then charged. The miner produces his box of explosive and the shot-firer selects from it the required amount of material, usually two and one-half sticks in the case of a "tight" shot, and from three-fourths to one and one-half sticks for a "butt" shot. The explosive is examined to see that it is in good condition and marked "Permissible." Keeping one full stick out, the others are carefully inserted one at a time and pushed to the back of the hole by means of the tamping rod. A

hole is then made in one end of the cartridge kept out. The hole is made parallel to the axis of the cartridge by using a wooden pin or a small piece of No. 0 copper wire pointed at one end. An electric detonator is then obtained, the 6-ft. iron wires straightened out and the cap inserted in the cartridge, the wires being wrapped once around the cartridge to prevent the cap from being pulled out. The cartridge being thus prepared is inserted in the hole, the end containing the cap going in last. The shot-firer allows the iron lead wires to slowly slip through his left hand while the cartridge is driven carefully in by the tamping rod held in the right hand.

Clay used in tamping the hole having been prepared beforehand by the miner, it is now ready for use. Kneeling before the shot hole, the miner inserts one piece at a time, while the shot-firer, still holding the ends of the lead wires, drives the clay home with the tamping rod, taking care to merely press the first two or three pieces against the explosive, while the rest are solidly packed in until the hole is completely filled.

After a shot hole has been properly loaded and tamped, the shot-firer attaches his cable to the ends of the detonator lead wires. This is done by tightly winding the latter five or ten times around the ends of the cable. Following this, the shot-firer walks slowly away from the charge, uncoiling his cable carefully to avoid breaking the connection between lead wires and cable. Also, to prolong the life of the cable, it is advisable to string it over cars, timber, a pick driven into the rib, or any other means of support that can be utilized to keep it elevated from the bottom and prevent its being buried under the coal about to be shot down. Of course this precautionary measure is found much more useful in the firing of the "butt" shot, but it is considered worth while even in the case of "tight" shots.

All the preliminary steps having been taken, the shot-firer then must select a safe place in which to stand while firing the charge. The place must be out of range of flying pieces of coal and also free from unsafe roof, or else properly timbered. Moreover, there is also the final precaution of seeing that all other persons have been warned and are out of danger and that no person, unaware of the shot-firer's presence, is walking into the danger zone.

Being satisfied that all is in readiness and that the men in adjacent places have been warned, the shot-firer calls out loudly his warning, "Fire!" Then he inserts the key in the battery, presses the ends of his cable against the contact buttons of the battery, pushes down on the buttons, completes the circuit, and fires the charge.

Following the firing of a shot, it is always advisable to wait a minute or two until the ventilating current has cleared away the smoke, and to afford an opportunity to listen carefully for any sound of roof movement.

Then the shot-firer proceeds carefully toward the point at which the shot has been fired, at the same time coiling his cable over the palm of his hand and the back of the upper arm. A careful examination of the roof follows. Then, by means of the safety lamp, any evidence of the liberation of gas is looked for. The face and sides of the place are examined for dangerous slabs that may drop off and injure the miner. Finally, the timbering and sprags are noted to see whether they have been disturbed by the blast. At the completion of his reëxamination, the shot-firer carefully explains to the miner each possible source of danger he has noted and instructs him how to remedy any dangerous condition existing. Or else, in the case of the discovery of any serious danger, the shot-firer fences the place off, fixes danger signals on the fences, and sends the miner affected to report to the mine foreman.

Misfires

Upon completing the circuit when firing with electric detonating caps, the charge may fail to explode for one or more of the following reasons:

1. Defective or exhausted battery.
2. Broken wire or connection or short-circuit due to imperfect insulation.
3. Defective detonating cap.
4. Deteriorated explosive.
5. Detonating cap detached from cartridge.

A battery of the type described in Fig. 2 has been found to give satisfactory results for an average of 600 shots, before it becomes necessary to renew the cells. Therefore, if the shot-firer keeps a record of his shots and the time at which a battery was placed in service, he can readily predict the date at which to expect the failure of his battery. On the other hand, some misfires are caused by imperfect contacts within the battery due to the presence of a small amount of coal dust at contact points. These difficulties can, of course, be avoided by frequent inspection and sandpapering of contacts.

Misfires due to a broken or disconnected, or short-circuited cable are the most frequent of all misfires, and are therefore generally the first suspected following the failure of a charge to explode. Proper care of the cable when about to shoot and again after shooting is quite necessary to reduce this source of misfires to a minimum. Fortunately, in most cases of this kind, the cause is easily discovered and quickly remedied.

In the firing of the 6,020 shots on which this paper is based, only three misfires resulted from defective electric blasting caps, indicating quite

positively the dependability of this method of blasting. In order to avoid any danger from the subsequent handling of these caps, they were destroyed by placing them on the haulage road and running a locomotive over them. When subjected to this treatment, two of the caps failed to explode, while the third was exploded. The conclusions reached were that the first two had not been filled with fulminate, while the other was evidently filled but either the bridge wire or the iron lead wires were broken.

It sometimes occurred that a cartridge of explosive that had deteriorated found its way into a shot hole. Ordinarily the shot-firer may readily detect such cartridges because of the fact that they may be either unusually hard or soft. If, however, such a cartridge is used, the report of the blasting cap may generally be heard quite distinctly, but the charge of explosive fails to detonate. In two or three instances, misfires were the result of blasting caps becoming detached from the cartridge. This was due to the cap working loose during the charging of the shot hole.

In the case of misfires due to either of the first two causes, the remedy lay in the repair or replacement of cable or battery or in reestablishing a broken contact. On the other hand, the last three causes of misfires always involved the drilling of a new shot hole. It was the general practice to circumscribe the hole with a rough circle at least 12 in. in radius and instruct the miner to drill a new hole outside the circle, and parallel to the original hole. In any case, the miner was forbidden to drill out the contents of the misfired hole, and reminded of the dangers of that procedure. When it became necessary to fire a second charge following a misfire, it rarely, if ever, happened that the first charge was exploded. Therefore, the miner was required always to search the coal shot down to locate the cartridges of the charge that failed. These were generally ready for inspection by the shot-firer on his next visit to the working place.

In the firing of the shots discussed in this paper there were no so-called windy shots and only one blown-out shot. The latter was in an entry and was a small charge intended to break down the coal remaining after the coal broken by the tight shot had been loaded out. The blown-out shot was apparently caused both by a charge that was too light and a hole that was slightly in the solid.

Shot-firer's Report

A form of report for shot-firers is illustrated in Fig. 5. It has been found that a report of this kind is useful in checking up the work of the shot-firers and in determining the cause of any difficulties in shooting. Also, it serves as a daily reminder of the necessity for exercising care.

FORD COLLIERIES COMPANY

SHOT-FIRER'S REPORT

MINE	
SECTION No. _____	DATE _____
1. Did you examine each place for gas and other dangers before firing? _____	
2. Did you give warning to persons in adjacent places? _____	
3. How many shots did you fire today? _____	
4. Give location of any place in which you refused to shoot and reason. _____	
5. Misfires.	
Location	Cause
1 _____	_____
2 _____	_____
3 _____	_____
6. Blown-out Shots.	
Location	Cause
1 _____	_____
2 _____	_____
3 _____	_____
7. Did you examine each place for all dangers after shooting? _____	
Mine Foreman	Shot-firer

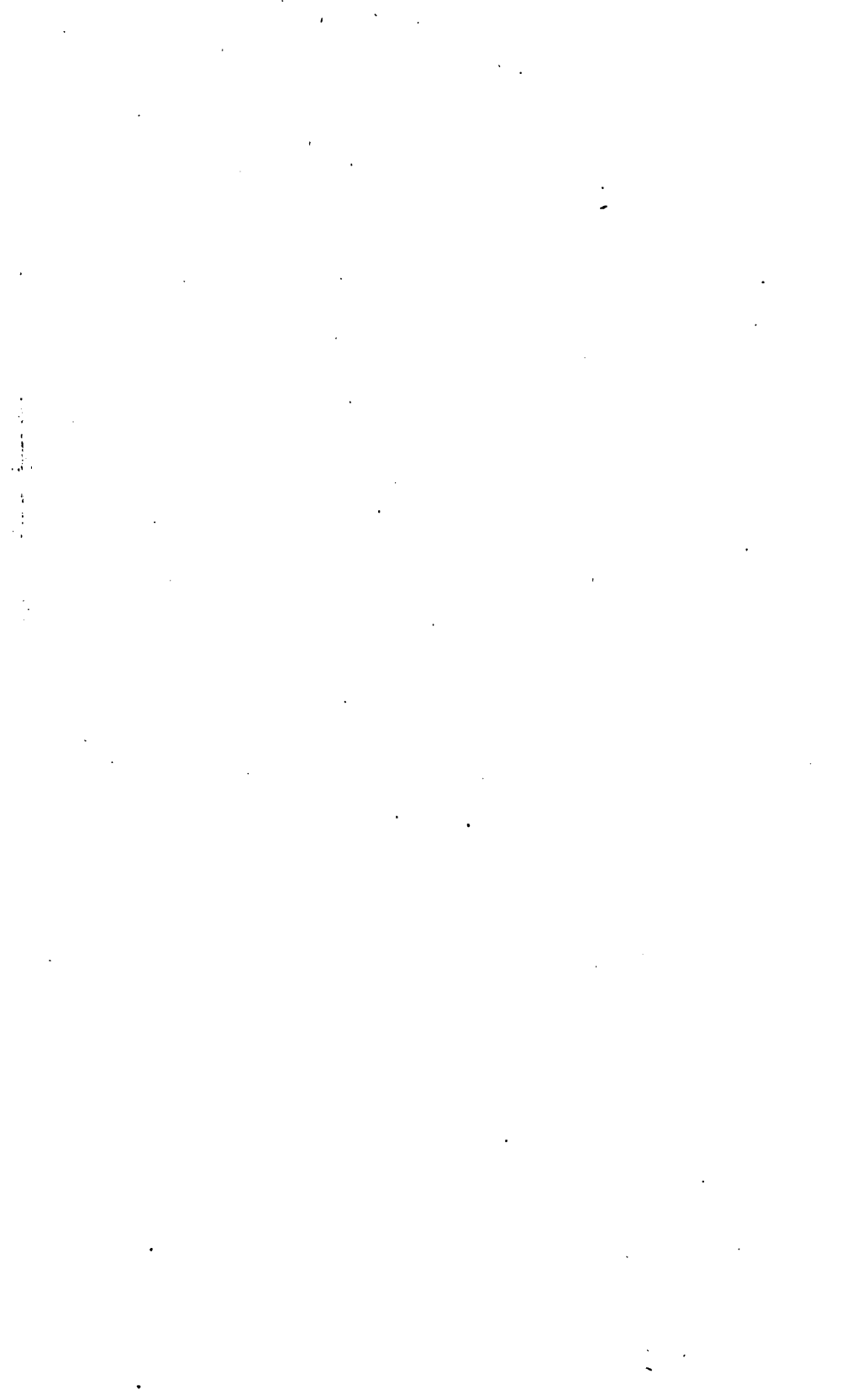
FIG. 5.—SHOT-FIRER'S REPORT.

Precautions in Shot-firing

The most essential precaution to be taken in an effort to safeguard the firing of shots is in the selection of properly qualified men to do the work. Moreover, it is especially desirable that one or two competent men be made available to act in the capacity of shot-firer in case of the absence of the regular incumbent of that position. It is a dangerous practice to place the battery and wire in the hands of a partly qualified man and tell him to shoot during the shift. Nor is it sufficient, as a rule, to assume that because a man was once considered competent to act as shot-firer, he will always continue to be careful in the performance of his duties. It is advisable to issue special instructions from time to time for the purpose of keeping shot-firers alert and informed of any dangerous practices to be avoided.

The importance of giving adequate warning before shooting ought never to be lost sight of by the shot-firer. The best practice is to personally warn all men on all sides within dangerous proximity to the charge, then to call out loudly to warn persons approaching from other parts of the mine, and wait 5 or 10 sec. after calling before firing.

So much has been written concerning the advantages of permissible explosives, that it is unnecessary to enumerate them here. For gaseous mines, they constitute a very important element of safety. To insure their most efficient and safe use, they should be carefully looked after in the magazine outside, examined before being issued to the miners, and transported into the mine in proper containers. Finally, they should be charged and fired by competent persons.



A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron

Discussion of the paper of W. McA. JOHNSON, presented at the New York Meeting, February, 1916, and printed in *Bulletin* No. 110, February, 1916, pp. 233 to 235.*

HENRY M. HOWE, Bedford Hills, N. Y. (communication to the Secretary.)†—Mr. Johnson's explanation, that the rounding of the graphite masses in oxygen-bearing cast iron is due to their being in part re-precipitated after re-solution in forming carbonic oxide temporarily with that oxygen, is certainly most attractive and suggestive. It remains to be shown whether this rounding occurs also when the solidification is too rapid to permit this process to go on. J. E. Johnson might easily test this by casting a single ladleful of his oxygenated iron in masses of varying size. If this present hypothesis is true, then the rounding ought to increase progressively with the size of the casting, and with the distance from the outside, that is to say with the time available for re-solution and re-precipitation.

I question whether a like benefit is to be expected even on this hypothesis, from the oxygenation of steel. The spheroidizing of graphite does good because graphite itself is so weak. But the constituent which can be spheroidized in steel is its cementite, which is a source of strength.

Lamellar pearlite, in which this cementite in effect forms long dendrites, is far stronger than divorced pearlite, in which these dendrites have been broken up into spheroids. Even the surfaces of contact between ferrite and cementite are probably sources of strength because of the amorphous iron which probably fills them.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 111, March, 1916, pp. 627 to 650.

J.W. RICHARDS, South Bethlehem, Pa. (communication to the Secretary‡).—Mr. Howland's paper, data and conclusions have interested me intensely, and have led me to study carefully the record of his 26 furnaces therein given.

Mr. Howland is undoubtedly right in his conclusion that when expressing the amount of carbon burned at the tuyères in percentage of the total

* Also in *Trans.*, vol. 53, pp. 451 to 453 (1916).

† Received Nov. 15, 1916.

‡ Received Nov. 8, 1916.

carbon gasified in the furnace, there is no apparent relation between this percentage and the fuel consumption of the furnace. In the diagram accompanying this discussion the broken line represents the complement of this percentage; that is, the percentage of the carbon gasified which is gasified above the tuyères, or the difference between Mr. Howland's percentages and 100. The irregular character of this percentage is at once evident, showing that there is no simple relation between the percentage of carbon gasified above the tuyères and the fuel consumption of the furnace. It proves conclusively that Grüner's law does not apply either directly or inversely to the working of the blast furnace when the amount of carbon burned at the tuyères and gasified above the tuyères is expressed in *percentage* of the total carbon gasified in the furnace. On this base, therefore, and with this manner of expression, Grüner's ideal working is proved to be neither right nor wrong, but simply no guide at all to the question of fuel efficiency in the furnace.

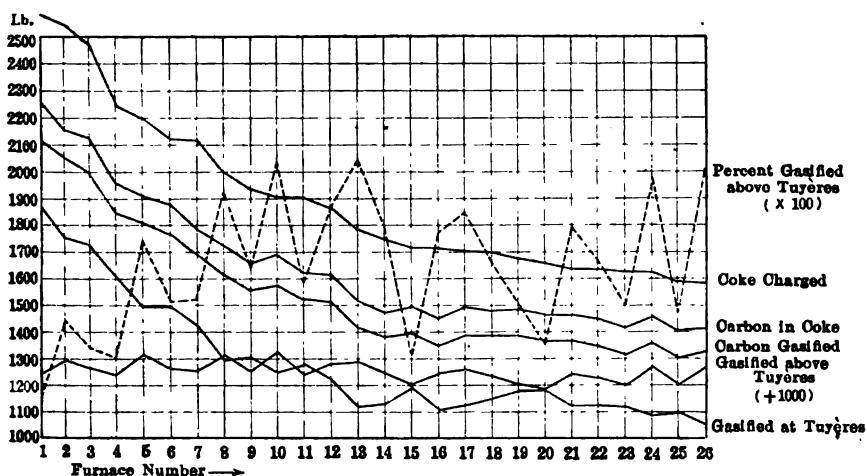
After contemplating for some time this knockdown for a very much discussed theory, the writer was led to plot diagrammatically Mr. Howland's 26 furnaces, and the accompanying diagram shows the amount of fuel charged, the carbon charged, the carbon gasified in the furnace, the carbon gasified at the tuyères and the carbon gasified above the tuyères (+1,000 in order to get it on the diagram), all expressed in pounds of carbon per ton of iron made.

The conclusion to be drawn from this diagram is evident at a glance. While the carbon gasified at the tuyères varies from 1,868 down to 1,057 lb., a variation of 811 lb. per ton of iron made, the line representing carbon gasified above the tuyères, in pounds per ton of iron made, is nearly horizontal, the maximum variation being from 323 to 184, or 129 lb. Of course, if these variations were expressed in percentages of the carbon gasified above the tuyères, they would be enormous; but such an expression is without significance, just as expressing them in percentage of the total carbon gasified appears to be without significance. The really significant fact is, that in these 26 furnaces with a variation in carbon gasified at the tuyères of 811 lb., the variation in carbon gasified above the tuyères is only 129 lb., that is, the latter item is relatively constant in amount, when expressed per ton of iron made.

Furthermore, the variation in pounds of carbon gasified above the tuyères does not coincide with the variations in amount of fuel used in the furnace, but, as inspection of the lines on the diagram shows, is a wavy line about as high at one end as at the other, having its maximum at furnace No. 10 and its minimum at furnace No. 20, but with no observable regularities in rise or fall paralleling even remotely the consumption of fuel in the furnace. It is very evident that Mr. Howland's figures prove that the amount of carbon gasified above the tuyères, expressed in pounds per ton of iron made, is practically constant in all his 26 furnaces, averag-

ing 250 lb. per ton of iron, and that it is unrelated to and independent of the total fuel consumption of the furnace.

A further conclusion from this diagram is that the amount of carbon gasified at the tuyères is closely parallel to the amount of carbon charged into the furnace. We therefore arrive at the comparatively simple view that in all these 26 furnaces, varying immensely in fuel consumption, the weight of carbon gasified above the tuyères is practically constant, while the weight of carbon gasified at the tuyères is practically a function of the total weight of carbon charged; that is, it is necessarily the total carbon gasified minus the relatively constant amount gasified above the tuyères. We therefore have before us the consideration of a *constant* factor in the fuel consumption, namely, the constant weight dissolved or gasified above the tuyères; and the variable amount burned at the tuyères is



simply proportional to the variable amount of carbon charged minus the above-mentioned constant.

This statement would require that the *percentage* of carbon gasified above the tuyères would decrease as the total amount of fuel consumed increased, or would increase as the weight of fuel charged per ton of iron decreased. This is the reverse of Grüner's ideal working, yet it is borne out in a roughly approximate way by furnaces 1 to 10 where the fuel consumption drops rapidly from 2,615 to 1,905 lb. and the percentage of carbon gasified above the tuyères increases rapidly from 11.4 to 20.5. In furnaces 10 to 26, in which the fuel consumption still further decreases, but slowly, no such regularity, even approximate, can be observed; there is practically no observable relation between fuel consumption and this percentage. It would appear from the above discussion that expressing the *amounts* of carbon gasified at the tuyères and gasified above the

tuyères, in *percentages* of the total carbon gasified in the furnace is a useless and misleading operation, and that no definite conclusions of value can be attained by doing it. On the other hand, for furnaces working under approximately similar conditions of ore, fuel, etc., if these amounts are kept in pounds or kilograms per ton of iron made, a very simple relation appears; namely, that the weight of carbon gasified above the tuyères is practically constant, while the weight of carbon gasified at the tuyères varies in parallel with the total carbon gasified in the furnace, that is, approximately with the fuel consumption.

From an inspection of Mr. Howland's data, the amount of carbon gasified above the tuyères in his 26 furnaces does not appear to vary with size of the furnace or the amount of pig iron made per day or with the kind of coke used. It appears to be a practically constant amount for the whole 26 furnaces, and independent of any of the detailed conditions of operation scheduled by Mr. Howland. It is possible that furnaces run with different kinds of ore of different sizes and with entirely different fuel, such as charcoal, would show greater variations in the amount of carbon dissolved or gasified above the tuyères. Such a comparison would be instructive and bring out the factors which really control the solution of carbon above the tuyères. Such a study is much needed to throw further light upon this interesting discussion of blast-furnace phenomena. But, as far as we have gone, we can say that Mr. Howland's splendid paper has proved this one important and illuminating point, viz.: that the solution or gasification of carbon above the tuyères is independent of the fuel consumption, is practically a constant amount in weight per ton of iron made, and does not condition or determine the fuel efficiency of the furnace in usual furnace practice.

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FEBRUARY, 1917



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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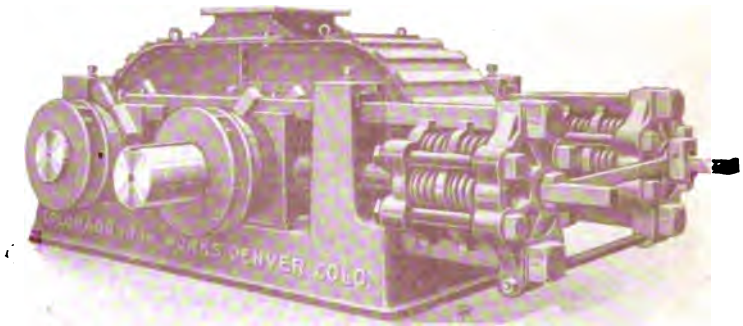
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Sustained Efficiency

NOT mere durability, but continuous satisfactory performance, with repairs reduced practically to renewal of shells. Pounds of steel per ton of ore crushed, uncomplicated by breakage of parts, is all that need be considered where our rolls are used.



48 x 20 Portland Type Crushing Rolls

The frame of our rolls serves mainly to keep the parts in alignment, and as the crushing strain is taken by heavy forged steel bars, Colorado Iron Works roll frames never break.

Our spring pressures are very high, and our rolls ordinarily act like rigid rolls; the springs are there, though, and save the machine from breakage due to excessive shocks.

Study the illustration. Note the simplicity and ruggedness of the design. Material and workmanship don't show in the picture, but high quality exists in the rolls, and that's why they last so long.

Pamphlet 30 shows our large and varied line.

Colorado Iron Works Co.

Established 1860

Denver, Colo.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 122

FEBRUARY

1917

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STOUGHTON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$12 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

TRANSACTIONS—VOL. 54

Vol. 54 will be sent about February 1 to members who have paid their dues for the current year. Members who do not receive copies within a reasonable time are asked to notify the Secretary.

NEW YORK MEETING

ONE HUNDRED AND FOURTEENTH MEETING OF THE INSTITUTE,
MONDAY, FEBRUARY 19, TO THURSDAY, FEBRUARY 22,
INCLUSIVE, 1917

Committee on Arrangements

DAVID H. BROWNE, *Chairman*

LAWRENCE ADDICKS
PERCY E. BARBOUR
GEORGE D. BARRON
KARL EILERS

LOUIS D. HUNTOON
H. A. MEGRAW
THOMAS T. READ
BURR A. ROBINSON
F. T. RUBIDGE

E. MALTBY SHIPP
BRADLEY STOUGHTON
EDWARD B. STURGIS
ARTHUR L. WALKER

Summarized Program

MONDAY, FEBRUARY 19

9.00 a.m. to 5.00 p.m. Registration at Institute Headquarters.
10.00 a.m. Simultaneous Sessions on Geology and on Metallography.

- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 2.00 p.m. Simultaneous Sessions on Petroleum and Gas and on Milling and Smelting.
- 8.00 p.m. Reunion Night.

TUESDAY, FEBRUARY 20

- 10.00 a.m. Annual Business Meeting followed by Technical Session.
- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 2.00 p.m. Simultaneous Sessions on Iron Blast-Furnace Practice and on Flotation.
- 6.30 p.m. Annual Dinner at Hotel Astor, followed by Dancing.

WEDNESDAY, FEBRUARY 21

- 10.00 a.m. Session on the Manufacture of Iron and Steel.
- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 2.00 p.m. Simultaneous Sessions on Metallography and Heat Treatment of Steel and on Mining Methods.
- 8.00 p.m. Entertainment and Lecture at Engineering Societies' Building.

THURSDAY, FEBRUARY 22

All-day excursion by special train to West Point.
(Exhibition horseback riding, inspection of buildings, etc.)

Program for Entertainment of Ladies

MONDAY, FEBRUARY 19

- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 2.30 p.m. Theater Box Party or similar entertainment.

TUESDAY, FEBRUARY 20

- 10.30 a.m. Open Forum on Current Events of the Day.
- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 3.00 to 5.00 p.m. Visit to the Art Galleries of Senator W. A. Clark, with organ recital.
- 6.30 p.m. Annual Dinner at Hotel Astor, followed by dancing.

WEDNESDAY, FEBRUARY 21

- 12.30 p.m. Luncheon at Engineering Societies' Building.
- 2.30 p.m. Entertainment, not definitely settled.
- 5.00 p.m. Visit to Motion Picture Studio to see pictures taken.
- 8.00 p.m. Entertainment and Lecture at Engineering Societies' Building.

THURSDAY, FEBRUARY 22

All-day excursion to West Point by special train as described later.

*Ladies' Committee*MRS. H. W. HARDINGE, *Chairman*MRS. BRADLEY STOUGHTON, *Secretary*

MRS. LAWRENCE ADDICKS
 MRS. GEORGE D. BARRON
 MRS. C. A. BOHN
 MRS. DAVID H. BROWNE
 MRS. ARTHUR S. DWIGHT
 MRS. KARL EILERS
 MRS. LEVI HOLBROOK
 MRS. LOUIS D. HUNTOON
 MRS. AXEL O. IHLSENG
 MRS. JOHN H. JANEWAY
 MRS. SIDNEY J. JENNINGS

MRS. ALBERT R. LEDOUX
 MRS. HENRY S. MUNROE
 MRS. H. A. PROSSER
 MRS. THOMAS T. READ
 MRS. LOUIS D. RICKETTS
 MRS. BURR A. ROBINSON
 MRS. E. MALTBY SHIPP
 MISS MADELINE P. STONE
 MRS. BENJAMIN B. THAYER
 MRS. ARTHUR L. WALKER

Hotel Accommodations.—The Committee on Arrangements has canvassed a number of hotels in New York City, securing, wherever possible, special rates to members and guests for the days of the meeting.

There is an extraordinary congestion in all the hotels in New York this year and most of them are filled up two or three weeks in advance. It will therefore be advisable for out-of-town members to make early reservation for accommodations. If the Committee can be of any assistance to any of the members, it would be glad to hear from them.

Name	Location	Rooms With Bath		Rooms Without Bath	
		Double	Single	Double	Single
Algonquin.....	59 W. 44th St.....	\$3.50	\$2.50		
Astor.....	Times Sq., 43d St. & Broadway.	4.00 up	3.00 up	\$3.00	\$2.00 up
Belmont.....	42d St. & Park Ave.....	4.00 up	3.50 up	3.50 up	2.50 up
Biltmore.....	43d St. & Madison Ave.....	5.50 up	4.00		
Breslin.....	29th St. & Broadway.....	4.00	2.00	3.00	1.50
Grand.....	31st St. & Broadway.....	4.00 up	2.00 up	2.00 up	1.00 up
Hermitage.....	42d St. & Broadway.....	3.00 up	2.00 up	2.00 up	1.50 up
Imperial.....	32d St. & Broadway.....	4.00 up	2.00 up	3.00 up	1.50 up
Knickerbocker.....	42d St. & Broadway.....	5.50 up	3.00 up	4.50 up	2.00 up
McAlpin.....	34th St. & Broadway.....	4.00 up	2.50 up	3.00 up	2.00 up
Manhattan.....	42d St. & Madison Ave.....	5.00 up	3.50		2.50
Martiniague.....	32d St. & Broadway.....	3.50 up	2.50 up	2.50 up	1.50 up
Monticello.....	35 W. 64th St.....	2.00 up	1.25 up		
Navarre.....	38th St. & Seventh Ave.....	2.00 up	1.50 up	1.50 up	1.00 up
Park Avenue.....	32d St. & Park Ave.....	4.00	3.00	2.00 up	1.50 up
Plaza.....	58th St. & Fifth Ave.....	5.00 up	3.50 up		
Prince George.....	28th St. & Fifth Ave.....	4.00 up			2.00 up
Seville.....	29th St. & Madison Ave.....	3.00 up	2.50	2.50	2.50
Waldorf-Astoria.....	34th St. & Fifth Ave.....	5.00 up	3.50 up	4.00 up	2.50 up
Wolcott.....	4 W. 31st St.....	5.00	2.50		2.00 up
York.....	36th St. & Seventh Ave.....		2.00 up		1.50 up

Reunion Night.—The evening of Monday, February 19, has been designated as "Reunion Night." A meeting will be held in the assembly rooms on the fifth floor of the Engineering Societies' Building at 8.30 p.m. Light refreshments and "smokes" will be served. Impromptu addresses will be made, with occasionally moving pictures of a non-technical character, and probably some of the colleges will give "stunts" on a stage provided for that purpose. College songs will be in order. Graduates of the different colleges will be seated together at tables reserved for the purpose and, as many of the more prominent engineers in the Institute are graduates of the College of Experience, a table will

be reserved for them, and it is hoped that a committee of these graduates will have prepared a special "stunt" by them. As the hall in which the festivities will be held holds only about five hundred persons, members must secure in advance tickets for themselves and guests. The tickets may be obtained without charge.

Annual Dinner.—The Annual Dinner will be held at Hotel Astor on the evening of Tuesday, February 20, and at this dinner the presence of ladies is especially urged. Tables are arranged for six to eight persons, but larger tables may be secured by application in advance. Those who do not secure dinner tickets by 2 o'clock on the afternoon of the dinner cannot make special arrangements for tables, but will all be seated together at large tables reserved for that purpose. The dinner will be followed by dancing.

Evening Entertainment and Lecture.—A lecture entitled "The Technicolor Process of Motion Pictures in Natural Colors" will be given on the evening of Wednesday, February 21, in the auditorium of the Engineering Societies' Building at 8.00 p.m. and members and guests will have an opportunity to see the first public exhibition of the technicolor standard films, and to hear an explanation of the projection of moving pictures in natural colors from black and white films.

Excursion to West Point.—A special train over the West Shore Railroad will leave the foot of West 42d St. between 8.00 and 9.00 a.m., on Washington's Birthday, Thursday, February 22. It is probable that arrangements will be made for serving a light breakfast on the train. Upon arrival at West Point the party will proceed to the riding hall where exhibition riding will be given by the cadets, followed by an address by an officer of the post. Luncheon in Cullum Memorial Hall will be served at 1 o'clock, after which opportunity will be given to inspect the buildings and observe the cadets at work. The special train will then convey the party back to New York, arriving about 6.00 p.m. Every effort will be made by the Committee to make this occasion an opportunity for promoting better acquaintanceship among the members, for which no better means exists than an all-day excursion together. Members are especially urged to circulate freely throughout the train during the trip and the equipment will be arranged, as far as possible, to facilitate this.

Luncheons.—Luncheons will be served each day on the fifth floor of the Engineering Societies' Building between the morning and afternoon technical sessions, and luncheon will also be served at West Point. There is no charge for any of these luncheons, as they are provided by New York members of the Institute.

Registration Facilities.—Registration facilities will be available during each day of the meeting. The Registration Bureau will be in the lobby of the Engineering Societies' Building, and there will be additional registration facilities on the train to West Point.

Technical Program

MONDAY, FEBRUARY 19

10.00 a.m. *Session on Geology.*

The Manganese Ores of the Lafayette District, Minas Geraes, Brazil.
By Joseph T. Singewald, Jr., and Benjamin Leroy Miller.

Manganese Ores of Russia, India, Brazil and Chile. By E. C. Harder.

Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba. By Max Roesler.

Recent Geologic Developments on the Mesabi Iron Range, Minnesota. By J. F. Wolff.

The Geology of the Bawdwin Mines, Burma, Asia. By M. H. Love-
man.

Geology and Ore Deposits of Mohave County, Arizona. By Frank C. Schrader.

10.00 a.m. *Session on Metallography*

Grain Growth Phenomena in Metals. By Zay Jeffries.

On Grain Growth. By Henry M. Howe.

Recrystallization after Plastic Deformation. By Henry M. Howe.

Tungsten and Molybdenum Equilibrium Diagram, and System of Crystallization. By Zay Jeffries.

The System Tungsten-Molybdenum. By Frank Alfred Fahrenwald.

2.00 p.m. *Session on Petroleum and Gas*

Problems Connected with the Recovery of Petroleum from Unconsolidated Sands. By William H. Kobbé.

Reservoir Gas and Oil in the Vicinity of Cleveland, Ohio. By Frank R. Van Horn.

Evidence of the Oklahoma Oil Fields on the Anticlinal Theory. By Dorsey Hager.

The Possibility of Deep Sand Oil and Gas in the Appalachian Geosyncline of West Virginia. By David B. Reger.

The Influence of the Movement in Shales on the Area of Oil Production. By Richard A. Conkling.

The Need and Advantages of a National Bureau of Well Log Statistics. By W. G. Matteson.

2.00 p.m. *Session on Milling and Smelting*

Magnetic Concentration of Low-Grade Magnetic Iron Ore. By S. Norton and S. LeFevre.

An Investigation of Rock Crushing Made at McGill University. By John W. Bell.

Countercurrent Decantation. By Luther B. Eames.

The Function of Alumina in Slags. By Carl Henrich.

The Viscosity of Blast-Furnace Slag. By Alexander L. Feild.

Matte Granulation at Herculanum, Mo. By S. Paul Lindau and Henry B. Smith.

TUESDAY, FEBRUARY 20

10.00 a.m. *Session on Non-Metallic Minerals*

(Following Annual Business Meeting)

A Study of the Silica Refractories. By J. Spotts McDowell.

The Genesis of Asbestos and Asbestiform Minerals. By Stephen Taber.

The Conservation of Phosphate Rock in the United States. By W. C. Phalen.

2.00 p.m. *Session on Iron Blast-Furnace Practice*

Dry-Hot versus Cold-Wet Blast-Furnace Gas. By Linn Bradley, H. D. Egbert and W. W. Strong.

Some Suggestions Regarding Construction of Hot Blast Stoves. By Linn Bradley, H. D. Egbert and W. W. Strong.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces. By Henry Phelps Howland.

Potash as a By-product from the Blast Furnace. By R. J. Wysor.

2.00 p.m. *Session on Flotation*

Notes on Flotation—1916. By J. M. Callow.

WEDNESDAY, FEBRUARY 21

10.00 a.m. *Session on the Manufacture of Iron and Steel*

The Seasoning of Castings. By Richard Moldenke.

Significance of Manganese in American Steel Metallurgy. By F. H. Willcox.

Roll Scale as a Factor in the Bessemer Process. By A. Patton and F. N. Speller.

Temperature Measurements in Bessemer and Open-Hearth Practice. By George K. Burgess.

The Manufacture of Weldless Steel Tires for Locomotive and Car Wheels. By Guillaem Aertsen.

2.00 p.m. *Session on Metallography and Heat-Treatment of Steel*

Notes on the Heat Treatment of High-Speed Steel Tools. By A. E. Bellis and T. W. Hardy.

Effect of Time in Reheating Hardened Steel Below the Critical Range. By C. R. Hayward and S. S. Raymond.

Erosion of Guns—The Hardening of the Surface. By Henry Fay.

The Effect of Sulphur on Low-Carbon Steel. By Carle R. Hayward.

A Method for Distinguishing Sulphides from Oxides in the Metallography of Steel. By George F. Comstock.

2.00 p.m. *Session on Mining Methods, etc.*

Anthracite Stripping. By J. B. Warriner.

Portable Miners' Lamps. By Edwin M. Chance.

Shot-Firing in Bituminous Mines. By M. D. Cooper.

The Pennsylvania Mine Fire, Butte, Mont. By C. E. Nighman and R. S. Foster.

Report of the Secretary of the Committee on Safety and Sanitation.

ST. LOUIS MEETING

Although the meeting of the Institute in St. Louis will not occur until September, 1917, the committee in charge is already making attractive

plans, and we append hereto a tentative skeleton program which will give some notion of the opportunities that will be offered. St. Louis is situated in the midst of a district which contains not only a great many interesting examples of mining, but also all types of industrial manufacture of interest to members of the Institute, and it affords abundant opportunities to visit and see these plants. Further announcement will be made from time to time as the committee's plans take more permanent shape.

MONDAY

Morning: Registration.

Afternoon: Technical session.

Evening: Smoker.

TUESDAY

Morning: Technical session.

Afternoon: Boat ride to Herculaneum, Mo., to visit smelter.
and

Evening: Dinner on boat and social evening.

WEDNESDAY

Morning: Visit to various plants in St. Louis.

Afternoon: Technical session.

Evening: Banquet.

THURSDAY

Morning: Special train to Southeast lead district, spending the entire day and returning in the evening.

Evening: Leave by special train for the Southwest district.

FRIDAY

Morning: Strip-pit coal mining district, Pittsburg, Kansas.

Afternoon: Tulsa.

Evening: Dinner. Leave for Joplin.

SATURDAY

In the Joplin district.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, DEC. 22, 1916

Benjamin B. Thayer was elected as representative of the Institute on the John Fritz Medal Board of Award vice Albert Sauveur, whose term expires.

Robert M. Raymond was confirmed as member of the Engineering Foundation Board for two years in accordance with the By-Laws of the United Engineering Society.

The following preambles and resolutions were passed:

WHEREAS, the Chamber of Commerce of the United States is taking a vote, through the Chambers of Commerce, Boards of Trade and other constituent members of the National Chamber situated throughout the country, on the question whether or not the primary natural resources, comprising timber, the ores and deposits of useful metals, and the deposits of minerals, which are a source of heat, light and power, should be exempt through remedial Congressional legislation from the restrictive limitations of the Anti-Trust Acts, and

WHEREAS, a Committee of the National Chamber, the Chairman of which is a Past President of the American Institute of Mining Engineers, has recommended favorable action upon this question, and

WHEREAS, the subject is one which directly affects the moral and economic welfare of the mining industry and the industrial strength of the United States in both peace and war, therefore be it

RESOLVED, That the Directors of the American Institute of Mining Engineers at a meeting held on Friday, Dec. 22, 1916, recommend to the individual members of the Institute, and especially to the Sections located in the United States, that an earnest effort be made throughout the country, and particularly through the various Chambers of Commerce and Boards of Trade, to the end that an affirmative vote may be received by the National Chamber on or before January the 12, 1917, on this important question.

Messrs. Hennen Jennings, Charles D. Walcott and Gardner F. Williams were appointed delegates of this Institute to the meeting of the National Security League in Washington on Jan. 25, 26, and 27, 1917.

The Special Committee appointed to consider the suggestions of the Nominating Committee regarding a change in the Constitution, recommended that the Board shall not initiate a change in the Constitution, but shall call to the attention of the Nominating Committee the steps which may be taken by any 25 members of the Institute to initiate a change and to accompany this communication with a statement that the Board will not object if this action is initiated.

The report of the Treasurer and Finance Committee was accepted and ordered on file.

RESOLVED, That the Institute will be responsible for one-quarter of the deficit on the building additions of the upper three stories of the building.

The budget for the year 1917 was approved.

The sum of \$50 was appropriated to the Treasurer's clerk.

RESOLVED, That this Board approves of the principles and aims of the Civic Federation of Engineers and Scientists as set forth in their communication to the Board.

The Mexican Section of the Institute was established under the name of Instituto Mexicano de Minas y Metalurgia.

New members were elected in accordance with the recommendations of the Committee on Membership.

The dues of seven members engaged with the armies at the front were remitted.

The dues of all members of the Institute residing in the countries at war, and all those other members who had requested an extension of time for payment, were suspended.

The resignations of four members were accepted.

The form of annual ballot was determined as well as the communication to be sent with the ballot.

COMMUNICATION

Lehigh University, South Bethlehem, Pa.,
January 13, 1917.

MR. BRADLEY STOUGHTON, *Secretary*,
American Institute of Mining Engineers,
New York City.

Dear Mr. Stoughton:—

I would be considerably obliged to you and to the Institute if it may be made known to our members that the "J. W. Richards" who signed

the recently-issued circular recommending Mr. Sydney J. Jennings for the Presidency of the Institute is Mr. John W. Richards of Denver.

Yours very sincerely,
Joseph W. Richards

REPORT OF THE LIBRARY COMMITTEE FOR THE YEAR 1916

In accordance with the requirements of By-Law LX, I beg leave to submit herewith the report of the Library Committee for the year 1916.

The activities shown by the sale of the Institute's publications during the year are somewhat larger than last year, due principally to the increased sale of special editions on account of the larger sale of the Posepny and Emmons Volumes, owing to increased advertising and to the reduction in price of the Emmons Volume. The sales were distributed as follows:

Sale of Transactions.....	\$3,237.07
Sale of Special Editions.....	1,722.50
Sale of Bulletins and Pamphlets.....	2,263.95
Total.....	\$7,223.52

The Institute library has now been merged with the libraries of the other three Founder Societies and is known and administered as the Library of the United Engineering Society. When this merger was effected, a proviso was made that the books contained in the Institute library up to the date of the merger were still to be considered as the individual property of the Institute, and this arrangement applied also to the libraries of the other two Founder Societies.

At the time that the American Society of Civil Engineers joined as a fourth Founder Society, they immediately turned over their entire library to the United Engineering Society, relinquishing all property rights in the books entirely.

In view of the changed situation since the first merger, your Committee suggests that it might be well to consider a similar relinquishment of ownership to the books which formerly comprised the library of the American Institute of Mining Engineers.

The addition of the United Engineering Society's library during the year amounted to a total of 2,902 volumes, pamphlets and maps, distributed as follows:

	Volumes	Pamphlets	Total
Gifts.....	795	311	1,106
Exchanges.....	658	13	671
Purchases.....	980	27	1,007
Old material.....	22	35	57
	2,455	386	2,841
Maps.....			61
Total.....			2,902

The activities of the library during the year have been very much increased, the attendance having amounted to 13,848.

At the time the Library Board came into control of the united libraries they found a very large number of duplicate books stored away and unavailable. These duplicates have now all been carefully gone over, and

hidden away amongst them were found quite a number of books which were not duplicates; these have now been added to the shelves of our library. All of the real duplicates were properly listed and sold, the proceeds being added to the endowment fund of the library.

Your Committee takes great pleasure in reporting to you the receipt of a further gift of \$95,000 from our Member, Dr. James Douglas, which gift has been added also to the library endowment fund.

During the year, the Library Board has installed a photostat machine for the reproduction of articles from periodicals, maps and drawings. The necessity for this installation has been shown by the constant work it has been called upon to perform. Since its installation in April last, 4,772 reproductions have been made.

Preparations are now in progress to receive and house the library of the American Society of Civil Engineers, and the actual moving of the books will take place early in 1917.

The Library Service Bureau, instituted less than a year and a half ago, has proved that the service was greatly needed, as it at once became extremely popular. Previous to its organization, the demands for search work in the library amounted to less than \$200 per annum. The past year the receipts have reached approximately \$5,800, received from 589 applicants and classified as follows:

	Since Jan. 1, 1916
Civil Engineering.....	47
Electrical Engineering.....	49
Mechanical Engineering.....	130
Mining Engineering.....	54
Metallurgical.....	46
Chemical.....	95
Miscellaneous.....	74
Total (Searches).....	495
Translations (orders).....	82
Abstracts.....	7
Copying.....	5
Total (Miscellaneous).....	94
Grand Total.....	589

The expenditures for salary and work done amounted to \$5,366. The initiation of this work was made possible by a loan of \$250 from each of three Founder Societies, to be repaid out of any balances existing in the year's operation. I am glad to be able to say that this amount has now been practically earned and is ready to be returned to each of the Societies.

Owing to the amount of extra work which will be entailed by the accession of the 67,000 volumes from the American Society of Civil Engineers, the expenses of the library will be considerably increased, but this will not require any increased appropriation from the Founder Societies as with the additional \$4,000 received from the American Society of Civil Engineers and the \$5,000 interest from the endowment fund these extraordinary expenses will be met.

As required by the By-Law, I attach herewith the stock of publications on hand.

Respectfully submitted,

E. GYBBON SPILSBURY, *Chairman*.

REPORT OF TREASURER, 1916

*Receipts**General Funds:*

Initiation fees.....	\$ 6,008.50
Arrears of dues.....	1,772.69
Current dues.....	46,576.56
Advance dues.....	1,820.97
Sale of binding.....	10,554.43
Sale of advertising.....	6,003.42
Sale of Transactions.....	3,237.07
Sale of special editions.....	1,722.50
Sale of Bulletins and pamphlets.....	2,263.95
Miscellaneous Receipts:	
Interest on investments and deposits.....	480.06
Sale of pins and fobs.....	312.80
For the collection of checks.....	161.84
From Finance Committee, Feb., 1916, Meeting.....	101.02
Subscriptions to the land fund.....	86.14
Arizona Meeting refund from N. Y. Central Railroad..	52.48
Local Section refund from N. Y. Central Railroad....	31.85
Miscellaneous.....	425.68

Total Receipts General Funds.....	81,611.96
Cash balance Dec. 31, 1915.....	1,063.39

\$82,675.35

Special Funds:

Life memberships to be invested.....	\$750.00
Barron Fund to furnish the Member's room.....	637.02
Interest on Hadfield Prize.....	33.53
Interest on Thayer Prize.....	2.99

Total Receipts Special Funds.....	1,423.54
Cash balance, Dec. 31, 1915.....	2,065.84

3,489.38

\$86,164.73

*Payments**General Funds:*

Bulletin.....	\$17,789.60
Year book.....	1,218.72
Transactions.....	8,777.17
Binding.....	7,175.25
Special editions.....	292.73
Editorial and office.....	25,748.06
Treasurer.....	1,009.89
Library.....	4,580.00
Advertising.....	1,989.05
Meetings.....	2,385.01
Local sections.....	2,105.98
Technical Committee.....	84.30
Committee on Increase of Membership.....	1,042.62
Back volumes.....	998.18
Circulars.....	505.20
Miscellaneous.....	3,136.42

Total Payments, General Funds.....	78,838.18
Balance, Dec. 31, 1916.....	3,837.17

82,675.35

Special Funds:

Life memberships invested.....	939.53
Barron Fund to furnish the Members' Room.....	637.02

Total Payments, Special Funds.....	1,576.55
Balance, Dec. 31, 1916.....	1,912.83

3,489.38

\$86,164.73

GEORGE C. STONE, *Treasurer.*

REPORT OF THE COMMITTEE ON MEMBERSHIP FOR 1916

The total number of applications brought before the Committee during the year 1916 was 962; the total number of persons who were elected and became members of the Institute during the same period was 750.

The total membership of the Institute on Dec. 31, 1916, was 5,781, consisting of 19 Honorary Members, 5,246 Members, 215 Associate Members, and 301 Junior Members. The changes in membership during the year are shown on the accompanying schedule:

Total membership, Dec. 31, 1915.....	5,221	
Loss by resignation.....	53	
Loss by suspending.....	125	
Loss by death.....	57	
Total loss during 1916.....	235	4,986
Gain by election.....	750	
Gain by reinstatement.....	45	
Total gain during 1916.....		795
Membership, Dec. 31, 1916.....		5,781
Change of status:		
Associates to Members.....	4	
Junior Members to Members.....	11	
Total.....	15	

KARL EILERS, *Chairman.*

REPORT OF THE COMMITTEE ON INCREASE OF MEMBERSHIP FOR 1916

During the year Jan. 1 to Dec. 31, 1916, 962 applications for membership in the Institute were received, as compared to 558 during the corresponding period in 1915.

The lines along which the Committee has worked during the year group themselves under seven principal heads.

Large Companies.—Early in the year a letter was addressed to each member of the Institute who is the executive head of an organization, pointing out to him the advantages that would accrue to that organization through having as many of its subordinate members as were qualified participating in the activities of the Institute. This was followed by a second letter to those who made no reply to the first. The response to these letters was very gratifying, 12 new members having been secured at a single plant, and smaller numbers at others. This was effectively supplemented by our President, who wrote the executive heads of some of the largest mining and smelting companies, pointing out to them the advantage to be derived from sending some of the junior members of their staff to attend the Arizona Meeting. One organization sent 24 of its younger men, many of whom were not members of the Institute, but who subsequently sent in applications for membership. It is to be hoped that the precedent thus set will be acted on in fuller measure in succeeding years.

Mexican Section.—Early in the year arrangements were consummated whereby the Mexican Institute of Mining and Metallurgy became the Mexican Section of our Institute. It is gratifying to have the relations between the two countries thus strengthened, and when more peaceful conditions are there attained it is reasonable to expect that a large number of new members can be secured in Mexico.

Prospective Members.—Up to the present time this Committee has chiefly busied itself with presenting to the attention of those known to be engaged in mining and metallurgical work, but not members of the Institute, the advantages of membership therein. There are, however, throughout the country, a very large number of men actually engaged in mining and metallurgical work who are not known by any member of this Committee, and possibly not by any member of the Institute. Reaching these men is a difficult task, for even if we were able to obtain their names and addresses we have no personal knowledge that they are qualified for membership, and a general invitation to apply for membership might result in embarrassment, since the Membership Committee rigidly upholds our standards of membership. Much thought has been given to this problem, and its solution has been aided by the Board of Directors by placing at the disposal of this Committee such funds as are required for the effective conduct of its work. Arrangements have been made to provide the Committee with a paid assistant secretary, with headquarters at the Institute building, and a systematic campaign is being made to secure the names and addresses of men actually engaged in mining engineering and metallurgical work who are not members of the Institute, with good results. The securing of personal contact with these men has not been so well worked out. Efforts have been made to secure the aid of local members of the Committee, and of the Secretaries of local sections, and this may perhaps be more effectively done in the future. Meanwhile a more comprehensive plan is under discussion and will be submitted to the Board of Directors for its approval at an early date.

Oil and Gas Men.—Through the courtesy of Messrs. Tinsley and Woodworth, the names and addresses of a large number of men engaged in the oil and gas industry, and who would make desirable members for the Institute, were secured. These gentlemen were invited to join, but the response has so far been somewhat disappointing. The papers and discussions dealing with oil and gas before the Institute are of much importance, and means of bringing that fact home to those engaged in the oil and gas industry should engage our attention during the coming year.

Iron and Steel; Coal.—Many members of the Institute are engaged in the iron and steel industry, but in proportion to the relative numbers of men involved our representation in this field is smaller than it should be. The same may be said of the coal-mining industry. With the coöperation of the Technical Committees for these branches, we hope to secure a steady increase of membership in these fields.

Foreign Members. The growing importance of the Institute is reflected by the increasing number of members who are residents of other countries. The greatest proportionate increase has been in China and Japan, but Canada, Australia, and the South American countries have shown large gains. South America offers an especially attractive field for the work of this Committee.

Total Membership.—Two years ago the membership roll of the Institute passed the 5,000 mark. Since that time over 1,500 applications for mem-

bership have been received, but we have not yet reached the 6,000 mark. This is due to the losses by death, resignation, and those members who, in accordance with the Constitution, are suspended from the rolls for non-payment of dues. The total membership of the Institute on Dec. 31, 1916, was 5,904, but as 130 members were dropped on Jan. 1, 1917, for non-payment of dues, we shall not pass the 6,000 mark until the middle of this year. As the losses through non-payment of dues are heavy it may eventually prove good policy for this Committee to exert itself in order to aid, if possible, in preventing the loss of members through this cause. Another source of loss arises from the fact that only 85 per cent. of those who apply for membership finally becomes members by accepting election and paying their dues. It seems probable that closer coöperation between this Committee and the Local Sections will be of service in meeting these problems.

Respectfully submitted,

THOMAS T. READ, *Chairman.*
W. H. SHEARMAN, *Secretary.*

PRESENTATION OF JOHN FRITZ MEDAL TO PROFESSOR ELIHU THOMSON

On Dec. 8, 1916, at a meeting held at the Massachusetts Institute of Technology, the John Fritz Medal was presented to Professor Elihu Thomson, "for his achievements in electrical invention, in electrical engineering, in industrial development and in scientific research."

Professor Albert Sauveur presided.

John J. Carty, chairman of the Presentation Committee of the Board of Award, made a short address on the history and significance of the medal. He was followed by E. W. Rice, Jr., president of the General Electric Co., who spoke at length of the achievements of Dr. Thomson, whom he called "one of the world's most prolific and industrious inventors and scientists." Among other things, he said:

"In the field of electrical inventions alone, Professor Thomson has been awarded something over 550 United States patents, and in other fields, notably mechanical work, about 150 patents. . . . he has made contributions to the world's scientific and technical literature, in several hundred articles describing not only his own discoveries and inventions but making remarkably interesting contributions to scientific speculation and thought.

"A skilled workman, he is always able himself to do the work which he asks others to perform.

"His inventions are the result of such profound and accurate knowledge, and developed with such skill, that they almost invariably work on the first trial in accordance with expectation."

Dr. Richard C. Maclaurin, President of the Massachusetts Institute of Technology, spoke of the personality of the man as well as of his accomplishments, as follows:

"He has not only done great things himself, but has shown an intense desire to help all who are struggling earnestly with scientific problems. He has proved an inspiration to an ever-widening circle of engineers and others who have entrusted him with their secrets and sought his help in overcoming their difficulties. They have done this knowing that they had only to ask in order to get the full benefit of his

imagination and his power, and that they need have no misgivings that he would take any advantage of their confidence or any credit for their work, for he has no touch of selfishness.

"It can hardly be necessary to say that a man who has achieved what Thomson has done must be more than I have pictured, an unselfish, generous, well-trained, well-rounded, well-balanced man of science. Above all and pervading all must be imagination, not necessarily the imagination of a poet, but something akin to that in quality and in power, and it is, of course, mainly because Thomson is a man of imagination in the highest sense that he has achieved so much success and earned so much respect not only in this country but throughout the scientific world. He has been literally showered with honors and it must be almost a unique thing to obtain two great national medals within almost a week, one from the Royal Society of London, and the other the great honor of the Fritz Medal that is now to be awarded."

The medal was presented by Dr. Charles Warren Hunt, Past-chairman of the Board of Award.

Professor Thomson made a short speech of acceptance, in the course of which he said:

"In the years past I have simply worked on the problems before me—things needed to be done or worthy to be done. I verily believe I could not have helped it if I had tried. Difficulties often arose, but I certainly enjoyed to the full the satisfactions and rejoiced if the general progress was in any degree assisted.

"A final word to the younger men who must be intrusted with the future: If to them the way may at times seem long, the obstacles many, the self-denial and effort extreme, I would say, enthusiasm, the habit of constant thought, never shrinking from seemingly hard tasks, honesty of purpose and the realization that obstacles are often only stepping stones in disguise, will go far to insure coveted success."

COURSE FOR PROSPECTORS

The State School of Mines, University of Utah, is this year giving a four weeks' course for prospectors—Jan. 8 to Feb. 3, 1917—covering the fundamentals of geology, mineralogy, mining and the metallurgical treatment of ores. There will be 36 lectures of 1 hr. each and about 20 laboratory periods, each 3 hr. in length.

It is the purpose of the School of Mines to make the course entirely a practical one, and purely theoretical considerations will be omitted. The laboratory work is designed to supplement the lectures and to give those taking the course a working knowledge of the principles involved.

In order to bring the course within the reach of all, no laboratory fees will be charged, and the only fee will be a nominal one for registration in the University, of \$1.00, but it is expected that those who attend will be genuinely interested in the work.

AMERICAN MUSEUM OF SAFETY

The American Museum of Safety has installed a large collection of exhibits at 18 West 24th St., New York City, and extends a cordial invitation to the members of the Institute to visit this exhibition. The Museum is open daily from 9 a.m. until 5 p.m., and special arrangements will be made for the admission of a large party in the evening. The exhibition covers all phases of industrial and general safety, and also the important questions of sanitation and hygiene.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who called at Institute headquarters during the period Dec. 10, 1916 to Jan. 10, 1917:

F. L. Antisell , Perth Amboy, N. J.	William Kelly , Vulcan, Mich.
F. J. Brulé , Toronto, Ont., Canada.	W. H. Lanagan , San Francisco, Cal.
Zar T. Crittenden , The Pas, Mau, Canada.	S. Lefevre , Forest Glen, N. Y.
Thomas F. Donnelly , Tucson, Ariz.	D. A. Lyle , St. David's, Pa.
John Davenport , Brighton, Mass.	Domingo Nugués , Buenos Aires, Argentine.
E. E. Ellis , Birmingham, Ala.	H. L. Scaife , Clinton, S. C.
N. H. Emmons , 2d, Copperhill, Tenn.	Mortimer A. Sears , Washington, D. C.
W. E. Erdofy , New York, N. Y.	E. Spurny , New York, N. Y.
C. L. French , New York, N. Y.	R. K. Stockwell , Santiago, Chile.
M. H. Gannon , Butte, Mont.	Benjamin F. Tibby , Salt Lake City, Utah.
R. E. Hoffman , Hannibal, Mo.	Arthur P. Watt , St. Francois, Mo.
C. Vey Holman , Rockland, Me.	

Arthur P. Allen has left the employ of the Calumet & Hecla Mine and is now with the Highland Min. Co., Ashcroft, B. C., Canada.

H. C. Bellinger has been appointed general manager of the Chuquicamata mine of the Chile Exploration Co.

Harry B. Byrne has resigned as manager of the Butte office of Paine, Webber & Co., to accept a position in the copper department of Hornblower & Weeks. New York.

Robert A. Bull has been made vice president and general manager of the Chicago Steel Foundry Co.

Milton A. Caine has resumed the position of superintendent of mines of the Tennessee Copper Co., Copperhill, Tenn.

H. B. Carpenter, for the past nine years connected with the Cambria Steel Co.'s By-Product Coke Plant, has resigned to become superintendent of the plant of the Colorado Fuel and Iron Co., Pueblo, Colo.

Henry F. Collins has resigned the executive management of the Huelva Copper & Sulphur Mines to accept the position of consulting engineer.

Edward B. Durham has joined the engineering staff of the Traylor Engineering & Mfg. Co., Allentown, Pa.

C. A. Filteau has resigned as manager of the St. Lawrence Talc Co., to take the management of the National Mines, Ltd., Cobalt, Ont.

F. B. Gleason, formerly in charge of the Western Electric Co.'s Far East business, has been appointed manager of the U. S. Southern District, of the same company with headquarters at Atlanta, Ga.

George B. Holderer has recently been appointed superintendent to the Northern Pyrites Co., Northpines, Ont., Canada.

H. A. Keller has opened a branch office at San Ignacio 25, Havana.

Mark R. Lamb has resigned as manager of Allis Chalmers Mfg. Co. in South America and will be in New York after January. The Santiago office of the company will be closed during most of 1917.

R. H. Lyman, formerly manager of the Seneca-Superior Silver Mines, Ltd., Cobalt, Ont., is now president and managing director of the Elliott-Kirkland Gold Mines, Ltd., also of Cobalt.

E. W. McMurray has been appointed general superintendent of the Rambler property, New Albany, Wyo.

H. D. Richardson has resigned as superintendent of the Boise-Rochester mine, Atlanta, Ida.

A. L. Tuttle has been appointed acting manager of the Tennessee Copper Co., Copperhill, Tenn.

Frederick L. Vahrenkamp has been made managing director and consulting engineer for the Empire Copper Co., Mackay, Ida.

Charles E. Van Barneveld has been appointed supervising mining engineer and metallurgist to the U. S. Bureau of Mines.

H. J. Wallace, formerly field engineer for the Anaconda Copper Min. Co., Boston & Montana Reduction department, has been appointed superintendent of construction.

E. H. Webb has resigned as superintendent for the Pittsburgh Crucible Steel Co., to become superintendent of the Virginia Iron, Coal & Coke Co. blast furnace, Roanoke, Va.

H. M. Wolfen has been placed in charge of the California co-operative work of the U. S. Bureau of Mines and the Industrial Accident Commission.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute of members or other persons introduced by members)

Young mining engineer, and geologist, member, technical, graduate with four years' varied mining experience in the West and Southwest as miner, surveyor and engineer. Past 18 months, work in mine geology, mapping and examinations. Would prefer assisting consulting engineer or work with exploration company in foreign service. Unmarried: Age, 28. No. 331.

Member, aged 35, married, with 10 years' experience in the Southwest, thoroughly competent to take full charge of mining and milling operations. Particularly fitted to handle either gold or tungsten producers. At liberty February 1. Will go anywhere. No. 340.

A member wants a roving commission to examine old mines of the Mother Lode in California, especially those which can be bought for their taxes. Some of these mines are reported to contain large quantities of low-grade ores which can now be treated at a profit. No. 341.

Columbia University, through the Appointments Office, recommends undergraduate and graduate men and women students for positions as com-

panions and tutors and also for all kinds of part-time work. No fee is charged to student or employer. Address Paul C. Holter, A. B., Secretary. (Telephone: Morningside 1400.)

LOCAL SECTION NEWS

CHICAGO SECTION

CHARLES H. MACDOWELL, *Chairman*,

LUTHER V. RICE, *Vice-Chairman*,

HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.

ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

There were 32 members present at the dinner of the Chicago Section which was held at the Chicago Engineers' Club on Friday Evening, December 22.

At the meeting after the dinner, the Chairman called attention to the work of Mr. Hoover in Belgium and spoke briefly of the assistance which could be given him through the Belgian Kiddies, Ltd.

Mr. Alonzo Kinyon read a paper on "Burning Powdered Coal" which so interested the members that many of them participated in the long discussion which followed.

Prof. Harry B. Pulsifier then described the metallurgical plants about Chicago. He mentioned briefly the equipment and interesting features of 30 metallurgical plants of importance and stated that there were also 200 foundries in the district. He said that when number, size, quality and diversity of the plants are considered, the Chicago District is justified in calling itself the greatest metallurgical center on earth. The paper was followed by a brief discussion in which attention was called to some smaller and incipient plants of an interesting character.

HENRY W. NICHOLS, *Secretary*.

NEW YORK SECTION

DAVID H. BROWNE, *Chairman*

PERCY E. BARBOUR, *Vice-Chairman*

ALBERT D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.

C. A. BOHN, *Treasurer*

JOHN V. N. DORR

LEWIS W. FRANCIS

The December Meeting of the New York Section was held at the Machinery Club on Thursday, Dec. 14, 1916. About 25 members attended. The meeting was preceded by a dinner. The speaker of the evening was Mr. Raymond B. Fosdick of the Bureau of Social Hygiene, whose subject was "Policing a Big City."

Mr. Fosdick gave us an interesting talk on the differences between the civic government in European and American cities, drawing his illustra-

tions from the methods of handling police problems. His address proved very instructive and enjoyable to those present.

A. D. BEERS, *Secretary*.

AFFILIATED STUDENT SOCIETY NOTES

During the year 1915-16, the **Mining Engineering Society of the Massachusetts Institute of Technology** had five meetings, at which the following gentlemen addressed the society on the subjects mentioned:

Professor Richards—Memory and the Business World.

Mr. Hutchinson—Mine Examinations.

Mr. Packard—Mining Law, Necessity of Amendment.

Mr. A. H. Rogers—Organization of Mining Property.

Mr. C. Hurter—Explosives.

Mr. W. Macgregor—Organization of a Gold Mine and Mill.

During the year 24 new members were admitted, making the present active membership 54. There was an average attendance of 34 members at the meetings.

HAIG N. SOLAKIAN, *Secretary*, 1915-16.

The **Pick and Shovel Club, Case School of Applied Science**, has elected the following officers for the current year:

President, G. D. Welty,

Secretary, C. E. David,

Vice-President, E. G. Hollman,

Historian, L. S. Kuehn,

Treasurer, C. E. Althouse,

Sergeant at Arms, H. H. Smith.

The club is expecting to have various addresses during the year from men in the metallurgical industries in and around Cleveland, and has already been favored by Professor Van Horn, who spoke on "The Geography of the West and its Mining Camps," and by Mr. Bertram D. Quarrie, Superintendent of the Central Furnaces of the American Steel and Wire Co.

On December 7, the senior and faculty members visited the plant of the Youngstown Sheet and Tube Co. After the tour of the plant was completed, an excellent dinner was served by the company. In the evening, the Case Alumni Association of Youngstown entertained the party at a banquet at the Y. M. C. A. This is the first of several trips that are planned for the year.

In accordance with our policy, Messrs. T. H. Barrett, H. M. Henton and H. R. Walcott, new members of the faculty of the mining and geological departments, have been elected to membership in the Pick and Shovel Club.

CARL E. DAVID, *Secretary*.

The **Students' Organization of the Michigan College of Mines**, together with the M. C. M. Copper Country Club, on December 16 enjoyed an address by Dr. Henry M. Payne, on Alaskan placer mining, which was made doubly interesting by vivid descriptions of the North country. The audience numbered approximately five hundred.

The 3d Vice-President and Treasurer of the Students' Organization are, respectively, M. W. Fronev and E. S. Liston. These names were spelled incorrectly in the list published in the December *Bulletin*.

GALE L. ADAMS, *President*.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy and Chemistry

- EXPLOITATION OF MINERALS IN MYSORE. (Department of Mines and Geology. General Ser. Bull. No. 8.) Bangalore, 1916.
- INTERNATIONAL ENGINEERING CONGRESS. Transactions Vols. I-II and Index. San Francisco, 1915. (Gift of International Engineering Congress.)
- MODERN THEORY OF SOLUTION. By H. C. Jones. New York, n.d.
- TITLES TO MINES IN THE UNITED STATES. By W. A. Harris, London, 1877. (Gift of Bradley Stoughton.)

Geology and Mineral and Metal Production

- GEOLOGY OF CINCINNATI AND VICINITY. Bull. No. 19, Ohio Geological Survey. Columbus, 1916.
- GEOLOGY OF THE NANAIMO MAP AREA. Memoir 51, Department of Mines, Canada. Ottawa, 1914.
- JAHRBUCH FÜR DAS BERG UND HÜTTENWESEN IM KÖNIGREICHE SACHSEN. 1914. Freiberg, 1914.
- LATE PLEISTOCENE OSCILLATIONS OF SEA LEVEL IN THE OTTAWA VALLEY. Museum Bull. No. 24, Department of Mines, Canada. Ottawa, 1916.
- LIGHT OIL FIELDS OF WYOMING. Bull. No. 12, Wyoming State Geologist. Cheyenne, 1916.

- MINERAL RESOURCES OF MYSORE. General Ser. Bull. No. 7, Department of Mines and Geology. Bangalore, 1916.
- NOTA SOBRE A OCCORRENCIA DE UM MINERAL DE NICKEL. By H. E. Williams. Rio de Janeiro, 1916. (Gift of author.)
- NEW JERSEY. Geologic Map. By J. V. Lewis and H. B. Kümmel. 1910-12.
- OIL SHALES IN IMPENDHLE COUNTY, NATAL. Report on. Pretoria, 1916.
- PLEISTOCENE AND RECENT DEPOSITS OF THE ISLAND OF MONTREAL. Memoir 73, Department of Mines, Canada. Ottawa, 1915.
- PRODUCTION OF IRON AND STEEL IN CANADA DURING 1915. Ottawa, 1916.
- PRODUCTION OF SPELTER IN CANADA, 1916. Ottawa, 1916.

General

- AMERICAN RAILWAY ASSOCIATION. Car Shortage Statistics. New York, 1916. (Gift of Association.)
- CARNEGIE STEEL COMPANY. Pocket Companion for engineers, architects, and builders containing useful information and tables appertaining to the use of steel. Ed. 19. Pittsburgh, 1917. (Gift of Carnegie Steel Co.)
- CATALOG OF THE HOPKINS RAILWAY LIBRARY. By F. J. Taggart. Palo Alto, Cal. 1895.
- CONGRESS AND THE SHIPPERS. By G. A. Post.
- HILL, JOHN A., Some of the writings of. (Gift of Power.)
- IRONMONGER DIARY. 1917. London, 1917.
- NAPIER, DAVID, ENGINEER, 1790-1869. An autobiographical sketch with notes. Glasgow, 1912.
- OXY-ACETYLENE WELDING AND CUTTING. By H. P. Manly. Chicago, 1916.
- ROAD MATERIAL SURVEYS IN 1914. Memoir 850, Department of Mines, Canada. Ottawa, 1916.
- U. S. ISTHMIAN CANAL COMMISSION. Annual report of the Governor of the Panama Canal, with maps and diagrams. 1916. Washington, 1916.

Trade Catalogs

- CHICAGO PNEUMATIC TOOL COMPANY. Chicago, Ill. Bull. E-45. Duntley portable electric hoists. Bull. 129. Hose, hose couplings and hose clamp tools. Nov., 1916.
- HANDBOOK OF WESTINGHOUSE WATTHOUR METERS. (Publication No. 5150). (Gift of Westinghouse Electric and Manufacturing Co.)
- INGERSOLL-RAND COMPANY. New York, N. Y.
- Form No. 3130. Air compressors. Ed. 2. Aug., 1916.
- Form No. 8311. "Little David" pneumatic riveting hammers. Oct., 1916.
- JEFFREY MANUFACTURING Co. Columbus, Ohio.
- Bull. No. 200. Storage battery trucks for industrial plants.
- Bull. No. 204. Jeffrey Arcmaster and solenoid switch.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Dec. 10, 1916 to Dec. 31, 1916.

BOWMAN, JOSEPH V., Chem., Andes Copper Mining Co., Chanaral, Chile,
South America.
CALDER, NORMAN L., Min. Geol., Cia. Minera y de Fomento, Watkins, Fomento,
Prov. de Santa Clara, Cuba.
CONRAD, RALPH A., Met. Engr., Magna Plant, Utah Copper Co., Garfield, Utah.
FARR, T. H. POWERS, JR., Asst. Assayer, Argonaut Mining Co., Jackson, Cal.
HAURY, PETER S., Min. Engr., Alta Loma, Cal.
HOULE, ARTHUR, Supt., Shattuck-Arizona Copper Co., Box 2009, Bisbee, Ariz.
LEIB, VICTOR E., Min. Engr., 4204 Ave. D, Austin, Tex.
LINDSLEY, THAYER, Min. Engr., 82 Beaver St., New York, N. Y.
POTTER, OCHA., Supt., Superior Copper Co., Houghton, Mich.
ROCCA, BERNARD T., Engr., Bluebell Mine, Cons. Arizona Smelting Co., Mayer, Ariz.
SAFFOLD, RAY P., P. O. Box 436, Douglas, Ariz.
THOMAS, JAMES EZRA, Asst. Chief, Milling, Roasting, Acid Depts.,
Mineral Point Zinc Co., Depue, Ill.
TOMITA, HARUYOSHI, Met. Engr., Mitsubishi Co., Osaruzawa Mine, Akita-ken, Japan.

Associate Members

ETHERINGTON, JAMES BELCHER, Western Representative,
The Zinc Concentrating Co., 10 Revere St., Winthrop, Mass.
TAUSSIG, GUSTAVE C., Asst. Sec'y, St. Louis Smelt. & Ref. Co., 722 Chestnut St.,
St. Louis, Mo.

Total Membership, Dec. 31, 1916 5,801

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

Applications Lacking Endorsement

Applications for membership have been received from Mr. Gupta and Mr. Sauvajol, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

Dhirendra Chandra Gupta, Sakchi, India.

Proposed by Edwin E. White, Shiv Raj.

1904, Passed Matric., Calcutta University. 1912, B. Sc. in mining and metallurgy, Harvard University, Cambridge, Mass. 1912-13, one term of special study at Mass.

Inst. of Tech. in Dept. of Mining and Metallurgy. 1913, Joined the Tata Iron & Steel Co., Sakchi, India, worked for 1½ years as melter, basic o. h. furnaces, casting house foreman, at gas producers, as stockyard and dolomite calcining plant foreman, 6 months as geologist with same company to investigate the coal, iron and tin resources of Central India, one year with the same company as foreman of the coke ovens department.

Present position: Genl. Foreman, By-Product and Coke Ovens Dept., Tata Iron & Steel Co., Sakchi, India.

Pierre Henri Sauvajol, Montpellier, France.

Proposed by A. R. Ledoux, G. Aertsen.

Born 1873, Montpellier, France. 1895, Grad., Ecole Centrale des Arts et Manufactures, Paris, France. 1895-98, Engr., Cie. de Fives-Ville, France. 1898-1904, Jacob Holtzer et Cie., Manufacturers of Tool Steel, France. 1904-13, Engr., Société d'Automobiles Panhard, Levasor, Paris. 1915, Cons. Engr., Compton Lyon Alémand, Paris. Also Acienes de Firming (Loire) France.

Present position: Consulting Engineer.

The following persons have been proposed during the period Dec. 10, 1916, to Dec. 31, 1916, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

C. Kremer Bain, Butte, Mont.

Proposed by J. L. Bruce, C. E. Bushnell, H. J. Stiebel.

Born 1889, Lexington, Ky. 1896-1904, Graded schools, Lexington, Ky. 1904-06, Lexington High School. 1906-07, College of Mechanical Engng., State University of Kentucky. 1907-11, College of Mines and Metallurgy, State University of Kentucky, B. E. M. 1910, Transitman, Kentucky Geological Survey. 1911, Engr., Consolidation Coal Co., Van Lear, Ky.; Chief Engr., Roundup Coal Mining Co., Roundup, Mont. 1911-18, Supt. of Construction and in charge of operations at Carpenter Creek Mine, Carpenter Creek, Mont. 1916, Engng. Dept., Butte & Superior Min. Co., Butte, Mont.

Present position: Asst. to Mine Supt., Butte & Superior Mining Co., Butte, Mont.

William A. Baker, Jr., Baltimore, Md.

Proposed by Joseph T. Singewald, Jr., William Bullock Clark, Raymond A. Walter.

Born 1893, Baltimore, Md. Public Schools, Baltimore. 1911, Grad., Baltimore City College. 1911-15, Johns Hopkins University, geology and chemistry, A. B. 1914-15, summers, construction work, Lock Joint Pipe Co. on tunnel of Baltimore City water supply. 1916, summer, Geologist, Maryland Geological Survey, engaged in work on coal report, including property map.

Present position: Asst. Geol., Maryland Geological Survey; Graduate student, Johns Hopkins Univ., Dept. of Geology.

William Turnley Bates, Mascot, Tenn.

Proposed by J. N. Houser, H. B. Henegar, C. B. Strachan.

Born 1888, Trenton, Ga. 1906-07, Peacock School for Boys, Atlanta, Ga. 1907-09, Student, Georgia School of Technology, Atlanta, Ga. 1909-11, Alabama Polytechnic Institute, Auburn, Ala., B. S. in E. M. 1911-12, Rodman and Levelman, Tennessee Coal, Iron and R. R. Co., Min. Division, Birmingham, Ala. 1912, Transitman in Engng. Dept., land and mining work, Tennessee Coal, Iron and R. R. Co., Birmingham, Ala. 1913, Resident Engr., Woodward Iron Co., Woodward, Ala.

1914, Draughtsman and Asst. Engr., Elkhorn Min. Corpn., Fleming, Ky. 1915, Engr., American Zinc-Co. of Tenn., Mascot, Tenn.

Present position: Shift Foreman, American Zinc Co. of Tenn.

Arthur E. Bellis, Springfield, Mass.

Proposed by Albert Sauveur, H. M. Boylston, Thomas W. Hardy, Jr.

Born 1889, Waltham, Mass. 1913, Mass. Inst. of Technology, B. S. 1913-14, Instructor, Mass. Inst. of Tech. 1915, Chemist, Watertown Arsenal.

Present position: Met., N. E. Westinghouse Co.

George Nelson Bennett, Butte, Mont.

Proposed by J. L. Bruce, H. J. Stiebel, C. E. Bushnell.

Born 1886, Deer Lodge, Mont. 1900, Graduated from Grade School, Helena. 1900-04, High School, Helena. 1905, Grad., Prep. School Notre Dame Vne. 1909, B. S. and E. M., Michigan College of Mines. 1909-10, Engr., Copper Queen Mining Co., Idaho. 1910-11, V. S. Deputy Mineral Surveyor, Helena. 1911-13, Supt., Black Friday Mine, Radersburg, Mont.

Present position—1913 to date: Engr., Butte & Superior Mining Co., Butte, Mont.

George Raymond Boggs, Los Angeles, Cal.

Proposed by C. Colcock Jones, Seeley W. Mudd, Alvin B. Carpenter.

Born 1883, Stockton, Kans. 1908, Sheffield Scientific School, Yale University, Metallurgy. 1908-09, Supt., Old Peacock Mine, Copper Butte Min. Co., Helena, Idaho. 1909-11, Cons. and operation, G. M. Weeks Co., Chicago, Ill. 1911-15, General consulting and operation.

Present position—1915 to date: Representing Miami Products Co.; Mgr., Western Min. and Sales Co.

Grenville Temple Bridgman, Chuquicamata, Chile.

Proposed by Fred Hellmann, Henry Hay, Huntington Adams.

Born 1881, Newport, R. I. 1904, Yale University. 1908, Mass. Inst. of Technology, S. B. in Mining. 1909, Asst. Engr., Cerro de Pasco; Asst. Engr., Esperanza Mining Co., El Oro, Mexico. 1910-11, Mgr., San Mauricio Gold Mining Co.; Philippine Exploration Co. 1911-14, Mgr., American Zinc Co. of Tennessee. 1914-16, Ashton & Bridgman, Engineers, San Francisco, Cal.

Present position: Acting Assistant General Manager, Chile Exploration Co.

Frederic Eric Butcher, San Francisco, Cal.

Proposed by Howard D. Smith, T. A. Rickard, F. W. Bradley.

Born 1877, Wapakoneta, Ohio. 1901, B. Ph., Ohio State University. 1901-07, Foreign Sales Mgr., Kilbourne & Jacobs Mfg. Co., Columbus, Ohio. 1907-14, Genl. Mgr., American Concentrator Co. 1915, St. Louis Smelt. & Ref. Co., St. Louis, Mo. 1916, Secy., Mine La Motte Co., St. Louis.

Present position: Secretary, Missouri Metals Corporation.

Dave P. Carlton, Red River, N. Mex.

Proposed by G. H. Clevenger, H. W. Young, W. F. Dietrich.

Born 1890, Mantua, Ohio. 1915, Leland Stanford, A.B. 1909-10, Mission Inn (Mr. F. A. Miller), Riverside, Cal. 1915-16, Geol., Standard Oil Co. of California, Whillier-Fullerton Fields.

Present position: Geol., Memphis Red River Mining Co., Raton, N. Mex.

Malcolm W. Clark, Tulsa, Okla.

Proposed by H. F. Wright, Jerry B. Newby, W. E. Dodge.

Born 1888, Charlevoix, Mich. 1907-16, Academic education, Carleton College, Northfield, Minn., B. S. 1912-15, Technical education, Minnesota School of Mines, E. M. 1909-10, Compass man, Scott Graphite Co., Duluth, Minn. 1911-12, Instrument man and inspector on construction of Dock No. 1, Duluth and Iron Range R. R., Twoharbors, Minn.; also summer, 1910, on maintenance. Summers, 1912-13, Engr. and miner, Olive Iron Min. Co., Eveleth and Ely, Minn. Summer, 1914, Instrument man, Gypsy Oil Co.

Present position—1915 to date: Asst. Geol., Gulf Oil Co., Mex.; Asst. Geol., Gypsy Oil Co. of Okla.

Reginald Elliott Clark, Bartlesville, Okla.

Proposed by Kurt Stock, K. C. Fraser, Harold C. Price.

Born 1885, London, Eng. 1893-1902, Ardingly College, Sussex, Eng. 1909-11,

University of Michigan, special work in Geology and Mineralogy. 1903, Jessop & Co., Calcutta, India. 1904-09, "Simmer & Jack" Last Rana Property, "Geduld" Mines in Transvaal. "Cam & Motor," "Globe & Phoenix," "Jumbo" and several other mines in Southern Rhodesia, as assayer, surveyor, supt. and so forth. 1912, Supt., United Min. & Mill. Co., Cadis, Okla. 1912-16, Chief Chem., Bartlesville Zinc Co., Collinsville, Okla.

Present position: Mech. Engr., Bartlesville Zinc Co.

Frank Jesse Coleman, Globe, Ariz.

Proposed by John Langton, C. E. Mills, George R. Lehman.

Born 1871, Beatrice, Nebr. Chemistry and general, Polytechnic Institute, Pasadena, Cal. 1899-1900, Assayer and Chemist, Detroit Copper Min. Co., Morenci, Ariz. 1901-13, Asst. Mine Supt., Detroit Copper Min. Co., Morenci, Ariz. 1915, Shift Boss, Mine.

Present position: Asst. Mine Foreman, Inspiration Cons. Copper Co., Miami, Ariz.

Walter James Eaton, Santa Ana, Cal.

Proposed by Frank E. Lathe, C. T. Durell, S. R. Brown.

Born 1888, Riverton, Nebr. 1909, Grad., Santa Ana High School, Santa Ana, Cal. 1913, Colorado School of Mines, E. M. 1913-14, Sample "Bucker," Asst. Assayer and Chemist, Granby Cons. Min., Smelt. and Power Co., Grand Forks, B. C. 1914, Supt., Union Mine, Grand Forks, B. C. 1915, Engr. and Assayer, Imperial Reduction Co., Ogilby, Cal. 1916, Assayer and Engineer, Plymouth Cons. Gold Mines, Plymouth, Cal.

Present position: Private examinations.

Frank Travers Eddingfield, Santo Domingo, R. D.

Proposed by Edward A. Blanton, Jr., W. L. Saunders, Geo. A. Howells.

Born 1884, Danville, Ind. 1906, Grad., Columbia School of Mines, E. M.; Asst. Supt., El Cobre Copper Mines, Cuba. 1907, Mgr., Deer Lodge Cons. Min. Co., Mont. 1908, Boss, Cyanide Plant, Dolores Mines Co., P. O. Madera, Mex. 1909, Asst. to W. T. Griswold in mapping West Virginia oil strata around West Union. 1909-14, Min. Engr., Philippine Government, Div. of Mines, Bureau of Science. 1915, Mgr., Blanton Copper Min. Syndicate, Santo Domingo, Dominican Republic.

Present position: Genl. Mgr., Blanton Copper Mining Syndicate.

Edward Hyndman-Ford, Peckville, Pa.

Proposed by Thos. M. Righter, R. V. Norris, T. A. Stearns.

Born 1882, Allegheny City, Pa. 1901, Grad. Harry Hillman Academy, Wilkes-Barre, Pa. 1907, Cornell University, Degree M. E. A. C. S., C. E. Course.

1½ years, Coal Dept. D. L. & W. R. R., Scranton, Pa. 2½ years, Eng. Dept., U. S. Forest Service Experimental work in mine timbering, Tenn. Coal, Iron and R. R. Co., Birmingham, Ala. 4 years Singer Mfg. Co., New York, in charge field construction work.

Present position: Supt., Mt. Jessup Coal Co.

George T. Geringer, Manila, Philippines.

Proposed by E. H. Clausen, Chas. C. Selbie, C. M. Eye.

Born 1881, Cincinnati, Ohio. 1903, B. A., St. Xavier College, Cincinnati, Ohio. 1904, M. A., St. Xavier College, Cincinnati, Ohio. 1910, Min. Engr., Colorado School of Mines. Summer, 1908, Golden Cycle Min. Co., Cripple Creek, Colo., miner. Summer, 1909, Miner, Portland Gold Min. Co., Cripple Creek, Colo. 1910-11, Genl. Mgr., Imperial Coal Min. Co., Wilhurst, Ky. 1912, Asst., J. Gordon Hardy; Deadwood Mines, Mogollon, N. Mex.; Socorro Mines, Mogollon, N. Mex.; Inde Min. Co., Inde, Durango, Mex., Colorado Min. Co., Aroroy, Masbate, P. I.; Keystone Min. Co., Aroroy, Masbate, P. I. 1913, Syndicate Min. Co.; Aroroy, Masbate, P. I.

Present position—1914 to date: Supt., Keystone Min. Co., Supt., Alabat Min. Asso., Manila, Philippines.

Alva Arthur Hammer, Muskogee, Okla.

Proposed by Ralph Arnold, L. K. Armstrong, J. C. Ralston.

Born 1882, Calfax, Wash. 1903-07, Grad., Dept. of Geology, Washington State College, Pullman, Wash., B. Sc. 1907, U. S. Topographical Survey; Chemist, Washington Portland Cement Co., Concrete, Wash. 1908, Washington Portland Cement Co. Later with Superior Cement Co. of the same place. 1909, Superior Cement Co.

1910-12, working on cement materials, Washington Geological Survey. 1912, Consultation and Chemist, International Cement Co., Spokane, Wash. 1912-15, Geol., Caribbean Petroleum Co., Venezuela, Company headquarters, Land Title Bldg., Philadelphia, Pa.

Present position: Geol., Oil and Gas Inspector, Bureau of Mines.

Alexander Smith Harvey, Los Angeles, Cal.

Proposed by Frank A. Keith, Philip Wiseman, Robt. E. McConnell.

Born 1869, Scotland. 1874-79, Public Schools, Colorado Springs, Colo. 1879-85, Public Schools, Leadville, Colo. 1885-86, High School, Glasgow, Scotland. 1886-87, Business College, Denver, Colo. 1887-89, Rider, Cattle & Horse Ranch, Colo. 1890-93, Clerk, Cary Hardware Co., Leadville, Colo. 1893-94, Time-Keeper, Ibex Mining Co., Leadville, Colo. 1894, Prospecting, British Columbia. 1894-95, Time-Keeper, Ibex Mining Co., Leadville, Colo. 1895-96, Foreman, Ibex Mining Co., Leadville, Colo. 1897-99, Mining, own account. 1900, Sec., Home Min. Co., Leadville, Colo. 1901 to date, Mining, own account.

Present position: Mining, own account.

Roy Rhodes Hornor, Salt Lake City, Utah.

Proposed by D. A. Lyon, Oliver C. Ralston, R. S. Lewis.

Born 1876, West Virginia. 1891-92, West Va. Wesleyan College. 1894, Lehigh Preparatory. 1895-99, Lehigh University, B. S. (Met). 1899-1900, Columbia School of Mines, E. M. 1900-02, Mexican Copper Co., Ramos, Mexico. 1902-03, Compania Minera de Penales, Mapini, Mex. 1903-06, Cons. Gold Fields, Ltd., Johannesburg, S. A. 1906-07, Buatson Copper Mines, Latuche Island, Alaska. 1907-16, Examinations and Reports in Northwest and British Columbia.

Present position: Mining Eng., U. S. Bureau of Mines.

John Reginald Horsley, Karagandy, Akmolinsk Province, Siberia.

Proposed by E. T. McCarthy, Herbert C. Woolmer, C. J. Hall.

Born 1882, Travancore, India. 1899-1902, Graduated at the Royal School of Mines, London, obtaining the Diploma of Associateship in Mining (First Class) A. R. S. M.; A. Inst. [M. M. 1902-03, Surveyor, Van Mines Ltd., N. Wales, Gt. Britain; Windsor Colliery Co., Ltd., S. Wales. 1903-04, Assayer & Surveyor, Minnie Moore Mng. Co., Idaho, U. S. A., and Tonopah Extension Mng. Co., Nevada and other properties for C. M. Schwab, N. Y. 1905, Demonstrator Mine Surveying, Royal School of Mines, London. 1905-09, Mine Supt., Mynbouw Maatschappij "Loemar," Dutch West Borneo. 1909-11, Mine Supt., Troctsk Goldfields Ltd., Russia. 1911-12, Asst. Manager, Troctsk Goldfields, Ltd., [Russia. 1912-14, Manager, Troctsk Goldfields, Ltd., Russia.

Present position: Resident Manager, The Spassky Copper Mine, Ltd., Siberia.

Cecil B. Hull, Butte, Mont.

Proposed by C. E. Bushnell, Paul A. Gow, J. L. Bruce.

Born 1885, Kearney, Nebr. 1900, Grammar Schools, Denver, Colo. 1903, Manual Training High School, Denver, Colo. 1909, Colorado School of Mines, E. M. 1908, 3 months' surveying, Great Western Sugar Co., Denver, Colo.; 5 months' tramming, Portland G. M. Co., Victor, Colo. 1909-10, Chem., Colorado Fuel & Iron Co., Sunrise, Wyo. 1910, Sampling, assaying, and making maps of Mary Murphy G. M. Co., Romley, Colo.; assaying and making mill test of Gophar M. & M. Co., Wallstreet, Colo. 1910-12, Min. Engr., Denver, Colo., surveying, sampling, examination work. 1912, Mill foreman, Big Reserve Min. Co., Breckenridge, Colo. 1912-13, Chem., Colo. Fuel & Iron Co., Sunrise, Wyo. 1913-15, Min. Engr., Mary Murphy G. M. Co., Romley, Colo. 1915-16, On my own lease, Madoma Mine, Monarch, Colo.

Present position: Engr., Butte & Superior Min. Co.,

Kyosuke Iwai, Nomigun, Ishikawaken, Japan.

Proposed by R. M. Keeney, W. G. Haldane, G. J. Young.

Born 1881, Chibaken, Japan. 1900-03, the Tokyo Higher Technical College. Took Applied Chemistry course and graduated. 1907-09, Colorado School of Mines, Golden, Colo. Took Metallurgical Engineering Course and graduated with a degree of E. Met. 1905-07, Chemist, Illinois Steel Co., 1909-10, South Works, So. Chicago, Ill. 1910-11, Visited Met. Plants in Europe. 1911-12, Professor, Waseda University, Tokyo, Japan (Metallurgy). 1912-16, Metallurgist, The Takenchi Mining Co., 31 Akashicho, Tsukiyi, Tokyo, Japan. 1916, Gen. Supt., Yusenji Copper Mine of the Takenchi Mining Co., Kokubununa, Nomigun, Ishikawaken, Japan.

Present position: Gen. Supt., Yusenji Copper Mine, Kokubununa.

R. C. Jacobson, Kingman, Ariz.

Proposed by T. D. Walsh, L. W. Wickes, R. L. Cornell.

Born 1880, St. Paul, Minn. 1902-04, School of Mines, University of Arizona. 1904-06, Minnesota School of Mines. 1907-10, Assayer, Amalgamator, Mill Supt., Assist. Supt., Minnesota Arizona Mines Co. 1911-12, Consulting Engineer—Civil and Mining—Arizona South Western Copper Co., Yucca, Ariz. 1912-16, Partner, Mohave Assay and Eng. Office, Kingman, Ariz.; Manager, Leviathan Mines Co., Yucca, Ariz. Present position: Mgr. Hualapai Metals Co.; Consulting Chemist and Engineer, Emerald Isle Copper Co., Kingman, Ariz. and Arizona South Western Copper Co., Kingman, Ariz.; Partner, Mohave Assay and Engineering Office in charge of Mining and Metallurgy.

Erling Lossius Jørgensen, Chuquicamata, Chile, S. A.

Proposed by E. A. Cappelen Smith, Paul H. Mayer, Wm. K. Page.

Born 1881, Trondhjem, Norway. 1903, Chem. Engr., Trondhjem, Norway. 1905, Dipl. Rütteningenieur, Freiberg, Saxony. 1906, Chem. and Met., Baltimore Copper Smelt. & Refining Co., Baltimore, Md. 1910, Supt., Leaching and Acid Plants, Braden Copper Co., Rancagua, Chile, S. A. 1915, Met. Engr., Guggenheim Bros., New York, N. Y., in charge of Tank House, Smelter, Acid Plant, etc., of Chile Exploration Co., Chuquicamata, Chile, S. A.

Present position: Mr. E. A. Cappelen Smith's assistant in Chuquicamata.

Chokichi Kato, Tokyo, Japan.

Proposed by Shinji Harada, M. Yamashita, F. Fuketa.

Born 1887, Saitama Prefecture, Japan. 1894-1902, Elementary School; 1902-07, Middle School, Tokyo, Japan. 1907-10, Higher Middle School. 1910-13, Mining and Metallurgical College of Tokyo Imperial University, Tokyo, Japan. 1913, Graduated, the whole course of the University. August, 1913, Engaged by Mitsu-Bishi-Goshi-Kwaisha (Mitsu-Bishi Company) as a Mining Engineer. Since then with the Takatori Mine (Copper, Tungsten Mine of that Co.) until October, 1915. From Oct., 1915, up to the present time, in the head office of that company in Tokyo as an engineer of Engineering Bureau of Metal Mining Department.

Present position: Mining Engineer to Metal Mining Department, Mitsu-Bishi Co.

Charles T. Kirk, University, Albuquerque, New Mexico.

Proposed by James C. Ray, Charles P. Berkey, C. K. Leith.

Born 1876, Francisco, Indiana. 1904, B. S., University of Oklahoma (Geology). 1905, A. M., Same. 1911, Ph. D., University of Wisconsin. 1910-13, Asst. Prof. Geol., Hunter College, N. Y. City. 1910-13, Copper Co., Oscar Rohn, Mgr., Butte, Mont. 1903-11, Five seasons, intermittently, with U. S. Geol. Surv., hydrography, statistics of metals and division of mineral fuels. 1906-08-09, Instr. Geol., Mont. Sch. of Mines, Butte, Mont. Petrographic study, Pittsmt mine, East Butte Copper Co., Butte. 1908-16, Examinations for metals, coal, and oil and gas in Rocky Mts. and Mid. Continent. Present consulting geologist for Albuquerque and Cerrillos Coal Co., Albuquerque, N. M.; Continental Oil & Gas Co., Bartlesville, Okla.; Keiser Mining Co., Albuquerque (and Scholle) N. M.; John W. Wilson Co., Albuquerque, N. M.

Present position: Prof. Geol., Univ. N. Mex., Albuquerque; State Geol. of N. Mex., Albuquerque.

Julius Koebig, Los Angeles, Cal.

Proposed by C. Colcock Jones, Philip Wiseman, S. W. Mudd.

Born 1855, Mettlach, Germany. 1874, Chemistry and Min. Engrg., Karlsruhe, Germany. 1878, Strassburg, Germany, Ph. D. 1874-79, Teaching Chemistry and Mineralogy, Stuttgart, Strassburg, Germany. 1879-81, Chemist in charge Frankfurter Aniline Farben Fabrick, Germany. 1881, Geological survey, European American Tunnel Co., Gilpin Co., Colo. 1882-1902, Cons., Chemical and Min. Engr. with headquarters in San Francisco.

Present position—1902 to date: Cons. Chemical and Min. Engr. in Los Angeles, Cal.

Ambrose Joseph Mandell, New York, N. Y.

Proposed by Wm. Campbell, Robert Peele, James T. Kemp.

Born 1895, New York, N. Y. 1914, B. S., Columbia University. 1916, M. E., Columbia University. 1916, Chem., Cananea Cons. Copper Co.; Engr., Electrical Alloy Co., Morristown, N. J.; Chem., Iron Cap Copper Co., Copper Hill, Ariz.

Present position: Engr., Iron Cap Copper Co., Copper Hill, Ariz.

Bartley F. Noehl, Kingman, Ariz.

Proposed by T. D. Walsh, L. Webster Wickes, Roy L. Cornell.

Born 1880, Minnesota. 1907, Degree of E. M. from Minnesota School of Mines. 1907-08, Timberman, Greene-Cananea C. C. Jan.-June, 1908, Mechanic and Engineer, Rosario Mt. M. Co. July-Dec., 1908, Underground Foreman, Esmeralda Mining Co., Santa Eulalia, Chihuahua, Mexico. Mar.-July, 1909, Underground Foreman for Yaguivo Development Co., Yaguivo, Chih, Mex. July, 1909-May, 1910, Supt. for Jos. S. Qualey & Co. May, 1910-July, 1911, Supt. and Examining Engineer for Jos. S. Qualey & Co. July, 1911-Mar., 1912, Asst. Supt. and Engineer, Lake Superior & Nevada Development Co. May, 1912-Jan., 1913, Mining Engineer, Los Angeles, Cal. Jan., 1913-Aug., 1914, Geol. Carribean Petroleum Co. Aug., 1914-Mar., 1915, Supt. Metal Mines Development Co. Mar., 1915-Nov., 1915, Mining Engineer, Kingman, Ariz. Nov., 1915-Nov., 1916, Supt., Tyrogoll Mines Co.

Present position: Examination of Mines, etc.

Antenor Rizo Patron, Huaracaca, Peru, So. Amer.

Proposed by Edwin S. Berry, George P. Bartholomew, Frederick Hellmann.

Born 1867, Lima, Peru. 1887, Grad., Escuela de Ingenieurs of Lima, Peru, Min. and Met. Engr. 1888, Empresa Minera de Carahuacra, Taulu. 1892, The Backus and Johnston Co., Casapalca. 1899, Genl. Mgr., Ne gociacim Minera de Eulogio E. Fernandini, Cerro de Pasco.

Present position: Mgr., E. F. Fernandini.

Walter A. Perkins, Chuquicamata, Chile.

Proposed by Fred Hellmann, Henry Hay, Huntington Adams.

Born 1886, Los Angeles, Cal. 1903-05, University of Utah. 1905-06, University of Southern California. 1906-07, Surveying, A. S. & R., Garfield, and N. C. C. Co., McGill, Nev. 1907-08, Nev. Cons. Copper Co., Ely, Nev. 1908-10, Drill Supt., Nev. Cons. Copper Co., Ely, Nev. 1910-11, Construction work, Braden Copper Co. 1911-12, Supt. Drill Operations, M. Guggenheim's Sons, Coquimbo, Chile. 1913-15, Asst. to Genl. Mgr., Chile Exploration Co.

Present position: Business Mgr., Chile Exploration Co.

Thomas Reed Pickard, Anaconda, Mont.

Proposed by Frederick Laist, Louis V. Bender, R. B. Caples.

Born 1886, Chicago, Ill. 1905, Finished High School. 1911, E. M., Houghton School of Mines. 1911-16, Testing Dept. and laboratory of Anaconda Copper Mining Co.

Present position: Chemist.

Robert James Piersol, Salt Lake City, Utah.

Proposed by J. M. Callow, Ernest Gayford, E. R. Lidvall.

Born 1890, Marion, Kans. 1908, B. E., Pennsylvania S. W. State Normal. 1912, A. B., Allegheny College. 1913, M. S., Tulane University. 1916, Ph. D., University of California. 1912-13, Teaching Fellowship in physics, Tulane. 1913-14, Tyndale Research Fellow in physics, University of Pennsylvania. 1914, Head of Physics Dept., Summer Schools, Chautauqua, N. Y. 1914-15, Tyndale Research Fellow in physics, Penna. 1915-16, Asst. in physics, University of California.

Present position: Research in Flotation, General Engineering Co.

Gerald Mungo Ponton, Belleville, Ont., Canada.

Proposed by E. M. Sawyer, Donald G. Miller, Stephen L. Kaffer.

Born 1888, Belleville, Ont., Canada. 1897-1904, Belleville Public and High Schools. 1904-09, University of Toronto, Faculty of Applied Science. Summer, 1906, Concentrator, Craigmont, Ont. Summer, 1907, Concentrator, Coniages Mine, Cobalt, Ont. Summer, 1908, Prospecting, New Ontario. 1909, Asst. Engr., Trent Valley Canal, Frankford, Ont. 1910, Engr., West Canadian Collieries, Blairmore, Alberta. 1911-14, Partner "Harrison & Ponton," surveyors, mining and civil engineers, Calgary and Edmonton, Alberta. 1915, partner "Associated Engineers" Calgary, Alberta. 1916, Burro Mountain Copper Co., Tyrone, N. Mex.

Present position: Tunneling officer, Canadian Expeditionary Force.

Lester William Schnell, Red River City, N. Mex.

Proposed by G. H. Clevenger, H. W. Young, W. F. Dietrich.

Born 1891, Port Townsend, Wash. 1915, Stanford, A.B. 1909-10, Santa Fe

R. R., Raton, N. Mex. 1913, Engng. Dept. I. C. C., Balboa, I. P. 1915, Memphis Red River Mining Co., Raton, N. Mex.

Present position: Supt., Memphis Red River Mining Co., Red River, N. Mex.

Robert Schubert, Stavanger, Norway.

Proposed by Jos. W. Richards, K. Landgrebe, A. W. Allen.

Born 1887, Phila., Pa. 1894-1902, Public Schools of Phila., Pa. 1902-05, North-East Manual Training High School, Phila., Pa., graduated. 1907, Evening Course in Mechanics at Franklin Institute, Phila., Pa. 1909, Correspondence Course in Metallurgy from Dr. Albert Sauveur, Cambridge, Mass. Rest of education obtained by home studies in Chemistry, Physics, Metallurgy, etc. 1906-10, Midvale Steel Co., Phila., Pa., as Chemist. 1910, Bethlehem Steel Co. in Open Hearth Dept. 1911, Charge of Heat Treatment with Bergdoll Motor Car Co., Phila., Pa. 1912-15, American Steel & Wire Co. at Worcester, Mass. & New Haven, Conn., as Metallurgist. 1916, Illinois Steel Co., Gary, Ind., as Metallurgist in Open Hearth Dept. Inspecting Open Hearth Operations for Robert W. Hunt & Co. at Ensley, Ala.

Present position: Sailing on Dec. 23 for Stavanger, Norway, to accept position as Metallurgist with Stavanger Steel Co.

Julius Segall, Minneapolis, Minn.

Proposed by Hugh M. Roberts, E. J. Longyear, John E. Hodge.

Born 1884, New York, N. Y. 1914, University of Wisconsin, B. S. in Geology. 1915, University of Minnesota, N. A. 1906-08, Engr., Oliver Iron Min. Co. 1909, Geological Work for Mr. Fargo of American Express. 1910, Geological exploration for Longyear & Hartley. 1911-13, E. J. Longyear Co., Minneapolis, general geological for Longyear & Hartley. 1911-13, E. J. Longyear Co., Minneapolis, general geological work. Summer, 1914, E. J. Longyear Co. Summer, 1915, Adbar Devt. Co., geological work. Summer, 1916, Independent geological work.

Present position: Geologist, private work.

Leslie M. Sheridan, El Paso, Tex.

Proposed by John Vandemoer, W. J. Deavitt, R. F. Manahan.

Born 1882, Detroit, Mich. 1903, B. S., Mechanical Engineering, University of Montana. 1903-04, Draftsman, Mine Surveyor, The Alder Co., Virginia City, Mont. 1904-06, Smelter Draftsman, Anaconda Copper Min. Co., Anaconda, Mont. 1906-09, Draftsman, Engr. on Smelter Construction, Nevada Cons. Copper Co. 1909-11, Asst. Engr., Northwest Steel Co., Portland, Ore. 1911-13, Asst. Mech. Supt., Anaconda Copper Min. Co., Anaconda, Mont. 1913-15, Mech. Supt., Grandby Cons. Min. Smelt. & Power Co.

Present position: Asst. Chief Engr., Mex. Dept., Amer. Smelt. & Ref. Co.

John T. Shimmin, Butte, Mont.

Proposed by J. L. Bruce, Scovill E. Hollister, H. J. Stiebel.

Born 1886, Salt Lake City, Utah. General and practical education. 1905-06, Utah Copper Co. 1906-07, Federal Min. & Smelt. Co. 1907-11, Utah Copper Co. 1911-12, Ray Cons. Copper Co. 1912, Utah Copper Co.

Present position: Mill Supt., Butte Superior Mining Co.

Nelson Wood Sweetser, Hadley, Alaska.

Proposed by L. K. Armstrong, S. Shedd, Francis A. Thomson.

Born 1882, Peoria, Ill. Preparatory work, Peoria High School and Bradley Polytechnic Institute. 1906-10, College Work at Washington State College in Mining Eng., Degree B. S. 1916, Degree E. M. 1902-06, Miner in various mines of British Columbia. 1910-16, Worked for Granby Co., as Engineer and Supt.

Present position: Supt. of Mamie and It Mines, for Granby Consolidated Mining Smelting and Power Co.

Chesley Covington Thornburg, Felton, Cuba.

Proposed by S. B. Patterson, Jr., W. M. Shoop, De B. Whitaker.

Born 1890, Nashville, Tenn. 1896-99, Private School, So. Bethlehem, Pa. 1899-1907, Moravian Parochial School, Bethlehem, Pa. 1907-11, Lehigh University, So. Bethlehem, Pa., Graduated as Civil Engineer. June-Aug, 1906, Construction Dept. Bethlehem Steel Co. June-Sept., 1907, Machinist's Helper, Bethlehem Steel Co. June-Sept., 1908, Instrument man for T. H. Riddle & Co., So. Bethlehem, Pa. July-Nov., 1911, Levelman, Buffalo Div. Lehigh Valley R. R., Buffalo, N. Y. Nov.,

1911, Rodman for Allegheny Div. Pennsylvania R. R. 1912, Oil City, Pa., under Div. Engr. Apr.-Oct., 1912, Transitman for City of Oil City, Pa. 1912-13, Rodman for Pennsylvania R. R., Oil City, Pa. 1914-16, Transitman and Asst. Engr., So. Penn. Oil Co., Oil City, Pa. May-Sept., 1916, Transitman for H. Koppers & Co., Jersey City, N. J.

Present position: Civil Engr. with Spanish Amer. Iron Co.

Fred Warren Varney, Frisco, Utah.

Proposed by P. M. McHugh, A. L. Bloomfield, E. R. Ramsey.

Born 1885, Minneapolis, Minn. 1900-01, Hackley Manual Training School, Muskegon, Mich. 1904-05, Michigan College of Mines, Houghton, Mich. 1908, Michigan College of Mines, Houghton, Mich. 1902-04, Construction Foreman, B. C. Riblet Tramway Co., Spokane, Wash. 1906, Bridge Foreman, Nevada Northern R. R. Co., Ely, Nev. 1906-08, Hoist Engr., Mine Engr., Shift Boss, National Development Co., Salt Lake City, Utah. 1909-10, Supt. of Construction, Intersection Min. & Mill. Co., Silverton, Colo. 1910-11, Mill Supt., Idaho Cons. Min. Co., Bellevue, Idaho. 1911-12, Mill Supt., Veta Colorado Min. & Mill. Co., Parral, Mexico. 1912, Mill Shift Boss, Compania de Real del Monte y Pachuca, Pachuca, Mexico. 1913, Supt. of Construction, Argo Mill, Idaho Springs, Colo. 1914, Assaying, Aurora, Nev.

Present position—1915 to date: Shift Boss (mill), Caldo Min. Co.

Walter Washabaugh, Jones Store P. O., Spotsylvania Co., Va.

Proposed by J. S. Grasty, Thomas T. Read, Wm. H. Shearman.

Born 1870, Chester, Pa. 1890, Degree Civil Engr., Pennsylvania Military College. 1890-92, Instrument man on railroad work. 1892-93, Second Asst. Engr., Homestake Min. Co., Lead City, S. D. 1893-1903, Electric railway and lighting work. 1903-04, Resident Engr., R. F. & P. Ry. 1905-06, Asst. Engr., Southern Ry. 1907-08, Chief Engr., Va. Air Line Ry. 1909-10, Genl. Supt. construction, Central Georgia Power Dam, Jackson, Ga. 1910-14, Chief Engr., Greensboro, Northern & Atlantic Ry.

Present position: Supt. and Engr., Valzinco Mines, Virginia Lead & Zinc Corp'n.

Edmund Wendel, Cleveland, Ohio.

Proposed by Zay Jeffries, J. Burns Read, Frank R. Van Horn.

Born 1886, Cleveland, Ohio. 1892-99, Cleveland Public Schools. 1903-05, Central Institute, Cleveland. 1906-08, Case School of Applied Science. 1912-13, B. S., Case School of Applied Science. 1909, Oliver Min. Co., Iron Mountain, Mich., Mines; Mill man, Calumet & Hecla Min. Co., Lake Linden, Mich. 1910, Chem., Chemical Laboratory, National Carbon Co., Cleveland, Ohio. 1913-14, Testing Dept., Anaconda Copper Co., Anaconda, Mont.

Present position—1914 to date: In New York City in charge of third rail work and at present closing experimental work on cast copper welding and brazing of copper bands.

Berkley Williams, Richmond, Va.

Proposed by J. H. Batcheller, J. S. Grasty, Thomas T. Read.

Born 1878, Richmond, Va. 1889-95, McGuire's School, Richmond, Va. 1895-97, University of Virginia. 1897-1900, Asst. to Supt., Richmond Traction Co., Richmond, Va. 1900-03, Purchasing Agent, Railways & Light Co. of America, Richmond, Va. 1904-13, Banking and Brokerage, Middendorp, Williams & Co., Baltimore. 1913-15, Active Vice-Pres. with active supervision operations, The Alabama Co., Birmingham District. 1915, Member firm John L. Williams & Sons, Richmond, Va.; Pres., Virginia Lead & Zinc Corp'n.

Present position: Member firm John L. Williams & Sons; Pres., Virginia Lead & Zinc Corp'n.; Pres., Richmond Mica Co.

Thomas M. Williams, Dover, N. J.

Proposed by W. L. Saunders, George A. Howells, L. D. Albin.

Born 1853, Nelson, Glamorganshire, Wales. 1876-84, In charge of Schofield Mine, Nova Scotia, of the Crane Iron Co. 1884-88, In charge of mines, of the Copley Iron Co. of New Jersey. 1884-93, In charge of Bristol Mines in Canada. 1897-99, In charge of mines, Shenandoah Furnace Co., Virginia. 1899, In charge of Hurd Mine, New Jersey. 1901, In charge, Arnold Mines, New York. 1901-17, General Examination work.

Present position: Consulting Engr.

George F. Williston, Hackberry, Ariz.

Proposed by T. D. Walsh, L. W. Wickes, Roy L. Cornell.

Born 1886, Laramie, Wyo. 1901-05, Denver Public Schools, West Side High School, Denver, Colo. 1906-08, Colorado College (Engineering). 1907, Summer, Rodman, Instrument Man. 1905-06, Colo. Fuel & Iron Co., Trinidad, Colo. 1908-09, Rodman, Levelman, Topographer, Denver, Laramie & Northwestern Ry. 1910 until May, Asst. Engr., Northern Coal Co., Denver, Colo. 1910-11, Instrument Location and Construction, Denver Laramie & Northwestern Ry. 1912, Concrete Construction and Design, Medino Valley Irrig. Co., San Antonio, Tex. 1913, Supt. Construction, Laramie Water Co., Laramie, Wyo. 1914, General Engineering Deesz & Williston, Deming, New Mex. 1915, Churn Drilling, Mine Sampling, Exploration Work with H. E. Crawford. 1916, Engineer and Mill Supt., Nevada Arizona Mines Co.

Present position: Mill Supt., Nevada Ariz. Mines Co.

*Associate Members***Frank Edward King, New York, N. Y.**

Proposed by Arthur Feust, Richard Tarantous, Eugene L. Steindler.

Born, 1867, Dewitt, Ill. 1875, Illinois College. 1878, Golden University. 1890, Pueblo Smelt. & Ref. Co., Pueblo, Colo. 1895, Chem. and assayer, Mexico. 1900-13, Mine owner and operator in Mexico.

Present position: Oil Producer.

Harry Christian Paske, El Paso, Tex.

Proposed by John Vandemoer, W. J. Deavitt, R. F. Manahan.

Born 1885, Cincinnati, Ohio. 1906-07, Civil Engineering, International Correspondence School, Scranton, Pa. 1911-13, Cincinnati University, Cincinnati, Ohio. 1905-12, School, Surveying, Railroad Construction and Drafting. 1912-13, Structural Engr. and Supt., Elzner & Anderson, Archts., Cincinnati, Ohio. 1913-14, Engr., Highland Coal & Lumber Co., Parkersburg, W. Va., development and surveys. 1914-15, Asst. to City Engr., structural and architectural, El Paso, Tex. 1915-16, Draftsman and Supt. of Construction, Burro Mountain Copper Co., Tyrone, N. Mex.

Present position: Engr. and draftsman, Amer. Smelt. & Ref. Co.

*Junior Members***Harrison Morton Bray, State College, Pa.**

Proposed by W. M. Weigel, W. R. Crane, C. E. McQuigg.

Born 1889, Alden Station, Pa. 1905, Wanamie High School. 1907, Bloomsburg State Normal School. 1912-14, summers, Laborer, Alden Coal Co., Alden Station, Pa. 1912, winter, Laborer, Markle & Co., Jeddo, Pa. 1915-16, summers, Engrg. Corps, Lehigh Valley Coal Co., Hazleton, Pa.

Present position: Senior Student in Min. Engrg., Pennsylvania State College.

George M. Cheney, Golden, Colo.

Proposed by Harry J. Wolf, W. G. Haldane, George J. Young.

Born 1887, North Adams, Mass. 1902-06, Williamstown High School. 1906-10, Williams College, A. B. 1913-14, 1915-16, Colorado School of Mines. 1910-13, completed student course and worked in shops and offices, Western Electric Co., Chicago, Ill. 1914-15, Vindicator Cons. Gold Min. Co., Victor, Colo.

Present position: Student, Senior class, Colorado School of Mines.

Max M. Dixon, Brooklyn, N. Y.

Proposed by Robert Peele, R. M. Raymond, Homer L. Carr.

Born 1890, Oakland, Cal. 1908-13, Princeton Univ., L. B.

Present position: Student, Columbia School of Mines.

Walter Harry Freudenberg, Rolla, Mo.

Proposed by Chas. Y. Clayton, H. T. Mann, H. A. Buehler.

Born 1893, St. Louis, Mo. 1913-15, 1916-17, Special Student, Missouri School of Mines and Metallurgy. June-Oct., 1915, Mining, Anaconda Copper Co. Oct., 1915-Aug., 1916, Millman, Butte & Superior Mining Co.

Present position: Student, Missouri School of Mines and Metallurgy.

Arthur Meyn, Reno, Nev.

Proposed by Francis C. Lincoln, J. C. Jones, J. G. Scrugham.

Born 1892, San Francisco, Cal. 1909-11, visited but not registered at Univ. of Cal. Summer, 1916, Tonopah Belmont Dev. Co., Tonopah, Nev.

Present position: Student Mackay School of Mines.

John Kennedy Walsh, Rolla, Mo.

Proposed by Chas. Y. Clayton, H. T. Mann, H. A. Buehler.

Born 1895, St. Louis, Mo. 1906-10, St. Louis Academy, St. Louis, Mo. 1910-14, St. Louis Univ., St. Louis, A. B. (Arts & Science). Summer of 1916, Underground Geology Work, Geol. Dept., Detroit Copper Co., Morenci, Ariz.; Drill Runner and Timberman, Ryerson Mine, Detroit Copper Co., Morenci, Ariz.

Present position: Student Mo. School of Mines.

Earl Joseph Welmer, Rolla, Mo.

Proposed by C. R. Forbes, A. K. McRae, H. A. Buehler.

Born 1892, Moline, Ill. 1897-1905, Grammar School, Cambridge, Ill.; Viola, Ill. 1905-08, High School, Viola, Ill. 1912-15, Kansas School of Mines, Weir, Kansas. 1908-11, Miner, Coal Valley Mining Co., Mathersville, Ill. 1911-12, Miner, Weir Coal Co., Pittsburg, Kans. 1915, June, July and August, Miner, Empire Mining Co., Galena, Kans. 1916, June, July and August, Miner, American Zinc, Lead & Smelting Co., Cartersville, Mo. 1915, Surveyor, Roy & Millner Coal Co., Weir, Kansas.

Present position: Student, Missouri School of Mines, Rolla, Mo.

John Hezekiah Winchell, Jr., Golden, Colo.

Proposed by Harry J. Wolf, W. G. Haldane, Geo. J. Young.

Born 1892, Le Mars, Iowa. 1911, Manual Training H. S., Denver, Colo. Summer, 1912, Juniper Coal Co., Oak Creek, Colo. Summer, 1915, Granite Gold Mining Co., Victor, Colo.

Present position: Student, Colo. School of Mines, Golden, Colo.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Dec. 10, 1916 to Dec. 31, 1916.

This list together with the list published in Bulletin Nos. 110 to 121, February, 1916, to January, 1917, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Jan. 1, 1916 and brings it up to the date of Dec. 31, 1916.

ALEXANDER, CURTIS	Spring Valley, San Diego Co., Cal.
ALLEN, CHESTER A.	117 W. Armstrong St., Peoria, Ill.
BARBOUR, PERCY E., <i>Engineering & Mining Journal</i> ,	10th Ave. & 36th St., New York, N. Y.
BAUMANN, ALBERT P.	228 E. 52d St., New York, N. Y.
BELL, ALAN D.	11 Marchhall Crescent, Edinburgh, Scotland.
BETTERTON, JESSE O.	3018 Hamilton St., Omaha, Neb.
BLOOMFIELD, EDWIN C., Canadian Engineers Training Depot,	Crowborough, Sussex, England.
BOOTH, EVERETT L.	478 W. 158th St., New York, N. Y.
BOUERY, PIERRE, Care Usine Bouery, Veyssyere, Rue Kleber,	Levallois-Perret, near Paris, France.
BOYLE, HARRY	1713 Smith Bldg., Seattle, Wash.
BRIDGE, SAMUEL D.	201 Clinton Ave., San Antonio, Tex.
BRUÉ, F. J., British American Nickel Corporation, Ltd.,	Royal Bank Bldg., 8 King St., E., Toronto, Canada.
BURGER, CLARENCE C.	20 Fifth Avenue, New York, N. Y.
BURNS, JAY J.	Vernon Mining Co., Ironton, Colo.
CALLOW, JOHN M.	159 Pierpont Ave., Salt Lake City, Utah.
CAPLES, RUSSELL B.	Box 651, Great Falls, Mont.
CHATIN, AUGUST H.	509 W. Northern Ave., Pueblo, Colo.
CLARK, GEORGE L., Thompson Krist Min. Co.	Box 607, Timmins, Ont., Canada.

POTTER, CHARLES F.	841 Monadnock Bldg., San Francisco, Cal.
REDFEARN, ALBERT M.	146 Belmont Ave., Detroit, Mich.
RICKARD, EDGAR	Room 2124, 120 Broadway, New York, N. Y.
RIGBY, WILLIAM A.	Feldspars, Ltd., Harrington R. R. L., Ont., Canada.
ROBERTSON, JASPER T.	Kenton Mine, Alleghany, Sierra Co., Cal.
SALAZAR, L.	Calle de El Oro N. 11, Mexico City, Mexico.
SCHUBLE, MAX C.	Cia Carbonifera Agujita y Anexas, S. A., Sabinas, Coah., Mexico.
SCOTTEN, FRANK C.	Holter, Mont.
SEAMON, W. H., JR.	Miami, Ottawa Co., Okla.
SINN, A. HENRY	Swiss Consul, Kristiania, Norway.
SMITH, LLOYD B.	824 South Detroit Ave., Tulsa, Okla.
SMYTHE, H. V., Genl. Supt. of Mines, Nichols Chemical Co., Ltd.	St. James St., Montreal, Canada.
SNELLING, WALTER O.	208 N. 5th St., Allentown, Pa.
SOMERS, RANSOME E., Asst. Prof. of Economic Geology, Cornell University,	Ithaca, N. Y.
STEWART, MELVILLE B., Min. Engr.	Lumaghi Coal Co., Collinsville, Ill.
STONE, LAWRENCE H.	Richard Mine, Thomas Iron Co., Wharton, N. J.
TIBBY, BENJAMIN F.	Min. Engr., 178 E. St., Salt Lake City, Utah
WAKABAYASHI, YAICHIRO, The Mitsubishi Kwaisha, Inc., No. 1, Ichi-chome,	Yaesumachi, Kojimachi-ku, Tokio, Japan.
WALTERS, M. B.	Hold everything.
WELLS, RALPH E., JR.	Am. Smelting & Refining Co., Murray, Utah.
WESTERVELT, EDWARD W.	1545 St. Paul St., Rochester, N. Y.
WILLIAMS, PERCY T.	1801 California St., San Francisco, Cal.
WILMOT, H. C.	Andes Exploration Co., Arequipa, Peru, South America.
WOLFLIN, HUGH M., Industrial Accident Commission, Underwood Bldg.,	San Francisco, Cal.
WRAMPMEIER, ERNEST L. S.	222 Scott St., Tucson, Ariz.
YEANDLE, WILLIAM H., JR.	Courtland Mine, Courtland, Cochise Co., Ariz.
YOSHIWARA, SHIGETAKE	No. 12, Shinryudo-cho, Azabu, Tokyo, Japan.

MEMBERS' ADDRESSES WANTED

Name.	Last address of Record from which Mail has been returned.
ABBOTT, JAMES W.	2253 Calumet St., Los Angeles, Cal.
BALFOUR, A. M.	P. O. Box 301, Republic, Wash.
DE FARIA, CICERO C.	Rua Conde do Bomfim, No. 46, Rio de Janeiro, Brazil.
LINHARDT, HARVEY W.	Columbus Iron and Steel Co., Columbus, Ohio.
MACAULAY, R. M.	Grande Allee St., Quebec City, Canada.
MACDONALD, AUGUSTUS	Box 207, R. F. D., San Gabriel, Cal.
RAIBER, NICOLAUS H.	127 Madison St., Wilkes-Barre, Pa.
READ, NORMAN HATFIELD	R. F. C., Care War Office, London, England.
RENO, CHARLES A.	Ripley, Nev.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Dec. 10, 1916, to Dec. 31, 1916.

Date of Election.	Name.	Date of Death.
1885.	*Simpson, C. D.	Dec. 19, 1916.
1902	*Hall, Newman G.	Nov. 8, 1916.
1887	*Dickerson, Winchester.	Dec. 18, 1916.

* Member.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

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Meets first Wednesday after first Tuesday of each month.

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A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer*.

JOHN V. N. DORR,

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

W. E. C. EUSTIS, *Chairman*. R. L. AGASSIZ, *Vice-Chairman*.
E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.

W. H. LINNEY, *Chairman*. OSCAR LACHMUND, *Vice-Chairman*.
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
STANLEY A. EASTON, S. SHEDD.

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Meets second Saturday of each month.

GLENVILLE A. COLLINS, *Chairman*. H. L. MANLEY, *Vice-Chairman*.
AMOS SLATER, *Secretary-Treasurer*, 1043 Henry Bldg., Seattle, Wash.
I. F. LAUCKS, JOHN N. POTT.

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WALTER E. GABY, *Secretary-Treasurer*, 319 North Jackson St., Butte, Mont.
W. T. BURNS, N. B. BRALY.

San Francisco

Meets second Tuesday of each month.

T. A. RICKARD, *Chairman*. W. H. SHOCKLEY, *Vice-Chairman*.
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
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*Pennsylvania Anthracite*R. V. NORRIS, *Chairman*.

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W. J. RICHARDS, *Vice-Chairman*. ARTHUR H. STORRS, *Vice-Chairman*.
PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

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F. W. DeWOLF, *Vice-Chairman*. M. M. VALERIUS, *Vice-Chairman*.
WALTER E. MCCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
C. R. FORBES, F. D. RASH.

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ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
E. R. ZALINSKI, WILLIAM WRAITH.

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NORMAN CARMICHAEL, 1st Vice-Chair. B. BRITTON GOTTSBERGER, 2nd Vice-Chair.
ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
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 FRANCIS A. THOMSON,
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¹ Until Feb., 1917.

² Until Feb., 1918.

³ Until Feb., 1919.

⁴ Until Feb., 1920.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Roll Scale as a Factor in the Bessemer Process

BY A. PATTON* AND F. N. SELLER,† PITTSBURGH, PA.

(New York Meeting, February, 1917)

Introduction

THE use of roll scale in the Bessemer process dates back, to the best of our knowledge, at least 20 years. It was first used by the Ohio Steel Co., Youngstown, Ohio (now the Ohio Works of the Carnegie Steel Co.) under the direction of Sam MacDonald, Superintendent of the Bessemer Department at these works. Two 10-ton vessels and one blowing engine capable of blowing one heat at a time were employed. The object of using the scale was to shorten the length of the blow, or in other words, to increase the production with the same equipment. Various means were tried out for introducing the roll scale into the bath of molten iron: It was shoveled into the vessel before the heat was charged, so that the metal would flow over the scale; it was shoveled into the bath after the vessel was turned up; it was dumped into the empty iron ladle by the wheelbarrow-load, and at times was dumped on top of the molten metal in the iron ladle. But the practice of introducing the scale into the iron ladle had to be abandoned on account of danger from explosions and of skulling the ladles. It was soon learned that the proper place to charge the scale was in the empty vessel, so that when the molten iron was poured into the vessel it flowed over the scale, causing a considerable reaction to take place before the heat was turned up. Eventually, cylindrical chutes similar to those now in use were installed. Into these chutes the scale is dumped and carried into the empty vessel. Before this convenient means of introducing the scale was adopted, the Ohio Works had satisfactorily demonstrated that roll scale would increase production, by blowing 107 heats in one 12-hr. turn (1,087 tons) with one blowing engine, blowing one heat at a time; whereas, prior to the use of scale, the best practice at these works was about 80 heats under the same conditions.

In using roll scale and other oxides of iron to facilitate and control the refining of pig iron in the converter, the Bessemer plant has borrowed one of the most active agencies used in the open-hearth process with, as would be expected, much the same results. The effect of such additions

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to the Bessemer charge may be discussed as to the influence of this practice on quality, production and cost.

Experience has shown that the judicious use of roll scale in the Bessemer operation will not only increase production and reduce cost, but that at the same time it will improve the general quality of the steel. Most of the data on which these conclusions are based were obtained at the Bessemer plant of The National Works, National Tube Co., McKeesport, Pa., with the assistance of George Hitchins, Superintendent of Steel Works, and others of this company.

Influence on Quality

Taking up these phases of the subject in order of importance and with special reference to the manufacture of soft, weldable steel for pipe, it can be assumed to start with that the most important element in this problem is uniformity of operation. A higher standard of uniformity, especially in those points which affect the welding quality of the steel, is demanded in the manufacture of steel skelp than for other products, and it is probably true that this is generally obtained under modern conditions. Prominent among the factors contributing to this success is the use of roll scale in the converter.

The influence of this practice on quality is primarily due to the wider latitude in silicon which is thereby given to the blast-furnace operations, resulting in higher average silicon and lower sulphur. To obtain uniformly low-sulphur metal from the blast furnace, the Bessemer department must be designed and operated so as to be able to use without complaint iron that will run from 1 to 3 per cent. in silicon. This gives the blast-furnace management a larger margin of safety, thus making unnecessary sudden changes to prevent making iron too hot or too cold for the Bessemer plant. At the steel plant of the National Works, the average sulphur in the steel for 1915 was 0.038 per cent. with 95 per cent. under 0.050 per cent., the blast-furnace iron averaged for that year 1.77 per cent. silicon.

The proper use of roll scale, pig and steel scrap enables the blower to turn down his heats nearer to the same point in carbon by giving a sharper contrast on the final changes in the flame. This makes the loss in manganese and the residual manganese in the steel more constant. It also enables all heats to be blown more nearly in the same time and to the same temperature, which, of course, is favorable to uniformity in heating and rolling and makes all dependent operations more systematic.

The introduction of roll scale provides oxide of iron which would otherwise have to be formed by burning about $1\frac{1}{4}$ per cent. of the metallic charge. Somewhat higher silicon must therefore be present when roll scale is used to make up the thermal deficiency.

Production and Cost

In the design of a Bessemer plant, about all the engineer can do with regard to production is to make the vessels sufficiently large to blow the size of heat desired, with a bottom having sufficient blast area to blow the heat in a given time. He would probably use in his calculations a pig iron with 1.00 to 1.25 per cent. silicon. If the blast area was figured sufficiently large to blow iron containing 2.00 to 2.50 per cent. silicon in the same length of time, the operator would be in trouble when required to blow iron carrying in the neighborhood of 1 per cent. silicon. The vessels would stop badly, increasing the loss, and the quality of the steel would deteriorate. A vessel properly designed, with a bottom having sufficient blast area, using 20 to 25 lb. per square inch (1.4 to 1.7 atmospheres) blast pressure and designed to blow a heat of 1 per cent. silicon iron with pig scrap in 10 min., will blow a heat of 1.25 per cent. silicon iron in the same time if steel scrap is used. When the silicon rises above 1.25 per cent., roll scale and pig scrap can be used, increasing the scale as the silicon rises and so continuing to blow heats in approximately 10 min. When the iron runs above 2.00 per cent. silicon, it may be advisable to use steel scrap and roll scale, although the proportions required cannot be laid down according to rule, but must be varied by the operator, depending on mill conditions.

By this procedure, heats can be blown in 10 min. which would otherwise require 18 min.; thus the full capacity of the plant can be maintained regardless of variations in the pig iron.

The Bessemer department of the National Works has three vessels 8 ft. (2.4 m.) in diameter of 9 tons capacity; the bottoms have 18 tuyères, each having seven holes $\frac{1}{2}$ in. (1.3 cm.) in diameter giving a blast area of 24.7 sq. in. (159 sq. cm.). Using 20 to 25 lb. (1.4 to 1.7 atmospheres) blast pressure, heats of metal with silicon in the charge ranging from 1.00 to 2.50 per cent. can be blown in 10 to 12 min. As most of these heats carry over 1.40 per cent. silicon, roll scale and pig metal are nearly always used for "scrapping," the roll scale occasionally running as high as 6 per cent.

Steel scrap is not used with roll scale as a rule unless it is desirable to blow the heats more quickly, and this usually becomes necessary only when the silicon in iron rises above 2.50 per cent.

The production of this plant has been increased at least 20 per cent. by the use of roll scale in this way. For example, the mill records for the month of March, 1916, show:

Ingots produced, 64,055 tons = 7,132 heats

Charge:

78.4 per cent. direct metal averaging 1.72 per cent. silicon

(only one cast was below 1.00 per cent. Si this month, the balance between 1.10 and 2.54 per cent.).

10.0 per cent. cupola metal 1.22 per cent. silicon.

11.0 per cent. scrap consisting of $\left\{ \begin{array}{l} \text{Steel 2 per cent.} \\ \text{Pig iron } 5\frac{1}{2} \text{ per cent.} \\ \text{Roll scale } 3\frac{1}{2} \text{ per cent.} \end{array} \right.$

Delays:

Hr.	Min.	
50	26	waiting for iron.
6	24	mechanical repairs.
0	8	electrical repairs.
7	50	converting mill delays.

Total 64 48

Actual operating time = 583 hr. 12 min., or $14\frac{3}{4}$ min. per heat per vessel is the average time required to charge, blow and pour out each heat.

The sulphur in the steel for the above month averaged 0.039 per cent. with 94 per cent. under 0.05 per cent. The time required to charge the heat, pour into ladle, dump slag, fix stopper, etc., is at least 4 min., which would make an average blowing time of less than 11 min. per heat. This month is fairly representative of current practice at this plant, as the average silicon in metal from the blast furnaces for 1915 and for the first 9 months of 1916 was 1.77 per cent. for each period. It is doubtful if this plant could make 80 per cent. of this amount of steel without the use of roll scale. The practice of using roll scale was started at this plant in December, 1906, prior to which the silicon in blast-furnace metal averaged 1.36 per cent. for 1905 and 1.37 per cent. for 1906. The period from June 15 to July 15, 1916, shows an average of 1.90 per cent. silicon in blast-furnace metal, the lowest cast being 1.03 per cent. and the highest, 2.98 per cent. In Table 3 under column heading "Time of blow" will be found some data obtained on heats blown at the same time and under the same conditions that illustrate the influence of roll scale on time of blow.

To analyze the comparative cost of the practice we have been discussing is naturally a very complicated matter. Some Bessemer superintendents claim that it takes 2 tons of roll scale to make a ton of steel. This estimate seems to us excessively high, but even if such were the case we have in this practice a direct method of producing steel far below the cost of any known process.

In order to determine the influence of roll scale by itself, we made several experiments in November, 1916, by running five heats with and without scale at the same time in this plant, all other conditions being maintained constant as nearly as possible. The results of these tests are given in Table 3. The difference in time of blow is the most striking feature. This is entirely accounted for in the shortening of the silicon blow. For example, we give the following records from heats which were carefully watched in this respect, from the last series in Table 3.

Without Roll Scale			With Roll Scale		
Heat No.	To First Carbon Flame, Minutes	Finish of Blow, Minutes	Heat No.	To First Carbon Flame, Minutes	Finish of Blow, Minutes
3	7	14½	3	2½	10
4	7¾	15	4	3¼	11
5	7¾	15	5	3	10½

The quicker removal of silicon is apparently due to the large excess of oxygen supplied by the roll scale, and to the oxide of iron which remains in the bath available for combination with the silica. Silica is thus removed from the bath as fast as it forms and we have a good slag from the time the vessel is turned up. The rapid reduction of the magnetic oxide is clearly indicated by the chilling effect on the heat, which is equal to about one and one-half times that produced by the same weight of pig iron.

To determine whether a larger percentage of iron was carried away in the slags under certain conditions, samples of mixer metal and slag were taken for analysis throughout the month of November, 1916, with results shown in Table 4. This indicates that these slags are fairly uniform. At least, the combined iron in the slags does not vary with the silicon in the charge, as would be expected if the variable amount of roll scale added had any material influence on the slag. As the converter slags were as a rule observed to have the same physical consistency as well as practically the same analyses under this practice, it may be assumed that the free iron or shot carried away in the converter slag is also proportional to the weight of slag produced, which varies with the silicon in the charge.

During the month of November, 1916, when the experiments referred to in Table 3 were made, we weighed all the converter slag that was shipped from the mill, and weighed or estimated the loss in the lining and bottoms of the converters. With these data and the analyses of the slags and the charge throughout the month, we have computed the loss in metal in the slag as follows:

Converter Slags

Weight of slag calculated from silica contents per charge of 22,400 lb.:

	SiO ₂ , Lb.
100 lb. from bottom × 80 per cent.	= 80.0
50 lb. from sides* × 97 per cent.	= 48.5

	128.5
1.74 per cent. Si in charge = $0.0174 \times 22,400 \times 9\frac{1}{2}\%$	= 835.2

Total SiO ₂ in charge	= 963.7

* Estimated by measuring vessel before and after 500 heats.

Slags carry 60 per cent. SiO_2 (averaged from analyses of samples after removing shot).

$$\text{Silicates in slag} = \frac{963.7}{0.60} = 1,606 \text{ lb.}$$

Slag per heat, as weighed for month = 2,016 lb. (= 9 per cent. of charge).

$$\begin{aligned} \text{Theoretical weight of slag from } \text{SiO}_2 \\ \text{in silicates} &= 1,606 \text{ lb.} \end{aligned}$$

$$\text{Shot and scrap} = 410 \text{ lb. or } 20.3 \text{ per cent. of slag.}$$

$$\begin{aligned} \text{Combined iron in silicates} &= 16 \text{ per cent.} \times 1,606 = \\ 256 \text{ lb.} &= 12.7 \text{ per cent. of slag.} \end{aligned}$$

$$\text{Total iron in slag} = 33.0 \text{ per cent.}$$

The silicon in the charge for the group of heats included in the test heats referred to in Table 3 was practically the same as the average for the entire month. The actual converter and cupola loss for this month amounts to 9.05 per cent., which is somewhat above the average, due to an unusually high percentage of shot in the slag and to the fact that the average silicon for this month was comparatively high. The converter and cupola losses for this month have been analyzed as shown in Table 1.

TABLE 1.—*Converter and Cupola Losses—National Works, Month of November, 1916*

Loss in Direct Iron		Loss in Cupola Iron	
	Per Cent.		Per Cent.
Carbon.....	4.11	Carbon.....	3.90
Silicon.....	1.89	Silicon.....	1.66
Manganese.....	0.70	Manganese.....	0.60
	6.70	Cupola loss.....	1.24
			7.40

Known Losses of Other Materials

	Per Cent.
Steel scrap.....	0.35
Ferromanganese.....	30.00
Roll scale.....	(?)
Per cent. vessel and ladle slag.....	9.00
Total iron in slag.....	33.00*

*This is about 10 per cent. higher than the yearly average for 1915 due to stiffer vessel slag and greater amount of shot included.

Material Charged	Per Cent. of Charge	Per Cent. Loss	Per Cent. of Total Charge Loss
Direct metal.....	76.58	6.70	5.13
Cupola metal.....	10.76	7.40	0.80
Steel scrap in vessels.....	4.27	0.35	0.01
Ferromanganese and ferrosilicon.....	0.57	30.00	0.17
Roll scale.....	3.93		(?)
Pig iron (for scrapping in vessel).....	3.89	6.70	0.26
Iron in slag (9.0 per cent. $\times$ 33.00).....			2.97
			9.34
Less credit received for iron in slag returned to blast furnaces for month...			0.15
Theoretical loss.....			9.09
Actual loss.....			9.05

For comparison we give, in Table 2, the converter and cupola losses at this plant for 1915, which are more representative of the average practice.

TABLE 2.—*Theoretical and Actual Losses for 1915—National Works*

Loss in Direct Iron		Loss in Cupola Metal	
	Per Cent.		Per Cent.
Carbon.....	4.25	Carbon.....	4.00
Silicon.....	1.76	Silicon.....	1.56
Manganese.....	0.70	Manganese.....	0.60
	—	Cupola loss.....	1.26
Total.....	6.71		7.42
Loss of Other Materials Added			
	Per Cent.		Per Cent.
Loss in steel scrap (Mn).....	0.35	Per cent. vessel and ladle slag...	8.20*
Loss in ferromanganese.....	30.00	Iron in slag.....	23.40
Loss in roll scale.....			

Material Charged	Per Cent. of Charge	Per Cent. Loss	Per Cent. of Total Loss
Direct metal.....	77.15	6.71	5.17
Cupola metal.....	11.49	7.42	0.85
Steel scrap used in vessels.....	3.57	0.35	0.01
Ferromanganese and ferrosilicon.....	0.59	30.00	0.18
Roll scale (metallic).....	2.80		(?)
Pig iron used for scrapping vessel.....	4.40	6.71	0.30
Pig iron in slag (8.2 per cent. $\times$ 23.40 per cent.).....			1.92
			8.43
Less credit received for iron in slag.....			0.09
Theoretical loss.....			8.34
Actual loss.....			8.45

* Estimated.

TABLE 3

Expt. Run No.	Analysis of Mixer Iron			Weight of Charge in Pounds					Per Cent. Si in Charge	Blast Press. Pounds per Sq. In.	Time of Blow, Min.	Analyses of Slags						
	Total C	Si	Mn	Mixer Iron	Pig Iron	Steel	Roll Scale	Total Weight				Converter Slag			Ladle Slag			
												SiO ₂	Iron	Mn	SiO ₂	Iron	Mn	
No. 1.	Without Scale ...	4.34	1.96	0.84	18,640		3,160		21,800	1.80	23	19.0	65.27	15.30	8.11	53.12	15.70	14.00
	With Scale	4.21	2.20	0.81	19,180		2,130	1,210	22,620	1.90	23	12.5	61.80	17.50	7.84	50.10	16.20	13.44
No. 2.	Without Scale ...	3.98	1.60	0.76	19,000	3,000			22,000	1.63	25	18.6	63.60	16.20	8.11	48.06	16.60	12.84
	With Scale	4.06	1.63	0.77	19,000	2,000		1,000	22,000	1.56	25	10.0	62.26	17.70	8.64	48.64	16.50	12.97
No. 3.	Without Scale ...	3.92	1.62	0.70	19,100	3,500			22,600	1.62	25	18.9	65.34	16.50	8.24	50.48	13.00	11.59
	With Scale	4.00	1.96	0.75	19,200	2,500		1,400	23,100	1.87	25	11.3	63.40	16.50	8.15	52.04	15.90	12.94
No. 4.	Without Scale ...	4.24	1.79	0.64	19,020	2,540			21,560	1.80	25	15.0	63.50	14.50	6.82	47.40	18.20	18.21
	With Scale	4.00	1.77	0.68	19,080	1,550		1,000	21,630	1.69	25	10.2	60.50	18.70	8.01	47.00	16.10	16.95
Averages		4.08	1.82	0.74						1.73								

NOTE.—The average weight of all converter slags produced for this month was 2,010 lb. per heat. The ladle slag averaged 417 lb. per heat and was quite constant.

TABLE 4.—*Analysis of Converter Slag and Mixer Metal for Same Heat or Group of Heats—National Works, November, 1916*

Date	Analysis of Mixer Iron			Analysis of Vessel Slag*		
	Si	C (Total)	Mn	SiO ₂	Fe (Combined)	Mn
Nov. 1-2-3.....	1.90		0.75	60.70	16.40	8.21
Nov. 4-6-7	2.00		0.76	61.50	18.10	8.40
Nov. 8-9.....	1.90		0.77	62.70	16.40	7.75
Nov. 10.....	1.63	4.06	0.77	62.26	17.70	8.64
Nov. 10.....	1.60	3.98	0.76	63.60	16.20	8.11
Nov. 11.....	1.74	4.02	0.86		16.70	
Nov. 13.....	1.78	4.12	0.83		16.70	
Nov. 14.....	1.80	4.58	0.74		20.20	
Nov. 15.....	1.93	3.90	0.72		16.50	
Nov. 16.....	1.68	4.16	0.82		17.10	
Nov. 17.....	2.40	4.28	0.85		14.80	
Nov. 18.....	2.00	4.04	0.74		17.30	
Nov. 20.....	1.96	4.00	0.75	63.40	16.50	8.15
Nov. 20.....	1.62	3.92	0.70	65.34	16.50	8.24
Nov. 21.....	1.46				16.70	
Nov. 22.....	1.92				15.40	
Nov. 23.....	1.96				16.30	
Nov. 24.....	1.42				18.00	
Average.....	1.82	4.10	0.77	62.78	16.85	8.21

The use of roll scale was started at the National Works in December, 1906. For that year the silicon in the charge averaged 1.29 per cent. Assuming all other conditions to be unchanged, the increase in converter loss due to raising the silicon in the charge to 1.61 per cent., as was the case in 1915, would be about 0.3 per cent., based on the slag carrying 24 per cent. iron in each case.

There is probably a little more loss due to the greater activity of the reactions in the converter when using dirty or moist roll scale, but this is recovered for the most part and is credited to the steel works. The free iron now lost in the converter slag might also be recovered to advantage so that the necessary excess loss by this practice would consist of only a little more than the combined iron in the larger volume of slag due to a higher content of silicon. As the per cent. of combined iron in the slag has been found to be practically constant, this would be lost in any case in proportion as the silicon in the charge is raised, which we maintain is warranted to a certain extent on the basis of better quality alone.

Against this loss, most of which is in consequence of the larger volume

* After removing metallic shot. The combined iron in average monthly samples of Bessemer slags for 1916 varies from 13.69 to 17.52 per cent., with an average of 15.92 per cent. for the year.

of slag produced, we must credit a 20 per cent. reduction on steam cost and a similar reduction in most of the other items of cost, together with a 10 per cent. increase in scrap-melting capacity due to the use of higher-silicon iron. It is hardly necessary to produce further figures, even if we had them, to indicate that there is a substantial saving in cost of production by this practice. It may be said that it costs a little more to make the higher-silicon iron, but this is obviously more than offset by the advantages obtained, some of which, such as increased yield of finished product, are difficult to compute.

We have attempted to give briefly the results of 10 years' experience in the use of roll scale under the conditions that prevail at this particular plant. However, in summing up the benefits derived, as "better steel," "increased production" and "lower cost," we would not have the reader conclude that this is a "cure-all," as the practical Bessemer operator will readily appreciate that there are many other details such as temperature, ladle reactions, etc., which will require just as much attention as ever. Aside from all other factors, however, the judicious use of roll scale is a study within itself.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Magnetic Concentration of Low-grade Iron Ores

BY S. NORTON,* CARBONDALE, PA., AND S. LE FEVRE,† FOREST GLEN, N. Y.

(New York Meeting, February, 1917)

IN the West, capitalists have expended many millions of dollars developing the low-grade porphyry ores of copper. Half a dozen of these great enterprises have proved to be wonderful commercial successes. They have demanded improved crushing and concentrating machinery and consequently it has been developed. Many improved methods, cheap power, superior business organization, all these have contributed to this success, but the main feature is the handling of the material in enormous quantities, on a manufacturing scale. The mining chance of "striking it rich" has been eliminated by the manufacturing certainty of handling large quantities of material of known value, which while of relatively low grade, is available in large tonnages, assuring a supply for many years' run of the mill. Then the returns on the money invested are sure.

The concentration of low-grade magnetic iron ores, separating the magnetite crystals from the gangue by the use of magnets, is a field of work in which the lessons taught by the development of the porphyry coppers can be studied to advantage. Large-scale operations, and the liberal expenditure of enough money at the start to insure the most economical operations, are the means of securing the desired results.

The problem is to utilize millions of tons, and we may safely say billions of tons, of now worthless iron-bearing rock and to produce from it 10,000,000 to 20,000,000 tons per year of high-grade ore carrying 60 per cent. iron or higher; to take the lean material as found in nature, varying widely in iron content, and bring it up to a uniform standard of shipping ore. At present these ores are mined carrying from 25 to 50 per cent. iron, and the shipping product is brought up to 60 or 65 per cent. Fe. If future economies of operation make it possible to extend this process so that 15 per cent. iron in the crude ore can be treated as a commercial success, the additional tonnage available will be enormous. A 15 per cent. Fe crude ore raised to 60 per cent. Fe concentrate with 5 per cent.

* Consulting Engineer.

† Consulting Mining Engineer.

Mr. Norton and Mr. Le Fevre were associated in the development of the plants at Mineville, N. Y.

Fe loss in tailings would require 5.5 tons of crude for 1 ton of concentrates. The cost of crushing and concentration can be brought down to 12 c. per ton crude or possibly to 10 c., and the cost of quarrying on a large scale, probably 40 c., would be low enough to leave a profit even now. There are mountains of gabbro rock in the Adirondacks that will average 15 per cent. iron in the form of magnetite crystals of good size, say $\frac{1}{8}$ to $\frac{1}{16}$ in., but the concentrate would also carry some titanium.

MAGNETIC IRON-ORE RESOURCES

A thorough examination of some of the iron-ore properties and the knowledge acquired by development of extensive underground workings makes it possible to make quite definite estimates of tonnage available in certain areas, which show very large reserves.

F. S. Witherbee in his paper read before the American Iron and Steel Institute last October gave an estimate of 1,100,000,000 tons of crude



FIG. 1.—OUTSIDE OF MILL NO. 3.

magnetic ore above 30 per cent. Fe available for concentration in the Adirondack region alone, not including any titaniferous ores except the one deposit at Lake Sanford. He practically confined his estimate to the area of the iron-bearing gneisses which surround the central core of later eruptives, the anorthosites and gabbros, in which the titaniferous ores are found.

There are also in New Jersey and southeastern New York large areas that give conclusive evidence of vast amounts of non-titaniferous magnetites. The map accompanying the report of the State Geologist of New Jersey, year 1910, shows the area of iron-bearing gneiss rocks running northeast and southwest across the State about 18 miles wide by 50 miles

long, from Phillipsburgh to Greenwood Lake. In this area are located by name 366 magnetite mines that have been worked more or less. There are also 24 limonite and 8 hematite mines. These lenses may easily be capable of producing an average of 1,000,000 tons each and there are probably double the number listed not opened up. Here we have 900 sq. miles of iron-bearing gneisses in New Jersey, or more than in the Adirondack region, with nearly as much more additional in southeastern New York, reaching from the New Jersey line across the Hudson at Fort Montgomery and extending to Brewsters.

Mr. Witherbee's method of computation estimated 20 ft. thickness of ore over 10 per cent. of the surface area. He afterward cut the estimate in half to be conservative, which was equivalent to 10 ft. thickness of ore on one-tenth of the surface. This would give 2,700,000 tons per square mile or on 900 sq. miles in New Jersey 2,300,000,000 tons, with a goodly area in New York to fall back on to make up deficiencies.

A recent examination of a small area of 4 sq. miles near Sterling and Tuxedo Lakes in New York near the New Jersey line, showed ore reserves estimated at 5,000,000 tons per square mile.

Magnetic ore is found quite widely distributed, in Canada, Minnesota, California, New Mexico, New York, New Jersey, Pennsylvania, North and South Carolina, Tennessee. A detailed study of these deposits might be an interesting subject for the Bureau of Mines to follow up.

HISTORY OF DEVELOPMENT OF MAGNETIC SEPARATOR

Some time in the year 1887 my¹ attention was called to the magnetic separation of ores. At that time Edison was experimenting with his deflecting magnet and the Wenstrom, a Swedish machine of the drum type, was in use. The Conkling machine, which was also on the market, was the forerunner of the modern belt machine, but the magnetic attraction came from a single magnetized plate.

My first experiment was with Port Henry old-bed ore, which I crushed to pass through $\frac{1}{8}$ in. mesh, and then ran through an old-fashioned fanning-mill, such as are used on farms. I had better results than those obtained by Mr. Edison with his deflecting magnet. I then made a trial of the Conkling idea but found that the magnetic plate picked up a large part of the gangue with the ore, so that the ore had to be sized and fed very slowly to get good results. The same trouble was experienced with the Wenstrom machine.

I then made a small machine, substituting common horseshoe magnets for the magnetic plate of the Conkling machine. Since the magnets were

¹ References are to Mr. Norton and his experiments leading to the invention of the Ball-Norton separator.

of north and south polarity the ore turned end for end in moving from one pole to the next—not only the loops of ore and gangue but each individual piece turning. In this way the gangue was allowed to drop out, the ore was held, passed on to the next magnet, and so finally cleaned of the non-magnetic rock.

This use of magnets of alternating polarity is the fundamental principle of the Ball-Norton separator and greatly increases the capacity and efficiency of the machine.

However, as I was not an electrical engineer, I went to a friend, Clinton M. Ball, explained the operation of the machine, and told him that if he would make electromagnets of sufficient size and power, of alternating poles, I thought they would be a great improvement over anything previously used. Mr. Ball made the magnets, a small machine was built (shown in Fig. 2), and taken to the Benson mines, where about

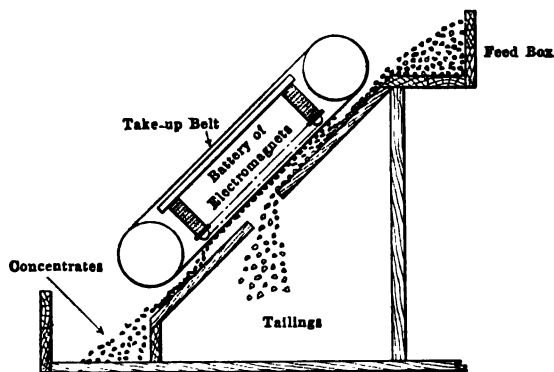


FIG. 2.—EXPERIMENTAL BALL-NORTON BELT SEPARATOR, 1888.

1,000 tons of 63 per cent. Fe ore was made, shipped to Braddock furnace and used there by James Gayley. This, I think, was the first use of magnetic iron-ore concentrates in a blast furnace.

The small machine was of the belt type. Mr. Ball soon after designed a drum-type machine, and later a double-drum machine in which a three-part separation was made. There are now magnetic machines of many types, but the majority use the alternating pole magnets.

Until 1908, I thought I was the first to use this form of magnet, but I then found that Z. Palmer had used the idea at Palmer Hill in 1854, so that it was not a new idea after all.

Mr. Palmer's machine is an interesting example of an early crude use of an important scientific principle. It was simple and primitive in the extreme, consisting primarily of a row of horseshoe magnets spiked around a log, like the spokes of a wheel. Finely crushed crude ore was allowed to slide through a wooden trough underneath the magnets, which were rotated by a crank attached to their supporting log. As the magnets

rotated, they dipped into the trough, the good ore became attached to them and was lifted up. It was then transferred to another trough, set above, by employing the simple device of a broom wielded by a husky Irishman.

The number of so-called magnetic separators for which patents have been taken out has been so large that it would be a waste of time even to try to enumerate them. Many of them were mere toys and a number were mechanical monstrosities. The belt and drum machines of the Ball and Norton patent have accounted for 90 per cent. of magnetic concentration by the dry process; while the wet magnetic process has been entirely monopolized in this country by the Gröndal-type machine. There are no patents today controlling magnetic separation, and there is no longer any chance for any new or startling discoveries in this line.

The first magnetic separator that I constructed was of the belt type. It was operated with a feed belt running 125 ft. per minute, while the take-off belt ran 250 ft. per minute. I wished to make a careful test of the capabilities of the machine when working on an ideal material, so I prepared a special mixture for the purpose. This consisted of crushed white marble, washed and sized between $\frac{1}{8}$ and $\frac{1}{20}$ -in. mesh, mixed with iron ore of the same size in a proportion of 2 parts marble to 1 part iron. It was evident that the particles of iron ore and marble would not be attached to each other, since the mixture was purely artificial. This mixture was then fed to the machine in a stream $\frac{1}{4}$ in. deep. The separation was almost perfect, giving an iron product over 99 per cent. pure. In this way, the possibility of a complete separation was conclusively demonstrated. In actual practice, however, such thorough preparation of material is impossible, and, owing to the difficulty of properly preparing the ore, there are some cases where separation cannot be made a commercial success.

DETERMINATION OF METHODS OF TREATMENT

The magnetic iron ores found in different localities vary widely, not only in their iron content, but also in their physical structure. The ores from the various districts require, consequently, radically different treatment.

In the first place, bodies of ore differ widely in crystallography. For example, the ores of the Champlain Valley are more coarsely crystalline than the ores of New Jersey, the Benson mine, or the Cornwall ore bed. Obviously the mill treatment of these ores cannot be the same. Among other things, ore containing the coarser crystals would not require to be crushed to so fine a size as ore of the Cornwall type. It is very important to find the exact size at which any particular ore is most economically separated, and this size can easily be determined by experimental

tests in a suitable laboratory. Moreover, the degree of fineness to which the ore must be crushed determines the process of separation to be employed. An ore which must be crushed to $\frac{1}{8}$ in., $\frac{1}{16}$ in. or lower will require the wet method of separation, while for larger sizes the dry method can be most profitably employed. The exact size that determines the method to be used is also somewhat dependent on the amount of moisture contained. Quite fine sizes can be separated if perfectly dry and fed in a thin film, but the dust problem is then somewhat difficult to deal with.

PRESENT PRACTICE, AND STATE OF DEVELOPMENT

The largest development in the iron-ore industry, using magnetic concentration, is at the plants of Witherbee, Sherman & Co. at Mineville, N. Y., where about 1,200,000 tons of crude ore were mined and separated in 1916. The dry process of separation is used. The Chateaugay Ore & Iron Co., at Lyon Mountain, N. Y., the Empire Steel &

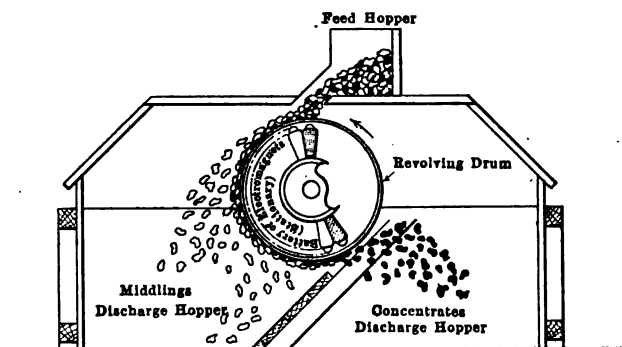


FIG. 3.—BALL-NORTON DRUM-TYPE MACHINE.

Iron Co. and the Ringwood Co. in New Jersey, also use the dry process successfully. The Gröndal wet separators have been recently installed at the Benson mines in New York. The largest development of the wet process in this country is on the Cornwall ore at Lebanon, Pa. This work is in charge of B. E. McKechnie, who is the highest authority on the wet process.

In the practical application of magnetic separation the most vital part is the preparation of the ore. It must be crushed so that the crystals of magnetite, or groups of crystals, are sufficiently freed from rock to bring the percentage of iron up to the standard set for shipping ore. On the other hand, it must not be crushed too fine, if it is possible to avoid it, otherwise the blast does not pass through readily in the furnace, or the ore blows over the top.

To maintain the best possible physical structure the separation must be made at each stage of the crushing.

If the material going to the separators is sized, the strength of the magnets can be adjusted to pick up the ore of more nearly uniform quality, but a separation can be made without very close sizing.

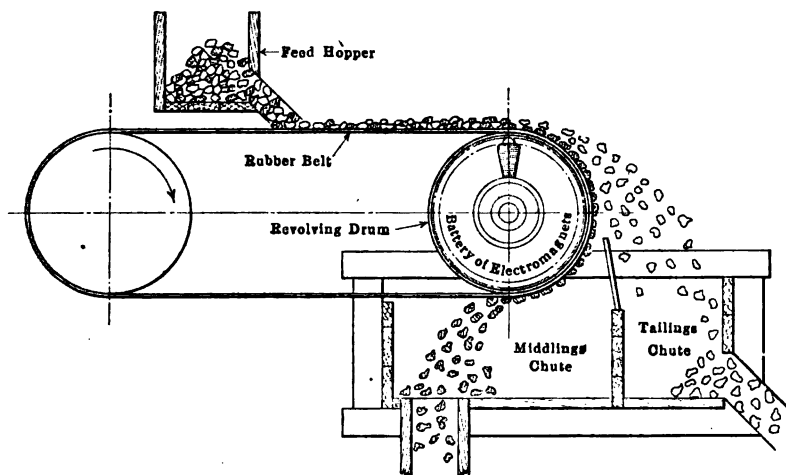


FIG. 4.—BALL-NORTON PULLEY MACHINE.

The drum type of machine (Fig. 3) has a section of fixed magnets inside of a revolving drum. It is used on the larger sizes where the richer

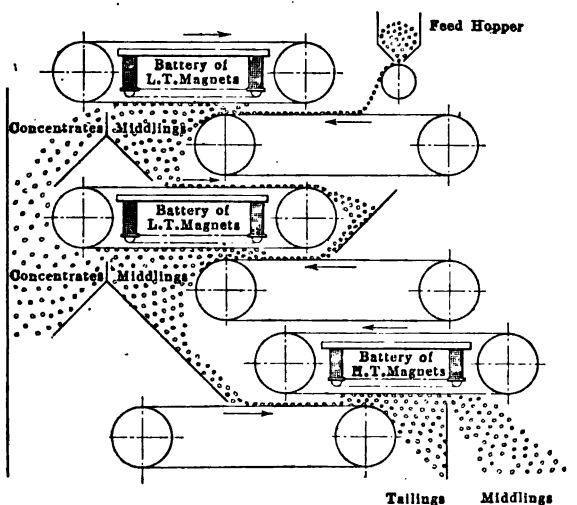


FIG. 5.—TRIPLE-DECK BALL-NORTON BELT MACHINE.

ores can be treated as coarse as $1\frac{1}{2}$ in. and a considerable percentage recovered. This size is as large as is desirable for rapid working in the furnace.

The pulley-type machine (Fig. 4) has a full circle of magnets which revolve with the drum. The magnets are wound to carry more current than the drum machine and will attract any lean ore, throwing off pure rock or tailings.

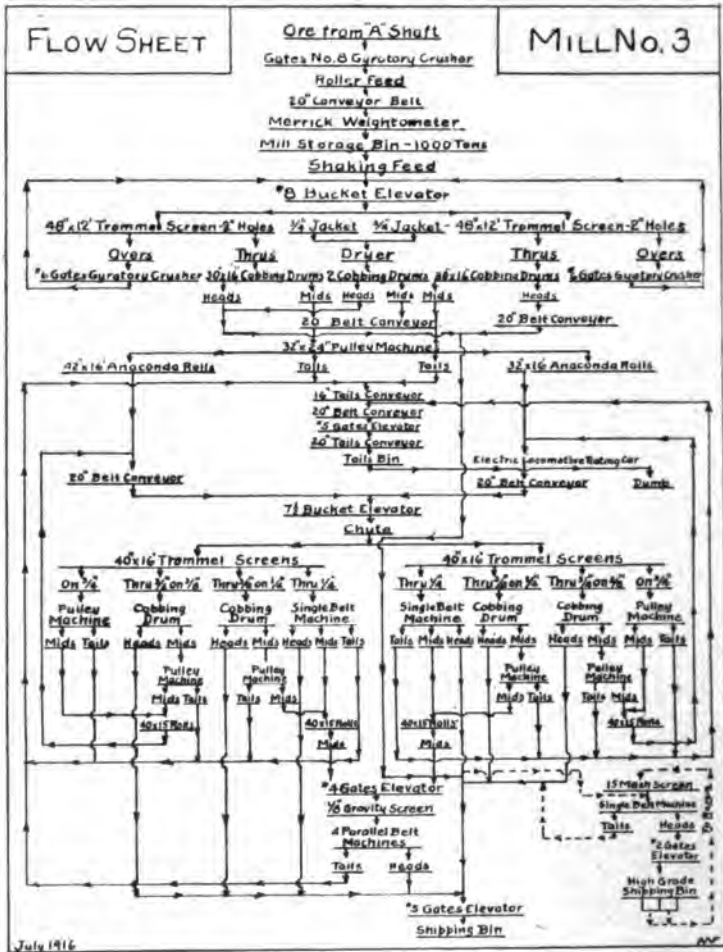


FIG. 6.

The drum and pulley machines will handle 30 to 50 tons per hour and are used together. The drum picks out any ore, as heads, rich enough for shipment. The pulley throws out rock lean enough to discard; what is left as middlings is crushed to about half its size and passed to machines treating finer sizes.

The belt-type machine (Fig. 5) is used when the ore is reduced to $\frac{1}{4}$ in. or below. The magnets are open to the air, so keep comparatively

cool and are easily inspected. Since the magnets of the belt machine lift the ore from the feed belt, the gangue is less likely to be held in suspension and a cleaner concentrate is insured. In the triple-deck machine shown in Fig. 5 the two top machines make heads and the bottom one makes tailings, and middlings to be reground.

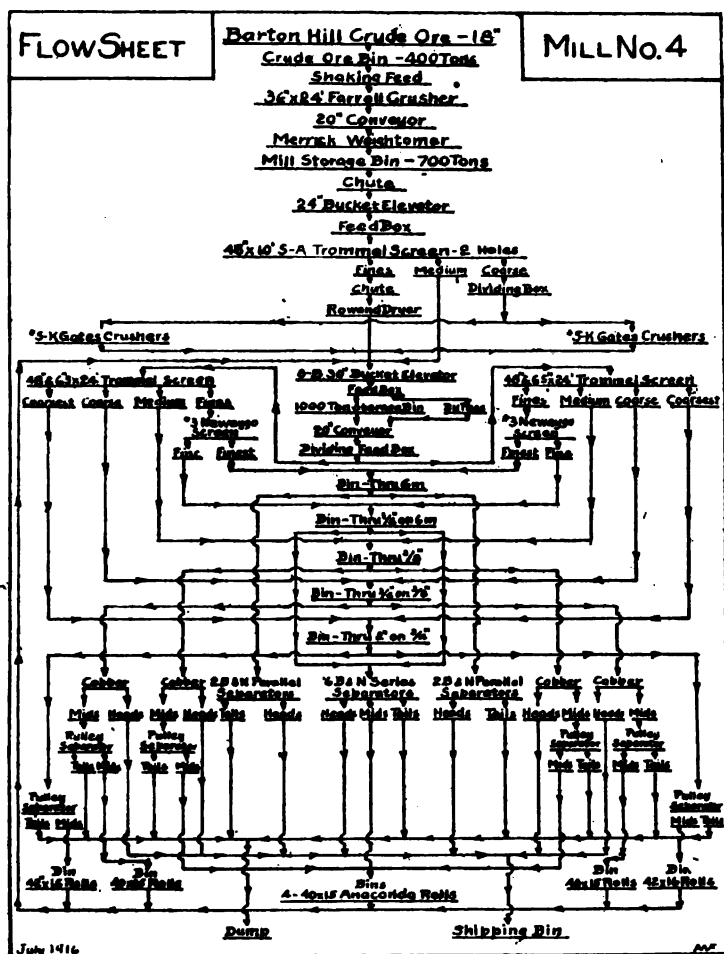


FIG. 7.

All these machines are used in the dry process.

If fine grinding is necessary to separate the crystals of magnetite from the gangue, wet separation is indicated. In this case treatment by sintering, or other processes, to agglomerate the ore is also required. The sintering process solves another difficulty by removing sulphur. Low iron and high sulphur content are handicaps which can now be both overcome by the combination of magnetic concentration and sintering.

The accompanying flow sheets of mill No. 3 (Fig. 6), mill No. 4 (Fig. 7), and mill No. 5 (Fig. 8), of Witherbee, Sherman & Co. at Mineville, N. Y., show arrangements for treating three different ores. The richness of the ore determines at what size the first separation can be made.

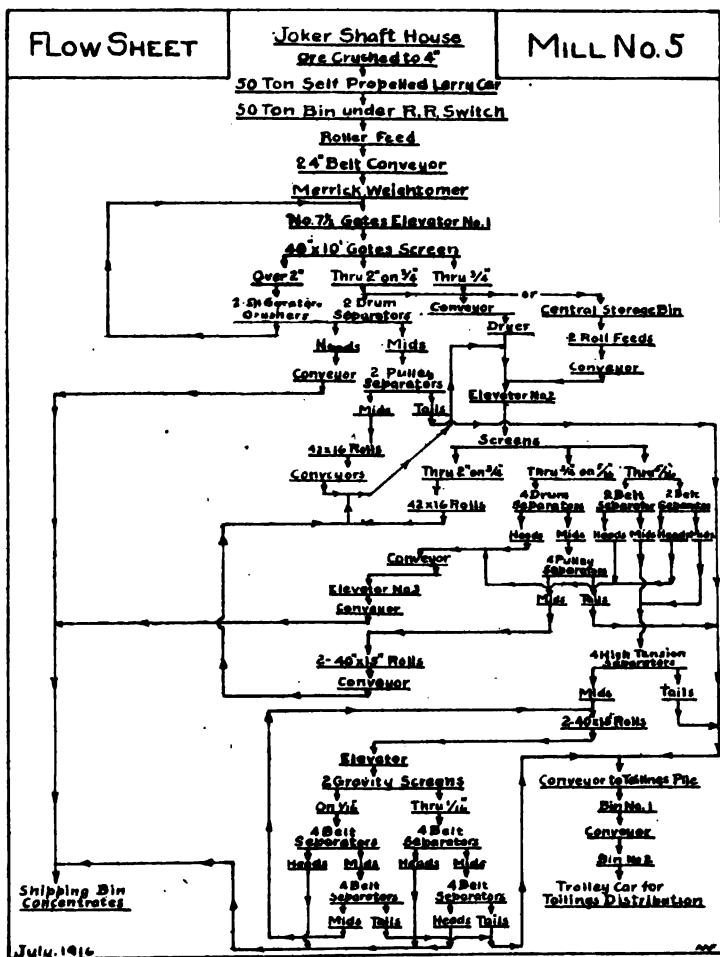


FIG. 8.

Mill No. 4 has eight sets of rolls instead of six in the other mills, because the ore is leaner and requires finer grinding.

WET MAGNETIC SEPARATION OF CORNWALL ORE²

During the dry magnetic separating tests on Cornwall ore, it became evident that this process of magnetic separation was not suitable for this ore, for the following reasons:

² Test conducted at Colebrook Furnaces, Lackawanna Iron and Steel Co., Lebanon, Pa. Described by B. E. McKechnie.

1. *Dust*.—The ore must be very dry in order to secure freedom of motion between the particles, or poor separation will result. This condition allows the very fine particles to escape as dust. No system of fans or other arrangements for eliminating or controlling this dust has been developed which can be successfully operated at a cost not prohibitive on this ore.

2. *Low Grade of Concentrates*.—Owing to the tendency of the fine particles of talcy gangue to cling to the magnetic pieces, it was found impossible to raise the iron constant above 52 per cent. when separating the average grade of Cornwall ore. This fact is demonstrated by washing concentrates from the dry magnetic separation, when the iron content was easily raised from 52 to 58 per cent. This suggested using a combined process of dry magnetic separation and of washing the magnetic product in some such apparatus as the Dorr classifier.

The objections to this arrangement are:

1. Dust problem is not solved.
2. The combination of a dry and wet process has the disadvantage of both cost of drying and the cost of pumping water supply and handling of wet products.

The same or better results could probably be obtained by a wet magnetic separation. This process would eliminate the cost of drying, the dust problem and should give a higher recovery of iron, due to the fact that a certain amount of iron would be lost in the slime from washing of dry concentrates. In the wet magnetic separation this washing is carried out in a strong magnetic field, which greatly reduces the loss from this cause.

In connection with the results obtained from the experimental wet magnetic separator constructed for investigating the wet process of magnetic separation of Cornwall ore, attention is called to the following points:

It is evident that in the separation of any ore by magnetic or other forces, the ore must be crushed sufficiently fine to free the valuable minerals from the gangue, and also that the degree of fineness required in the crushing depends upon the physical characteristics of the ore. As it is impractical to carry the crushing far enough to free all the mineral from the gangue, there will be a certain percentage of attached particles or "middlings" consisting of both mineral and gangue.

In the case of magnetic separation, these attached particles may go either as concentrates or tailings, depending on the strength of the magnetic field and the ratio by weight of magnetic to non-magnetic material in each. From this it follows that the stronger the magnetic field, the lower in iron will be both the concentrates and tailings product, due to a larger quantity of attached particles being attracted to the magnets. The reverse also holds true, that the lower the current, the higher in

iron will be both the concentrates and tailings as fewer attached particles will go to the concentrates and more to the tailings.

The richer the crude ore, the higher will be the grade of concentrates and the higher will be the iron content in the tailings. This is due to the fact that the rich ore carries a greater proportion of rich particles and a smaller proportion of rock. The grade of concentrates is raised, due to the smaller percentage of attached particles, while the percentage of iron in the tailings is greater, because of the smaller amount of clean rock present to balance the small quantity of magnetic material entering the tailings.

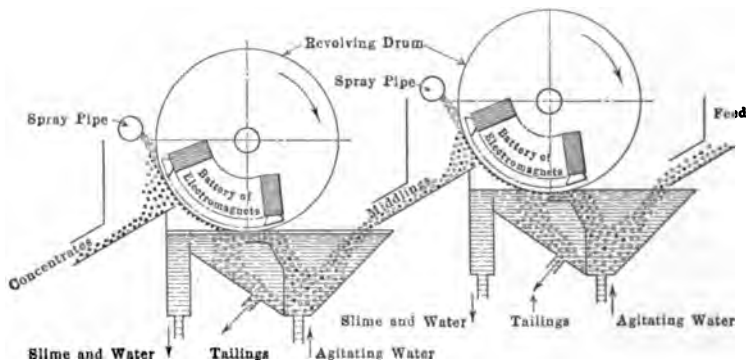


FIG. 9.—GRÖNDAL WET SEPARATOR.

Assuming that the amount of magnetic particles dropped by the separator is a nearly constant quantity, a higher percentage of recovery of iron is obtainable from a rich ore than from a leaner ore as the percentage of iron lost is evidently less.

The wet magnetic separator constructed for these experiments is a drum-type machine, constructed on the Ball-Norton principle. It consists of a number of stationary electromagnets, of alternate positive and negative polarity attached radially to a central shaft. About these magnets revolves a non-magnetic, water-tight drum, which carries a thin rubber belt.

In practice the magnets do not extend the entire circumference of the machine, but a gap is left between the points of feeding and delivery of concentrates. In this machine which was built for experimental work, any desired number of magnets could be cut out by short-circuiting the current around them.

Six arrangements of the drum were experimented with. Of these only Arrangement 1 was successful.

Arrangement 1.—The revolving drum drives the thin rubber belt which covers the face of the drum and passes over pulleys. Ore and water, or “pulp,” are fed by a launder or feed sole in such a manner that the feed is thrown against the moving belt. The magnetic particles are held to the drum, while the non-magnetic material falls into the tank and is drawn off. As the magnetic material held against the belt passes through the water, the influence of the alternating polarity of the magnets is to cause the magnetic particles to take a rolling action, which allows any entrapped gangue to fall out. As the drum further revolves, the magnetic concentrates are lifted out of the water and carried up the belt and around the pulley, where they are washed off by a spray of water.

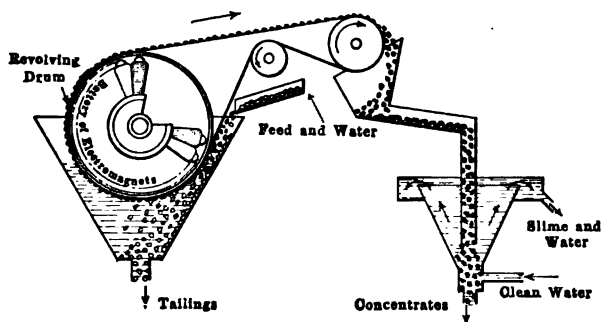


FIG. 10.—BALL-NORTON TYPE WET SEPARATOR.

In practice on Cornwall ore, it was found that a certain amount of very fine gangue was carried by the water into concentrates. They were, therefore, led to a classifier consisting of an inverted pyramid or tank, the bottom of which was fitted with a small hole and a connection above this hole for supplying clean water under slightly greater head than the depth of water in the tank. This water supply was regulated to furnish all the water required to supply the hole or the “spigot” and to furnish a slight raising current against which the heavy magnetic particles would fall but the very fine gangue could not, but would escape over the edge with surplus water.

Arrangement 2 was similar to 1 except that the water level in the tank was lowered until it was below the drum. This was done in an effort to reduce the amount of dirty water carried over the concentrates. The separator failed to make a separation operated in this manner, due to the fact that the surface tension of the water on the drum caused this water to act as a blanket, which did not allow the non-magnetic material to fall out.

In arrangements 3 and 4, the motion of the drum was reversed and the idler pulley removed. The feed sole was placed above to feed the pulp in the direction of travel of the belt. The tailings were to be removed at

the tank and the concentrates, carried past the division board placed under the last magnet were to be removed by the spray of water. Due to the surface tension of the water, no separation took place above the water level. The separation accomplished beyond this point was destroyed by currents set up in the water by the rotation of the drum.

Arrangements 5 and 6 were similar to 3 and 4 except that the belt was removed and the material fed directly on the drum. The action was the same as in 3 and 4.

Following are results obtained from Arrangement 1.

Test No. 1.—This was a preliminary run to test arrangement of separator. The crude ore was a mixture of concentrates and tailings from a dry separation.

14-mesh	10 Amp.
Crude.....	No analyses.
Concentrates.....	54.80 per cent. total Fe.
Tails.....	13.70 per cent. total Fe.

Test No. 2.

20-mesh	12½ Amp.
Crude.....	44.10 per cent. total Fe.
Concentrates.....	59.00 per cent. total Fe.
Tails.....	11.90 per cent. total Fe.
4.88 per cent. S, 1.14 per cent. Cu.	

A series of determinations showed the iron in these tailings distributed approximately as follows:

4.24 per cent. combined with sulphur.
4.76 per cent. combined as silicates.
2.90 per cent. combined as magnetite.

It should be noted that 9 per cent. represents non-magnetic iron, or that about 75 per cent. of the iron occurring in this tailings sample is non-magnetic and cannot be charged to the inefficiency of the separation.

Test No. 3.

14-mesh	11 Amp.
Crude.....	41.10 per cent. total Fe.
Concentrates.....	51.80 per cent. total Fe.
Tails.....	12.50 per cent. total Fe, 2.61 per cent. as magnetite.

Crude should be crushed finer.

Test No. 4.

20-mesh	12.5 Amp.
Crude.....	41.10 per cent. total Fe.
Concentrates	52.50 per cent. total Fe.
Tails.....	10.10 per cent. total Fe, 1.74 per cent. Fe as magnetite.

Concentrates were again separated with a lower current of 10 amp.

Concentrates.....	56.60 per cent. total Fe, 54.23 per cent. Fe as magnetite.
Tails.....	28.95 per cent. total Fe, 20.49 per cent. Fe as magnetite.

The tailings from this separation are really a middling product consisting of attached particles.

Test No. 5.—28-mesh, crude-ore sample divided into two parts, with 13.5 amp.

Crude.....35.20 per cent. total Fe, 30.30 per cent. Fe as magnetite.
 Concentrates.....57.20 per cent. total Fe, 54.95 per cent. Fe as magnetite.
 Tails.....12.50 per cent. total Fe, 2.18 per cent. Fe as magnetite.
 4.28 per cent. sulphur.

With 10 Amp.

Crude.....35.20 per cent. total Fe, 30.30 per cent. Fe as magnetite.
 Concentrates.....58.40 per cent. total Fe, 55.97 per cent. Fe as magnetite.
 Tails.....13.30 per cent. total Fe, 1.02 per cent. Fe as magnetite,
 5.70 per cent. S.

Test No. 6.

28-mesh

11 Amp.

Crude.....42.30 per cent. total Fe.
 Concentrates.....58.90 per cent. total Fe, 57.28 per cent. Fe as magnetite.
 Tails.....15.60 per cent. total Fe, 0.70 per cent. Fe as magnetite,
 10.20 per cent. S.

Note high tails due to large amount of iron combined with sulphur.

Test No. 7.—20-mesh crude divided into three parts, with 12.75 amp.

Crude40.20 per cent. total Fe, 34.60 per cent. Fe as magnetite.
 Concentrates.....56.70 per cent. total Fe, 55.68 per cent. Fe as magnetite.
 Tails.....10.10 per cent. total Fe, 0.87 per cent. Fe as magnetite.

With 10.5 Amp.

Concentrates.....57.40 per cent. total Fe, 56.12 per cent. Fe as magnetite.
 Tails.....10.30 per cent. total Fe, 1.31 per cent. Fe as magnetite.

With 8 Amp.

Concentrates.....57.40 per cent. total Fe, 56.30 per cent. Fe as magnetite.
 Tails17.00 per cent. total Fe, 8.70 per cent. Fe as magnetite.

Note that in using lower currents the iron content of the tailings increases more rapidly than in the concentrates.

A screen test on the crude ore sample above and on the concentrates from the separation with 12.75 amp. gave the results shown in Table 1.

TABLE 1.—*Screen Test on Crude and Concentrates*

Size	Per Cent. by Weight	Per Cent. Total Fe	Per Cent. Fe as Magnetite
Crude:			
On 28	13.04	37.00	30.74
On 35	15.19	38.30	31.61
On 48	13.27	39.50	33.93
On 65	9.86	42.80	39.73
On 100	8.05	46.30	42.05
Through 100	40.59	40.10	35.96
Concentrates:			
On 28	14.22	48.40	44.37
On 35	15.31	49.59	45.39
On 48	13.71	52.90	49.88
On 65	9.28	57.50	54.81
On 100	12.01	60.20	58.73
Through 100	34.70	63.40	62.78

Note low grade of the coarse sizes of concentrates and that iron content does not reach 60 per cent. until 100-mesh size is reached. This shows the necessity for fine crushing.

Concentrates upon which the preceding screen tests were made were re-separated with 12 amp.

Concentrates.....57.90 per cent. total Fe, 56.55 per cent. magnetic Fe.
A screen test on this sample follows:

Screen	Per Cent. by Weight	Per Cent. Total Fe	Per Cent. Fe as Magnetite
On 28	14.77	48.30	44.50
On 35	16.32	52.00	48.72
On 48	14.01	55.60	52.93
On 65	12.50	59.70	58.87
On 100	11.59	62.90	61.77
Through 100	30.50	66.50	65.54

Note that the greatest gain in Fe content is in the fine sizes, due to washing out of fine gangue.

Test No. 9.

20-mesh.

11 Amp.

Crude ore was crushed in rolls four times to secure a large amount of fine material.

Screen	Screen Test on Crude	Per Cent., Weight
On 20		0.00
On 28		6.03
On 35		10.70
On 48		12.25
On 65		11.28
On 100		10.12
Through 100		49.61

Crude.....38.30 per cent. total Fe, 33.64 per cent. Fe as magnetite.

Concentrates.....58.40 per cent. total Fe, 55.97 per cent. Fe as magnetite.

Tails.....10.20 per cent. total Fe, 2.17 per cent. Fe as magnetite.

Test No. 10.—Crude ore crushed three times through rolls. Flakes screened out with 10-mesh sieve.

12.5 Amp.

Crude.....39.40 per cent. total Fe, 34.65 per cent. Fe as magnetite.

Concentrates.....56.60 per cent. total Fe, 54.09 per cent. Fe as magnetite.

Tails.....17.50 per cent. total Fe, 8.99 per cent. Fe as magnetite.

Speed of separator and feed were too great in this test.

20-mesh

11 Amp.

Crude.....37.50 per cent. total Fe, 31.90 per cent. Fe as magnetite.

Concentrates.....58.30 per cent. total Fe.

Tails.....13.70 per cent. total Fe, 5.51 per cent. Fe as magnetite.

Rate of feed 1,890 lb. per hour.

Test No. 11.

10-mesh

12 Amp.

Crude.....	37.50 per cent. total Fe, 31.90 per cent. Fe as magnetite.
Concentrates.....	57.80 per cent. total Fe, 55.24 per cent. Fe as magnetite.
Tails.....	9.60 per cent. total Fe, 1.30 per cent. Fe as magnetite.

Test No. 12.

12 Amp.

Crude.....	38.80 per cent. total Fe.
Concentrates.....	56.80 per cent. total Fe.
Tails.....	8.80 per cent. total Fe, 1.16 per cent. Fe as magnetite, 2.58 per cent. S.

Test No. 13.

Weight of feed 139 lb.....12.5 Amp.

Crude ore three times through rolls.

Width feed 10 in.

Crude.....	38.70 per cent. total Fe, 34.37 per cent. Fe as magnetite.
Concentrates.....	57.30 per cent. total Fe, 54.23 per cent. Fe as magnetite.
Tails.....	13.10 per cent. total Fe, 1.16 per cent. Fe as magnetite.

Sulphur in tailings..... 6.70 per cent.

Iron in tailings due to sulphur..... 6.99 per cent.

Rate of feed 1,471 lb. per hour.

Estimated rate of feed for 30-in. separator, 4,410 lb. per hour.

RESULTS OF DRY SEPARATION IN TESTING LABORATORY

The following reports show results of samples tested to determine treatment required and quality of concentrates that could be expected. These tests were run on a regular mill size separator and the results could be duplicated in actual practice. The separate determinations of iron as magnetite, and total iron, were made so that the difference between the two would show the amount of iron combined as silicates in hornblende and other gangue minerals.

No. 93, Sample 5, 150 ft. North of 4

	Crude 93 lb. Per Cent.	Heads 59 lb. Per Cent.	Tails 34 lb. Per Cent.
Fe as magnetite.....	51.000		6.140
Total Fe.....	52.980	65.490	
SiO ₂		4.160	
Al ₂ O ₃		4.290	
CaO.....		0.550	
MgO.....		1.030	
P.....	0.010	0.005	
S.....		Trace	
Mn.....		0.050	
Ti.....		0.390	

No. 94, Sample 6, 20 ft. East of 5

	Crude 99 lb. Per Cent.	Heads 46 lb. Per Cent.	Tails 53 lb. Per Cent.
Fe as magnetite.....	32.640		6.140
Total Fe.....	35.120	63.140	
SiO ₂		6.510	
Al ₂ O ₃		4.600	
CaO.....		0.520	
MgO.....		0.970	
P.....	0.021	0.010	
S.....		Trace	
Mn.....		0.040	
Ti.....		0.450	

No. 95, Sample 7, from South End

	Crude 113 lb. Per Cent.	Heads 70 lb. Per Cent.	Tails 43 lb. Per Cent.
Fe as magnetite.....	45.740		6.030
Total Fe.....	45.740	65.440	
SiO ₂		4.780	
Al ₂ O ₃		3.980	
CaO.....		0.230	
MgO.....		0.530	
P.....	0.011	0.009	
S.....		Trace	
Mn.....		0.040	
Ti.....		0.390	

Jan. 20, 1913.

H. D. GEHRET.

No. 234, Separation Test on Jackson Hill Ore, Arnold, N. Y.

307 lb. crude ore was crushed to pass $\frac{1}{8}$ -in. screen; separated, by screening, into two sizes, on 16 and through 16-mesh.

Through 8 on 16-mesh 132 lb., through 16, 175 lb.

8-16 size, treated on belt machine using $3\frac{1}{2}$, 4, 5 amp. and finally with 4 amp. for heads. Then 12 amp. for midds and tails.

Crude 132 lb.....	
Heads $20\frac{1}{2}$	Fe 63.35, P 0.006.
Midds $42\frac{1}{2}$	Fe 25.10.
Tails 62.....	Fe 3.80

The midds were rolled through 16-mesh and added to crude through 16; the whole passed over belt machine using 2 amp.

Crude $224\frac{1}{2}$ lb.....	Fe
Heads 115.....	Fe 66.15, P 0.005
Tails $108\frac{1}{2}$	Fe 3.00
General crude 307 lb.....	Fe 30.85, P 0.008.
General conc. $135\frac{1}{2}$	Fe 65.60, P 0.005.
General tails $171\frac{1}{2}$	Fe 3.27.

Ratio, 2.265 tons crude to 1 ton concentrates.

Screen Test of Concentrates			
Through	8 on	10-mesh.....	2.56
Through	10 on	20-mesh.....	26.75
Through	20 on	40-mesh.....	33.70
Through	40 on	60-mesh.....	20.50
Through	60 on	80-mesh.....	5.86
Through	80 on	100-mesh.....	2.93
Through	100		7.70
			100.00

NOTE.—Owing to the iron being present in very small crystals it is necessary to crush this ore to at least $\frac{1}{8}$ in. before separation, but since the ore is extremely brittle this is easily accomplished with little power.

The tailings consists almost wholly of red feldspar which could be utilized for potash extraction by the Cushman process which is now commercially successful.

Dec. 10, 1915.

H. D. GEHRET.

No. 235, *Separation Test on Palmer Hill Ore, Arnold, N. Y.*

325 lb. crude ore was crushed through 4-mesh and screened through sizes:

		Per Cent.
Through	4 on 8	102½ lb., 31.54
Through	8 on 16	78 24.00
Through	16	144½ 44.46
		325 lb.

The 4 on 8 size, passed over belt machine using 4¾ amp. for heads and 15 amp. for midds and tails.

Heads	6 lb.	Fe 54.80
Midds	85	
Tails	11½.....	Fe 7.15

The above heads being too low in iron, were added to midds and rolled through 8-mesh, fines screened out and added to crude.

8 on 16 size, passed over belt machine on 5 amp. for heads and 15 amp. for midds and tails.

Crude	125 lb.	
Heads	31	Fe 61.65, P 0.006
Midds	62	
Tails	32.....	Fe 6.35

The midds ground through 16-mesh, added to crude and passed over belt machine on 8 amp.

Crude	250½ lb.	
Heads	146½.....	Fe 65.30, P 0.006
Tails	104.....	Fe 8.65,

General crude	325 lb.	Fe 39.00, P 0.018
General conc.	177½.....	Fe 64.50, P 0.006
General tails	147½.....	Fe 8.05

Ratio, 1.83 tons crude per ton of concentrates.

Screen Test of Concentrates

	Per Cent.
Through 8 on 10-mesh.....	3.47
Through 10 on 20-mesh.....	26.60
Through 20 on 40-mesh.....	33.50
Through 40 on 60-mesh.....	17.92
Through 60 on 80-mesh.....	6.36
Through 80 on 100-mesh.....	2.90
Through 100.....	9.25
	100.00

NOTE.—In order to reduce the iron in the tailings finer grinding through 16-mesh will be necessary at the last stage making a three-part separation on the through 16 size and retreating resulting midds.

Dec. 13, 1915.

H. D. GEHRET.

No. 236, *Separation Test on Ore from Battie, Arnold, N.Y.*

290 lb. crude ore was crushed through 4-mesh, screened to size as follows:

	Per Cent.
Through 4 on 8-mesh 130 lb.	44.81
Through 8 on 16-mesh 72 lb.	24.95
Through 16 88 lb.	30.34
	290 lb. 100.00

4-8-mesh size, passed over belt machine using $4\frac{1}{2}$ amp. for heads, and 15 amp. for midds and tails.

Heads 44 lb.....	Fe 60.50, P 0.009
Midds 74	
Tails 12.....	Fe 8.65

The midds were rolled through 8-mesh, screened and added to crudes. 8 on 16, 48 lb.; through 16, 26 lb.

8-16 size, passed over belt machine on 4 amp. and 15 amp.

Crude 120 lb.

Heads 43.....	Fe 62.40, P 0.011
Midds 55	
Tails 22.....	Fe 8.20

The midds ground through 16-mesh and added to crude. Through 16-mesh, over belt on 7 amp.

Crude 169 lb.	
Heads 90.....	Fe 61.80, P 0.006
Tails 79.....	Fe 8.30

General crude 290 lb.....	Fe 41.00, P 0.017
General conc. 177.....	Fe 61.70, P 0.008
General tails 113.....	Fe 8.30
Ratio, 1.64.	

Screen Test of Concentrates

			Per Cent.
Through	4 on	8-mesh	23.43
Through	8 on	10-mesh	19.15
Through	10 on	20-mesh	27.45
Through	20 on	40-mesh	20.00
Through	40 on	60-mesh	8.57
Through	60 on	80-mesh	2.85
Through	80 on	100-mesh	2.28
Through	100		6.26
			100.00

Dec. 15, 1915.

H. D. GEHRET.

The demonstration of the dry process of magnetic separation is the result of 14 years' work at Mineville, N. Y. Witherbee, Sherman & Co. have now in operation three mills having a combined capacity of 6,000 tons per day of crude ore. The Empire Steel & Iron Co. and the Ringwood Co. have demonstrated what can be done with New Jersey ores. The Ringwood Co. has also worked out a dry process of jigging for their tailings to recover the martite, which is non-magnetic. Martite is a hematite in composition, but is very similar in appearance and crystallization to the magnetite. Some of the magnetic ores have varying amounts of martite mixed with the magnetite.

SUMMARY

The known and partially developed orebodies of New York and New Jersey could, if equipped with the best modern mining and milling machinery and using the best methods, produce at the present time 25,000 tons of 60 per cent. iron ore per day. This can be delivered for an average freight charge of \$0.75 per ton from mill to tidewater. The operating cost of production should reach the "dollar rock" ideal of the Lake Superior Copper region, and the cost of mining and milling 1 ton of crude ore should be about \$1 for underground mining when handled in large quantities.

The ratio of concentration would be 2 tons of crude per ton of concentrates for an average. There are reserves of magnetic ore sufficient to double the above production, and then last probably 100 years.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

An Investigation on Rock Crushing Made at McGill University *

BY JOHN W. BELL, † M. SC., MONTREAL, QUE.

(New York Meeting, February, 1917)

Aim of Rock-Crushing Experiments

THE aim of the laboratory experiments described in this paper was twofold:

1. To measure as accurately as possible the maximum amount of crushing that can be effected by 1 hp. acting for 24 hr., by the two proposed methods for measuring the amount of crushing, namely: (a) by Rittinger's method; (b) by Stadler's method, based on Kick's law.

2. To show, if possible, that by one of these methods, the amount of crushing produced by 1 hp. in 24 hr. is a constant amount throughout a wide range in the diameter of the piece crushed, and that by the other method, the amount of crushing is a variable quantity over the same range of diameter; or else show that neither of the proposed methods for measuring the amount of crushing indicates a fixed relation between power and crushing and consequently a new method for computing this amount will have to be found to permit the establishment of a law of crushing.

Experimental Methods

Testing machines, calorimetric methods and commercial types of rock-crushing machines were considered with reference to their value for measuring power used in crushing. The following electrically driven crushing machines, installed and available in the McGill Ore-dressing Laboratory and which had been used in previous rock-crushing tests, on the whole seemed especially suited to the investigation: (1) Comet "A" crusher; (2) 7 by 9-in. Dodge crusher; (3) 10 by 16-in. rolls. In later tests a 3½-ft. Huntington mill was used.

* Abstract of a paper prepared for publication in the *Transactions of the Canadian Mining Institute* giving an account of an experimental investigation of the theory of rock crushing made by John W. Bell, M. Sc., W. G. Mitchell, M. Sc., P. P. Baily, M. Sc., and W. E. Cockfield, M. Sc., in the Ore-dressing Laboratory of McGill University, Montreal, Canada. This paper is a continuation of the discussion of the relative value, for measuring the efficiency of crushing machines, of Kick's law (advocated by Stadler) and Rittinger's theory (advocated by Gates). The abstract is published in this *Bulletin* to afford opportunity for a discussion of this important topic at the New York meeting in February. In this brief abstract only the most important data and conclusions are given.

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Rock Used

In all the series of tests, except one, the rock crushed was a tinguaité—an intruded eruptive obtained from a local quarry. It is a hard and brittle rock, with a tendency to break rather more easily in one plane than in another at right angles to it. It, therefore, is inclined to break in slabs, although the fractured pieces are always irregular in shape.

1914 Crushing Tests

In 1914, three series of tests were made to measure the work done per "apparent effective horsepower in 24 hr." In the first series the laboratory Comet "A" crusher was used; in the second series, the Dodge crusher; and in the third series, the rolls.

In certain respects the results of these tests were unsatisfactory, but the data obtained indicated strikingly the probability of a fatal error in Stadler's method and a degree of reliability in Rittinger's. The figures showed an enormous variation in the amount of crushing done by a measured horsepower through a wide range in diameter when calculated by Stadler's method, based on Kick's law, and a remarkable constancy when calculated by the law proposed by Rittinger in 1867. The comparative figures in Table 1 justify fully the indicated, but not necessarily proved, outcome of the whole investigation.

TABLE 1.—*Summary of the Most Efficient of the 1914 Tests*

Test No.	Crusher	Diameter of Piece Crushed, Inches	Work Done per A.E. Hp.* in 24 Hr.	
			Measured in Stadler E. U.†	Measured in Rittinger S. U.‡
6	Gyratory	3.50	710	947
27	Dodge	1.20	520	1,030
34	Rolls	0.50	286	1,128
41	Rolls	0.29	138	1,000
47	Rolls	0.18	68	1,022
48	Rolls	0.11	89	1,187
49	Rolls	0.07	52	823

Total range.....50 diameters.

Range in roll tests.....7 diameters.

* Apparent effective horsepower. † E. U. = energy units. ‡ S. U. = surface units.

1914-1915 Rolls Tests

In the fall of 1914, the fourth series was commenced to investigate in more detail the effect of "tonnage" and "work done per ton" before adopting a set procedure for the final series with rolls.

The decrease in efficiency which results by increasing the feed tonnage, or by increasing the crushing work per ton, for sizes 0.46 in. in diameter and upward was clearly indicated by the results obtained.

The sudden increase in efficiency, when the feed diameter is reduced to 0.30 in., is explained by the marked change in the sound and vibrations emanating from the machine. When crushing 0.46-in. feed at the 30-ton rate, the vibrations in the laboratory concrete floor were perceptible 20 ft. distant from the rolls. When crushing 0.30-in. feed, the floor vibrations practically disappeared, and there was a marked diminution in the smashing shocks, resembling a series of explosions, which were so pronounced during crushing of the larger feeds. In the next and succeeding smaller sizes the characteristic sound effect may be described as being in the nature of a steady grinding noise, indicating that a more uniform pressure replaces the fluctuating and violent forces transmitted to the journals during the crushing of the larger pieces. That there should be an enormous decrease in machine efficiency under the stated circumstances, is in the writer's opinion opposed to reason and common sense, and yet that is the one and only conclusion that can be reached by acceptance of Stadler's theory based on Kick's law, as will be apparent by examination of the data in Table 2 and Fig. 1.

TABLE 2.—*Summary of Results of 1914-1915 Rolls Tests*

Test No.	Remarks	Diameter Feed, Inches	Tons Crushed in 24 Hr.	Work Done per Ton, Rittinger S. U.	Work Done per A. E. Hp.	
					Stadler E. U.	Rittinger S. U.
50		1.00	12.2	6.9	623	1,198
53		0.70	10.2	9.8	472	1,192
60		0.46	5.8	20.6	272	1,097
79-88	Average 9 tests*	0.30	28.7	21.3	190	1,002
89-93	Average 5 tests	0.19	31.3	30.9	150	1,115
94-97	Average 4 tests	0.12	33.8	22.6	109	1,054
98-101	Average 4 tests	0.08	34.5	31.8	82	1,028
102-105	Average 4 tests	0.05	31.6	21.5	77	1,137
106-109	Average 4 tests	0.03	29.8	28.0	77	1,250

Range.....33 diameters. Average R. S. U.....1,120.

* Test 86 excluded.

The Stadler energy unit varies depending not only on the diameter of the feed, but on the work done per ton as well. In tests 50-53 the "work done per ton" is less than in the remaining tests (see Table 2 and Fig. 1) and the Stadler efficiency increases correspondingly.

It follows that there are an infinite number of values of this apparently misnamed "energy unit of crushing," consequently it is not a unit indicat-

ing a fixed relation between power and crushing when the material crushed is rock, however applicable it may or may not be in estimating the power required to crush an ideal and more or less imaginary substance. The unavoidable conclusion to be drawn from the experiments is that the Stadler energy unit is of no value in determining the relative efficiencies of various types of rock-crushing machines. For a single machine, crushing feed of limited range in diameter with only a small variation in the

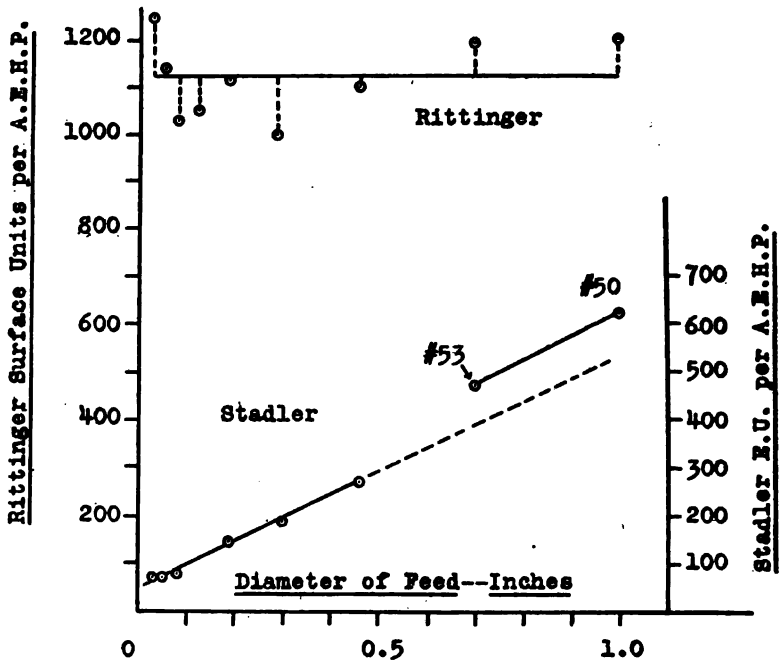


FIG. 1.

work done per ton, Stadler's theory, based as it is on one of the manifestations of crushing, shows the character of the changes in efficiency produced by changes in adjustment, but the Rittinger system, even if it fails to conform with a law, is preferable by reason of its greater sensitiveness to small variations in the amounts of crushing.

Power in Fine Crushing

The sudden increase in the work per apparent effective horsepower in the last four tests (see Table 2), indicated the importance of investigating the relation of power and the crushing of rock particles of very small diameter. Some preliminary work convinced the author that a *Huntington* mill was admirably suited for the investigation and this machine, with some modifications, was accordingly used.

TABLE 3.—*Summary of Average Results, Huntington Mill Tests*

Test No.	Remarks	Grade	Diameter Feed, inches	Work Done per A. E. Hp.	
				Stadler E. U.	Rittinger S. U.
114-115		+ 20	0.031	55.0	1,590
116-119		+ 30	0.020	47.5	1,860
120-122		+ 40	0.014	45.5	2,030
125-127		+ 60	0.009	34.2	1,970
128-129	Test 130 excluded...	+ 80	0.007	28.1	1,910
131-132		+100	0.005	20.2	1,800

Range of diameter tested.....6.

The results in Table 3 show the characteristic drop in Stadler units per apparent effective horsepower, although the range of diameters tested is not large. However, many more experiments would be required to determine the effect of alterations in tonnage, percentage of water in pulp, speed of machine, etc.

Test 116 was made to obtain an indication of the effect of a reduced tonnage. The number of Rittinger units appears to increase, but several check tests would be required to form a definite conclusion.

Considered as a whole, the results are in closer agreement with Rittinger's theory than could be expected, when two obstacles which conspire to defeat the purpose of the investigation are taken into account. One, is the difficulty of measuring the power used only in crushing. The second has to do with the measurement of the surface in the -200 grade. Nobody has yet made a reliable measurement of this, and it is probable that it is a variable factor. For the -200 grade, Stadler's ordinal number 28 was adopted, for although it was realized that it was not necessarily an accurate estimate of the average volume of the particles it was supposed to represent, it would at least serve to make the comparison as fair to one as to the other theory. The corresponding Rittinger number is 780 which was adopted. It is probable that the average factor is appreciably greater than 780.

If in going from coarse to fine crushing there is a considerable decrease in surface in the -200 grades, the results tabulated in this paper would be appreciably affected, although the effect of such a variation would be more disastrous to Stadler's theory based on Kick's law than to Rittinger's, and the Stadler units per apparent effective horsepower might in consequence approach perilously near to zero.

Crushing Tests Using Quartz

In this investigation, the desirability of crushing a quartz gangue was recognized as being of more interest to those engaged in practical work; unfortunately, however, none was obtainable locally, and tinguaites were employed for the experiments. It was feared that to some extent the habit of fracture of tinguaites might have prejudicially influenced the results obtained, hence upon the conclusion of the Huntington mill tinguaites tests, a series of grades of quartz sand was prepared and used to determine this point.

Convincing evidence was furnished by the results obtained that the relation between power and crushing indicated by the tinguaites tests remains unchanged when the rock crushed is quartz, excepting that the crushing force is less.

Conclusions

When it is considered that Gates, using a testing machine, obtained similar results for three entirely different rocks, a sufficient amount of experimental data now seems available to justify the following conclusions:

1. That in the reduction of any given rock there is a constant relationship between the power applied and the crushing effected.
2. That Rittinger's theory appears to conform agreeably with this relationship.
3. That Stadler's theory based on Kick's law does not so conform.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Pennsylvania Mine Fire, Butte, Mont.

BY C. EDWIN NIGHMAN, B. S.,* BUTTE, MONT., AND ROLLINS S. FOSTER,† BONNE TERRE, MO.

(New York Meeting, February, 1917)

THE following is a description of the methods used in rescuing men and extinguishing the underground fire at the Pennsylvania mine, Butte, Mont.

This fire, which cost the lives of 21 men, began about 9 p.m., Feb. 14, 1916, at or near a ventilating fan on the 1,200 air-shaft station. This shaft at that time was downcast, while the main hoisting shaft was upcast. Owing to the proximity of the fire to the air shaft, the air in the latter soon became heated, and as the fan on the level was not running—the power having been cut off shortly after the outbreak of the fire—the air shaft became upcast to the 300 level. At this point a blanket of cold air from the surface caused the smoke and gases to be drawn in on that level and carried down again through the workings to the 1,200. A short time after the discovery of the fire, water was turned down the air shaft through a series of pipes around the collar of the shaft, which caused the smoke to recede below the 300, so that within 2 hr. it was possible to search that level for the missing men.

From the above it will be apparent why the air shaft was upcast instead of downcast as under normal conditions, and why men were suffocated at considerable distances from the seat of the fire. The smoke and gases that came up the main hoisting shaft did not diffuse through the workings to any extent.

There are doors on all levels in the connections to the main hoisting and air shafts, but in the confusion and running around many of these doors were left open, and this undoubtedly caused the smoke and gases to spread more rapidly than they would otherwise have done. It was necessary for most of the men on the various levels to pass close to the air shaft in going to the hoisting shaft, and when they encountered the smoke and gases in that vicinity a number turned back and climbed down with the direction of the air currents and were then suffocated, while all those who passed through the relatively short pall of smoke to the main shaft were saved.

* Fireboss, Anaconda Copper Mining Co.

† Late Safety Engineer, Anaconda Copper Mining Co.

There were 220 men in the mine that night, and all but 25 were hoisted to the surface within 30 min. after the discovery of the fire. Of the 25, five went out through the Tramway mine, and one through the Mountain View mine, while the remaining 19 were missing. Their bodies, with two of a rescuing crew, were found later. All the men who came to the surface through the main hoisting shaft were able to go to their homes, with the exception of two, who were sent to the hospital but suffered no ill effects.

DISCOVERY OF THE FIRE AND RESCUE WORK

About 8:50 p.m. of Feb. 14, two shift bosses who were on the 1,600 and 1,800 levels detected the odor of smoke. One immediately went to the fan on the 1,600 station and discovered smoke coming down the air shaft. He then went to the 1,200 level. The other, whose run included the 1,500 and 1,600 workings, sent men to notify others in that part of the mine. About the same time, men were sent through the remaining workings to notify every one to come to the stations and be hoisted.

The men were notified as follows: A station tender notified the shift boss for the levels above the 600, who went to the 1,000 and notified the assistant foreman and another shift boss. The latter were conversing with two miners who were sent to warn two others that were working on the 1,000 sill. The assistant foreman and the shift boss then climbed through 1,284 manway, which was the only one open to the 1,200, and walked out on the sill toward the air shaft. They opened the inside one of two doors in the air-shaft crosscut (1,266) and encountered a wall of smoke through which they could not pass. They then went out to the main shaft, were hoisted to surface and immediately had water turned down the air shaft.

No men were working between the 1,000 and 600 but a large number were employed in the west stopes above the 600. Two skinners were sent to warn the men in these stopes, but encountered so much smoke in the vicinity of the air shaft on the 600 that they were obliged to turn back. They then climbed up a manway to the 500, went back to the top of these stopes and down into them, warning the men as they met them. On the first floor one skinner met eight miners; he told them of the fire and said that they could not get to the main shaft on the 600 level but should follow him to the 500. He said later that the men seemed to realize the serious nature of this news, but for some reason failed to follow him. Without delay he returned to the 500 and lined up nine men who had hesitated to enter the thicker smoke in the direction of the air shaft, and led them to the main-shaft station. These men were then hoisted safely to the surface, though the skinner was unconscious when he reached fresh air.

The other skinner later climbed down through these same stopes and

also met the eight men who had been warned, who by this time had gone from the first floor to the 600 sill. He also told them that they could not possibly reach the main shaft through the 600 level but must climb immediately to the 500. No one made any reply or attempted to follow him, so he returned to the 500 and proceeded toward the main shaft until he had reached the manway through which he had climbed from the 600. Once more he climbed down through this manway and attempted to go in on the 600, but found the smoke thicker than before and noticed that three horses in the barn were dead. He again returned to the 500 and found four men who had come out of the stopes above the 500. These he took to the main hoisting shaft and was hoisted to the surface with them.

About 9:15 p.m. a miner on the 300 station attempted to go in on that level to warn seven men who were working in stopes tributary to that level, but was stopped by smoke. He then went to the surface and down on the cage to the 300 station in the air shaft, with a shift boss and two others, but they found the smoke so dense that they were obliged to return to the surface.

After the men had been hoisted from the mine, which was about 9:30 p.m., the mine foreman, the foreman and assistant foreman of the Tramway, and others, kept going to the stations on all the levels above the 1,200, hoping to find some of the missing. About 9:45 p.m. a shift boss came to the surface and reported that he thought he had seen two men near the 600 station. The assistant foreman of the Pennsylvania and a volunteer from the St. Lawrence mine put on oxygen helmets and went to the 600 in search of these men. They failed to return in 20 min. and others went in search of them. They were found about 400 ft. from the station stretched out in a drift with their helmets still in place. They were removed to the surface in about 20 min., where every effort was made to revive them by both the Pulmotor and the Schaeffer method of resuscitation, but without success. It cannot be positively stated, but it is thought that the oxygen tanks of these helmets were partially depleted and that they were not aware of it and did not examine the tanks when they took the helmets.

About 11:30 p.m., the air was clear on the 300 level and men climbed down the air raises from the surface without helmets; in a crosscut which was only about 100 ft. from the air raises they found the bodies of seven men. Resuscitation by the Schaeffer method was employed on the warmest bodies, but without result. About 4 a.m. the following day these bodies were brought to surface and no further effort was made to find more of the missing men until the air shaft had been made an upcast. By 3 p.m. of the 15th the upper levels were clear enough to begin a search for the missing men. About 2 hr. later the bodies of two men were found with their arms around each other on the fourth floor of a manway

above the 800. Two more were found farther up in the same manway and two others on the sill near by. These were six of the eight who had been working in the 600 west stopes and who had been warned by the skimmers. The bodies were removed to the surface by 10 p.m.

About 1 a.m. Feb. 16, the second day after the fire, the bodies of the other two men from the 600 west stopes were found near the bottom of 1,284 manway, together with those of four men who had been working on the 1,000. Because of the smoke these bodies were hoisted through this manway to the 1,000 and taken to the surface from there. On account of the poor condition of this manway, it was 2 p.m. on this day before the bodies reached the surface.

From the foregoing, it is evident that all the men in the mine were warned, except the seven who were working in the vicinity of the 300 level. Of the 19 who remained in the mine, it is certain that 12 would have been saved had they done as they were instructed.

FIGHTING THE FIRE

Preparations

As noted above, the fire broke out at or near the ventilating fan on the 1,200 air-shaft station. The air shaft was downcast because this fan, which had a capacity of 55,000 cu. ft. per minute, and a smaller one of the 1,600 level, were drawing the air down and forcing it into the workings below. The workings above the 1,200 were ventilated through a series of air raises, down which air was forced by a surface fan. Shortly after the fire started the power was turned off the fans on the surface and the heat of the fire turned the air shaft into an upcast, igniting the timbers. The rapidity with which the fire traveled is shown by the fact that when rescue parties entered the crosscut to the air shaft on the 1,000 level about 3 hr. later, they found the shaft timbers already ablaze. The smoke and gases traveled with great rapidity, for although the adjoining mines were immediately notified of the fire, four men in the St. Lawrence, one in the Mountain View and one in the Tramway mine, were overcome before the doors and bulkheads in the connections to these mines could be closed.

The turning of water into the air shaft immediately after the fire was discovered was the only attempt made to fight it until a suction fan had been installed on the surface at the air shaft and all the bodies removed from the mine, which was on the afternoon of the second day.

As soon as the fire was discovered, preparations were made to insure the safety of the adjoining mines and to facilitate fire fighting by changing the ventilation, i.e., the air shaft was to be made an upcast and the main hoisting shaft a downcast by the use of a fan on the surface. To further safeguard the adjoining mines, as well as to hold back the smoke

and gases, concrete bulkheads with iron doors were at once built in all open connections to the Pennsylvania from other mines. Twenty or more of these bulkheads were built.

The air shaft carried the water, air, electric light and power lines—with the exception of one power cable to the pumps on the 1,800, which was taken down the main shaft—and as these were destroyed it was necessary to replace them before any fire fighting could begin. Work to install these new lines was commenced on the 15th, but it was some days before they were completed, for an old 4-in steam line had to be first removed and there was no manway or pipe compartment in the main shaft other than a space about 6 in. wide between the end-plate and the "chippy" guide center. However, while the fire fighting was going on, this space was enlarged to 30 in. in the clear from the 1,400 to surface.

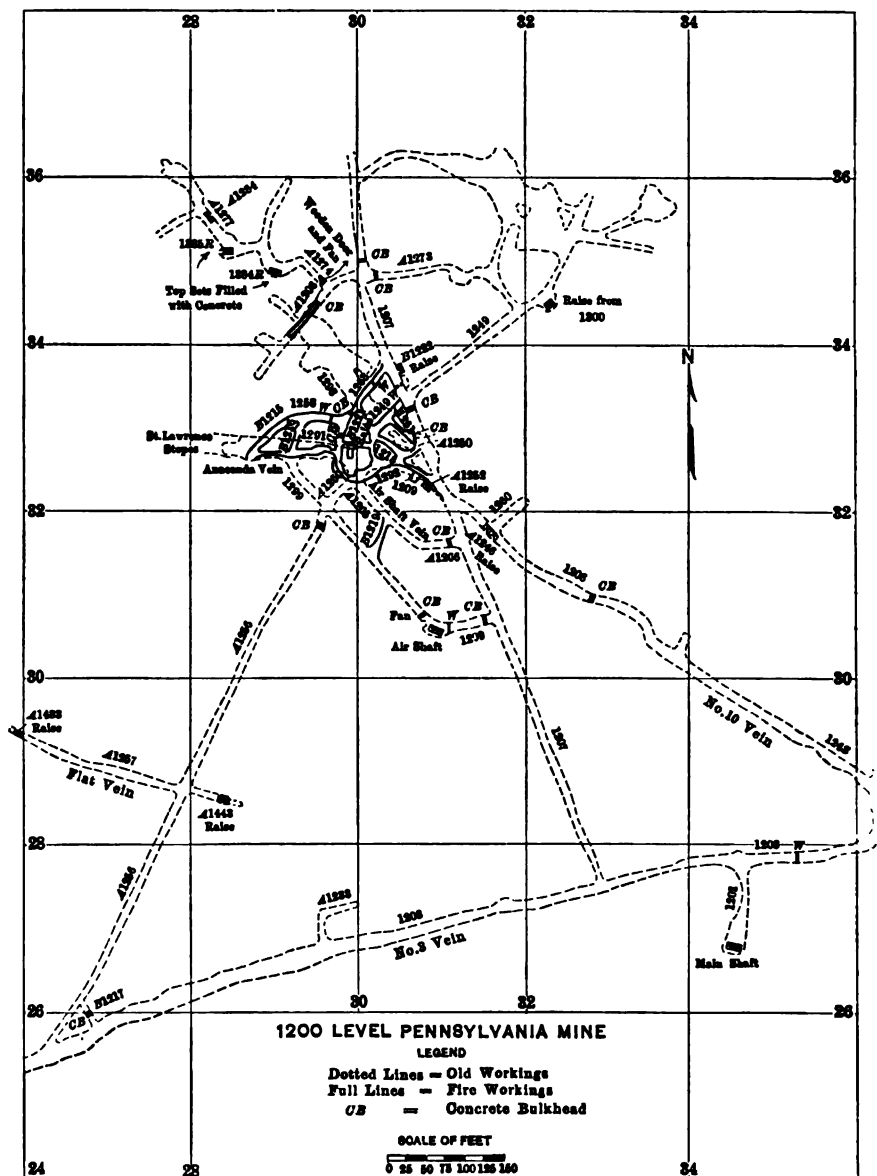
By the morning of the 15th, foundations for fan and driving motor had been placed at the mouth of a short fan drift, which connects with the air shaft about 25 ft. below the collar, and by noon this fan was installed and running. The fan used was a No. 13 Sirocco, whose sea-level capacity is about 83,500 cu. ft. per minute, but at this altitude (5,700 ft.) is somewhat less. So that there might not be any interference with the ventilation, a second of these fans having both motor and engine drives was installed, which could be operated independently of the first fan.

Locating the Fire

Up to the time the fan was started the air shaft remained downcast to the 300 and upcast from the 1,200 to that level, but it became entirely upcast very quickly and the main hoisting shaft a strong downcast, though it later became less so because of the caving of the ground around the air shaft above the 1,200 station. The effect of this change of air was at once noticeable, for that afternoon, the 15th, it was possible to go anywhere without the use of oxygen helmets. The extent and location of the fire could then be determined.

After the reversal of the air currents on the 15th, parties entered all the levels and found that the fire centered in workings north of the air shaft on the 1,200 level (see Fig. 1), and west of the main north-south crosscut (1,207). Also, that as far as could be determined it was only on the sill, in the stopes above, and in the drifts and crosscuts connecting to the air-shaft crosscut (1,299). These workings were connected through stopes with those of the St. Lawrence mine on the west, but were not accessible because the latter were worked out and filled with waste. The only other possible way of getting to that part of the 1,200 sill was by coming in from the Flat Vein through A 1,256 crosscut.

On the 15th, a party went in on the 1,400 sill and climbed up through the raise on the east end of the Flat Vein workings to the 1,200, but were



able to get only part way to A 1,256 crosscut because the timbers at the junction of the vein and crosscut were already on fire. That night helmet men also climbed through this raise to the 1,200, but by the following noon the fire had reached the 1,400 sill and no further investigations could be made. At the ends of the workings on this vein on the 1,400, pillars had been left which had some drifts through them. Solid concrete bulkheads were at once erected in these drifts and the fire sealed off. This was the most important work done in the first days of the fire, for, on the 1,400 sill the Flat Vein is separated from No. 3 Vein by a short-timbered crosscut and the latter vein had been stoped from the Silver Bow on the east to the St. Lawrence workings on the west. Also, this vein crosses the main haulageway near the main hoisting shaft and had the fire penetrated these stopes it would have been impossible to get to the original fire. It is well to note that the Pennsylvania workings are as a rule very dry and the upper workings are all square-setted and contain many timber bulkheads.

The west side of the fire was not accessible and all that was known of its extent and location was that it was to the west and north of the air shaft, in the Anaconda, Air Shaft and No. 10 Veins as well as in the Flat Vein, which is about 400 ft. south of the Anaconda vein and is approximately parallel to it. It is worthy of note that the Flat Vein and the crosscut to the air shaft are about 350 ft. apart and that the connecting crosscut (A 1,256) was not timbered and the only inflammable materials therein were the track ties and the insulated electric light wire with the occasional spreaders from which they were hung.

Methods of Fire Fighting

The methods used were in general those in use at the Mountain View fire of 1913.¹ Drifts and crosscuts were driven from the nearest accessible workings to the fire and as much as possible of it was extinguished with hose lines and by pouring water into the workings above the sill and other inaccessible places by means of diamond-drill holes. The air currents were controlled with bulkheads, doors and brattices, so that the men would always be working with the air; i.e., so that the smoke and gases would be carried away from them. As far as possible the air pressure was maintained at a point where it was just sufficient to hold back the gases without supplying fresh air to the fire. All places were securely timbered and air and water lines were kept in all faces. As many men as possible were employed in all the headings and frequent reliefs were given. The men worked 8-hr. shifts.

Where the gases were too strong, helmet men were employed. The helmets used were Draeger, 1909 and 1911 models. A number of extra

¹ C. L. Berrien: Fire-fighting Methods at the Mount View Mine, Butte, Mont. *Trans.*, vol. 52, p. 534 (1915).

sets and three $\frac{1}{2}$ -hr. helmets, as well as a good stock of accessories, were kept on hand at all times. The supplies were brought as needed from the company's rescue station at the Anaconda mine. Concrete for bulkheads was mixed on the surface in a motor-driven mixer and sent into the mine in regular mine cars. About 600 tons of concrete were used during the fire.

Fire fighting did not begin in reality until the 17th, as the water and air lines were not completed to the 1,200 level; the work of the first few days was limited to the cleaning up of openings and the building of bulkheads. After building the two bulkheads on the 1,400, others, having iron doors and 2-in. and 3-in. pipe connections, were built just off 1,207 crosscut in the air-shaft vein crosscut (1,299) and in the air-shaft crosscut (1,266). These two were very important, for they made 1,207 safe up to 1,209 and 1,250, where the fire was burning strongly, and they also prevented the air from short-circuiting to the air shaft. Many other bulkheads, the locations of which are to be seen on the accompanying map (Fig. 1), were constructed as required in the later work.

Mining

In driving the headings great difficulties were encountered because of the gases and smoke, the heavy ground and the water pouring down from the diamond-drill holes. Wherever filling was met, spiling or forepoling was necessary and in many places breast boards were also required.

On Feb. 18, drifting was begun in 1,209 (No. 10 vein) and in A 1,250 (Anaconda vein). As the timber had burned out, both of these places were caved. An added difficulty was met when 1,292 crosscut was started from their intersection. Here the old sill was about 25 ft. wide, all the timber was burned out, and the back could not be seen because of the smoke, while the hanging wall (dip about 60°) would slab off in large pieces and crush the timbers. It was necessary to retimber this place several times during the fire fighting. It was also necessary to crib over the sets to the back as they were put in. One-inch pipe lines were connected to the fire hoses and thrust ahead to reach the hot ground and burning timbers. Temporary canvas brattices—and later, wooden doors and concrete bulkheads—were built to control the air pressure so as to hold back the gases.

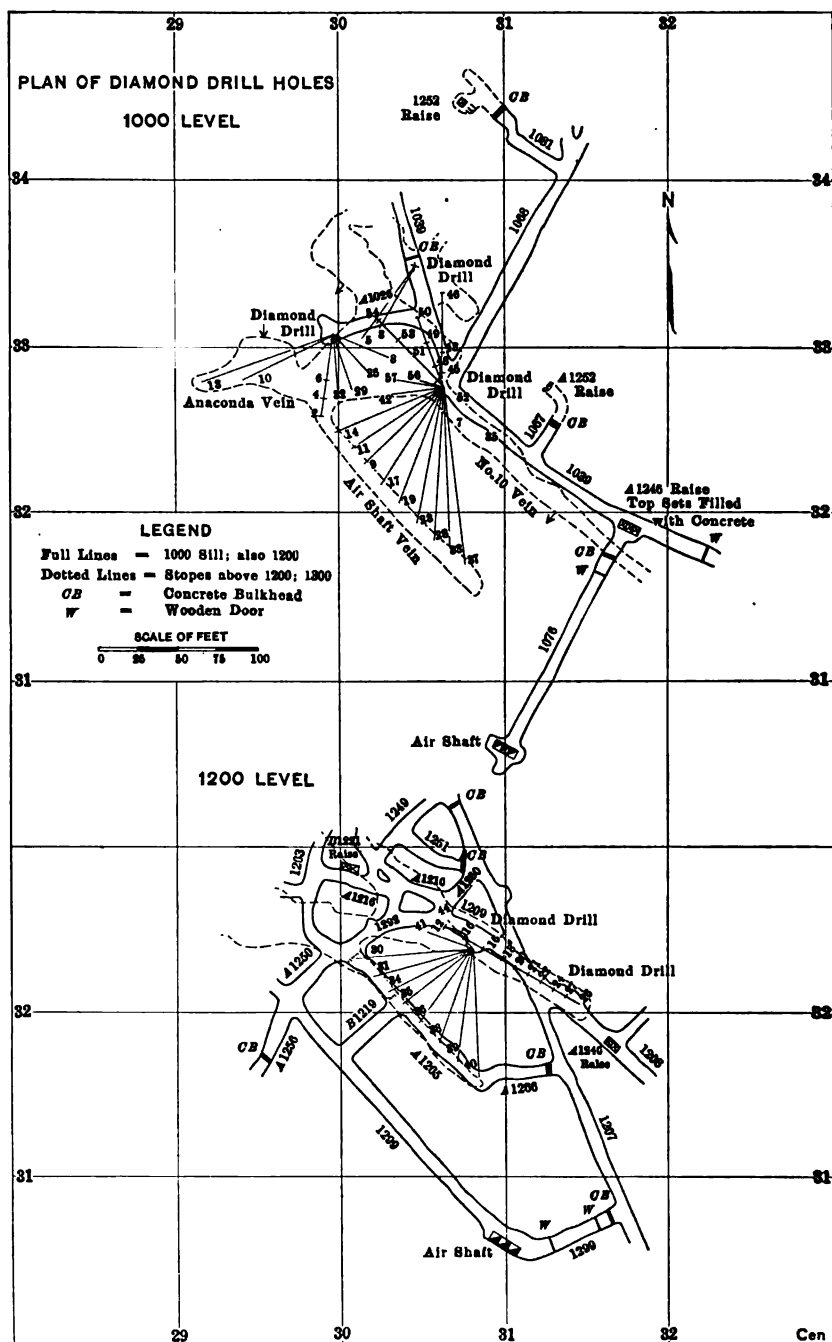
A few days later a drift (1,249) was started north of the above and driven west through a pillar and into the workings on No. 10 vein. In the solid, rapid progress was made but, as was the case in the other headings, extreme difficulty was experienced in getting through the old gob. Charred timbers were found when the west or hanging-wall side was approached. A large amount of water from the diamond-drill holes made it very hard to hold the drift open. At times there would be a run of ground of about the consistency of thin mud which would come

out in the drift for some distance, and several times the men had to run out to avoid being trapped in the mud. Whenever these flows occurred there was an influx of strong sulphur smoke which would drive the men out. This became of such frequent occurrence that only helmet men were employed there. This heading was stopped when the hanging wall was reached (39 ft. advance in 19 days).

On the same day a similar and parallel heading (1,263) was started just north of the above. Rapid progress was made here, the average daily advance for 40 days being 4 ft., including the time lost while two concrete bulkheads were being built. In passing through the gob, no evidence of fire was encountered, other than an inconsiderable amount of gas. Some water from one of the drill holes was coming down on the foot wall. At the junction of 1,263 and the Anaconda vein, some fire was seen, though the ground near by was thoroughly wet down through the drill holes. Having found fire here it was imperative to have further openings to the west; accordingly a foot-wall lateral (B 1,253) was started between the two bulkheads in 1,263. This was driven in the solid because of the greater rapidity of advance possible there, but was not to take the place of an opening in the vein workings themselves. Such an opening was made later. From B 1,253, two crosscuts were driven through the Anaconda vein gobs, but no fire was encountered. Work was resumed in 1,263 on Mar. 10. This place was a difficult one in which to work, because it was wet and because a great deal of gas, chiefly SO_2 , was present. The solid ground was so hot that it could not be touched, and blasting was delayed until the holes could be cooled down with water jets.

While this work was in progress it was found that the pressure from the air column in the downcast hoisting shaft was not sufficient to hold back the gases in all the headings, hence it was necessary to reduce the number of splits from the main air course, 1,207. This was accomplished by building a bulkhead in 1,207 just south of 1,249. Two canvas brattices were used until the bulkhead was completed. An immediate improvement was noted in the two splits, A 1,250 and 1,209. To maintain a counterbalancing pressure in the other workings, 1,249 and 1,263, a No. 2 Sirocco fan was placed in the frame of a wooden door on the west side of A 1,208 in A 1,274. By varying the opening of the door itself, the proper pressure was kept up. In order that only fresh air would be drawn in by this fan, two raises from the 1,300 were sealed off and the air brought down 1,284 raise from the upper levels.

On Apr. 5, after many crosscuts and drifts had been driven (see maps) a quantity of burning timber fell behind the bulkhead across A 1,250 at the junction with 1,207. This was quickly extinguished and was the last fire actually seen.



WORK ON THE 1,000 LEVEL

The fact that the fire was above the 1,200 sill having been determined, it followed that the best point from which to attack it was the level next above. Also, it was necessary to prevent the fire from getting to the upper levels. The first level above the 1,200 is the 1,000, the distance between the two being about 140 ft.

Other than in the air shaft, no fire was seen on the 1,000, but great quantities of gas filled the sill. A week after the fire had started, and on the fourth day of work in 1,067 crosscut (Fig. 2) a gas sample, which was analyzed by the Bureau of Mines, showed an oxygen content of only 14.58 per cent. while it contained 0.68 per cent. CO. A candle would burn here notwithstanding the low oxygen content, but that, with the large percentage of CO, made this an extremely dangerous atmosphere. This would indicate that the burning of a candle is not a test for a respirable atmosphere. This gas came up chiefly through two raises, A 1,246 and A 1,252. These were, therefore, sealed off with concrete bulkheads. Several other bulkheads were also constructed. The chief work done on this level was that of diamond drilling.

Diamond Drilling

As mentioned previously, one of the most important methods used in extinguishing the fire was that of turning water into the burning areas through diamond-drill holes.

Two drills were obtained on the 18th, but drilling was not begun until the 21st because the conditions existing on the level made it impossible to place them in the positions chosen. The first drill was placed in 1,039 crosscut just north of A 1,026, which place was directly over the No. 10 vein stopes and farther to the north than any fire had been seen on the 1,200. The second was placed at the end of A 1,026 crosscut. Considerable difficulty was experienced in getting the drills into place and in operating them, for the level was very gassy and there had been no time to prepare stations for them. At the first two setups it was necessary to use rods from 12 to 15 in. in length, and also to drill holes in the back in line with the drill holes, so that the rods could be raised and lowered. Furthermore, the waste water backed up so that the crosscuts were miniature lakes for some distance. Not alone were the setups inconvenient, but the nature of the ground itself made drilling very difficult. Practically all the holes from the 1,000 penetrated the Rarus fault, which is a wide complex fracture zone of finely crushed and altered granite with included blocks of sharp, rough quartzose vein matter. In many of the holes two standpipes were necessary.

The first holes were drilled with the idea of cutting off the spread of the fire, particularly on the north and west where the stopes connected

with those of other mines and were inaccessible. The remaining holes from the 1,000 were drilled at intervals over the fire area, as required by the conditions obtaining on the 1,200, and so that all portions of the stopes would be thoroughly wetted down, water being turned into them as fast as they were completed. In all, 37 holes having an aggregate length of 3,909 ft. were drilled from the 1,000 level.

On the 1,200 a series of short holes was drilled from 1,208 into the stopes on No. 10 vein below the 1,200. No traces of fire were found with these holes. Also, a series of holes was drilled from a station in 1,209 to the air-shaft vein below the 1,200, and a few into the upper floors of the stopes on the Anaconda vein below this level. None of these showed any evidence of fire having burned below the sill, even where it had been most intense above. By the time these holes were finished, the 1,200 sill had been thoroughly opened up and it was considered unnecessary to do further drilling. Twenty holes with an aggregate depth of 984 ft. were drilled from the 1,200.

WORK ON THE 1,300 LEVEL

To determine the lower limits of the fire, repair work had been started on this level about Mar. 1. Under very difficult conditions 1,329 and 1,330 crosscuts were driven across the gobs of No. 10 and the Anaconda veins. No signs of fire were found in these crosscuts, though considerable gas and much water were present at all times. It was later decided to connect these workings with the main hoisting shaft. The 1,311 drift on No. 10 vein was then started. Working 8-hr. shifts for a part of the time and 6-hr. shifts for 8 days, this heading was advanced 370 ft. at an average rate of $11\frac{1}{2}$ ft. per day. By the time this drift had reached the old workings, conditions had so improved throughout the fire district that it was not considered necessary to connect at once with 1,329; the heading was stopped temporarily and a crosscut driven to the air shaft.

WORK ON OTHER LEVELS

Elsewhere in the mine the only work done was that of repairing and putting the mine in shape for the resumption of mining. This included the retimbering of the air shaft below the 800 and the pumping out of the three bottom levels which had been flooded with the water used in fighting the fire.

RESULTS OF THE FIRE FIGHTING

As mentioned above, the last fire was seen on Apr. 5. Since that time the air shaft has been retimbered, the 1,200 sill and stopes including the Flat vein have been thoroughly reopened and no trace of smoke or gas has been found, proving that the fire has been extinguished.

ORIGIN OF THE FIRE

The true origin of the fire is not known and probably never will be. Three possible causes have been advanced: (1) That a lighted candle was left on some timber; (2) that it was an electrical fire; (3) that it was of incendiary origin.

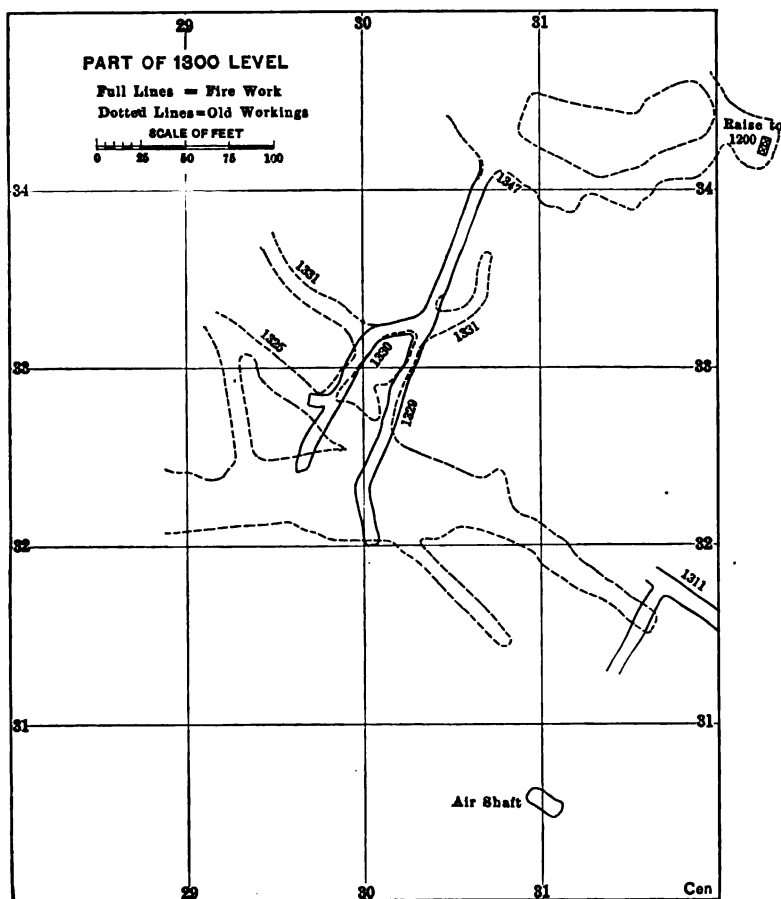


FIG. 3.

As to the first, it is known that it was practically impossible to carry a lighted candle through the double doors near the air-shaft station because of the strong draft induced by the fan there. That it was an electrical fire has been doubted by some, owing to the fact that the fire apparently jumped several hundred feet through the untimbered A 1,256 crosscut to the Flat vein workings. Because of this, it is argued that the fire must have started in two places and was therefore of incendiary

origin. But, about 2 hr. before the discovery of the fire the fan at the 1,200 air-shaft station had been oiled by a shift boss, and at that time everything was all right. Had the fire started at or near the fan the smoke would have been carried rapidly down through the Flat vein workings, as the air was being forced in that direction, and would have been noticed by the men at work there before it was noticed elsewhere. Also, it is argued, the fire could not have been electrical, because the fan must have been running in order to carry the fire to the Flat vein stopes. Had the fire started from a short-circuit at the fan, it would not have been possible for the latter to run long enough to do that. However, about seven o'clock on the night of the disaster, the power went off and came on again with a big surge. The voltmeter at the main power station registered 2,800 volts, or 500 above normal. This overload caused the circuit-breaker on the fan circuit at the Pennsylvania mine to kick out. An electrician from the Leonard mine substation was sent to throw it in again and when he did so the lights glowed for a short time instead of burning brightly, indicating some trouble on the line. The power was then left on till shortly after the discovery of the fire. The above are the important facts in connection with the origin of the fire and they are too meager to justify the acceptance of any of the three probable causes advanced.

METHODS OF PROTECTION AGAINST FUTURE FIRES

The collars of all shafts have been provided with 2-in. sprinkler lines which will throw water into all compartments. About 18 months ago, at the Mountain View mine, a fire at the collar of a downcast shaft destroyed the loading platforms, headframe and engine room but was extinguished by water from two 2-in. hose lines before it had burned more than 25 ft. below the collar. It was concluded from this experience that the sprinklers as mentioned would afford adequate protection.

In addition to the sprinkler lines, it is also recommended that the upper portion of all shafts be made of some non-inflammable material, say concrete. When the above-mentioned shaft was reconstructed, the upper 30 ft. was made of concrete and the shaft timbers for about 60 ft. were covered with asbestos paper and steel roll cap.

All hoisting shafts are to be downcasts. This was to have been done at the Pennsylvania mine as soon as the 1,600 and 1,800 levels had been connected to the air shaft. At the time of the fire, raising from these levels was in progress.

At intervals of 400 ft. in all shafts, water tanks with a capacity of 3,000 gal. have been erected. These are connected with float feeds to the water columns and are always full so that they may be used in case of emergency.

Downcast shafts are to carry all pipe, electric light and power lines.

There are water lines in all the main drifts and crosscuts on the different levels. Also, water may be turned into the air lines by means of interchangeable connections between them. In connection with these, it is believed that it would be advisable to have connections for hose and pipe at all manways and chutes and an automatic or manually operated sprinkler system on shaft stations and at the tops of the principal manways or raises. Tees with hose connections and valves should be placed in all water lines on the sills at intervals of, say, 100 ft. This has been done on some levels at one of the mines.

Hose, pipe and monkey wrenches, hammers and a sharp axe or two, should be kept for fire purposes only, on shaft stations and at other critical points on each level, depending on the extent of the workings on that level.

Watchmen or "firebugs" are employed at all the mines to examine all working places after the shifts have gone off. Abandoned workings are also examined occasionally to guard against incipient fires.

Many concrete bulkheads with iron doors now exist, but enough more should be used to make possible the isolation of any required section of the mine in so far as the workings will permit.

All surface and underground fan motors have been provided with the necessary protective apparatus. All underground fan stations have been lined with expanded metal lath and concrete. The fans are oiled at stated intervals, and cared for by either shift bosses or the mine electricians. It is recommended that the starting boxes, fuse blocks, etc., be so inclosed that they cannot be tampered with by any other than the persons responsible for their operation. This should also apply to switches and circuit-breakers on the trolley lines. Buckets of dry sand or Pyrene fire extinguishers should be kept near all electrically driven machinery.

Metal receptacles should be provided for oily waste. Recently a fire was started on a shaft station by a pile of waste, but was discovered before any damage had been done.

There are rigidly enforced regulations in regard to the careless use of candles and to smoking.

Surface fans are to be used for exhausting only. A short time ago a fire at the collar of a downcast fan shaft burned the shaft timbers for some distance even though the fire was discovered practically at its inception. Not alone is the danger that of burning out the shaft timbers, but the smoke is forced into the workings below and may cause the suffocation of workmen.

A telephone system is practically a necessity in any large mine. These are in use in many of the Butte mines. In general, the phones are on the shaft stations only, but it would be a great advantage to have others at important places on the levels. In case of danger, the men

could be promptly warned by the use of the phone and the loss of life thus avoided. In connection with the phone system, or without it, there should be some means of warning men to come to the shaft stations or to leave the mine. This can readily be done in electrically lighted mines by the use of a special flash signal to indicate the existences of danger and then, by giving the station signal, to show where it is.³ Carmen or dumpers would notify the men working in raises, winzes or stopes. The use of such a signal is not customary in Butte, though it is used elsewhere. Connections to other mines and to shafts are plainly marked by signs on the main traveling ways in the Butte mines.

The systems of mining now being tried out in Butte make use of much less timber than the ordinary square-set method and, therefore, decrease the danger from fire.

In some of the mines, when working near fire zones, a row of timber has been removed from hanging to foot wall of the stopes at some convenient place, the floor mined out and this space filled with fine waste which is thoroughly wet. This operation is repeated at the same place in each successive floor until the level above has been reached. This makes a vertical wall about 10 ft. thick from level to level, and it has proved cheap and efficient. This practice originated at the Anaconda mine.

Oxygen helmets and lungmotors or pulmotors should be provided at all mines, and men trained in their use should be available. This is well taken care of in the Anaconda Copper Co.'s mines by two rescuestations and by many men trained in the use of apparatus. These stations are located at the Anaconda and Tramway mines, which are about a mile apart. The stations are very modern and are practically alike in equipment. At the two, there are 14 sets of 2-hr. 1907 type, 10 sets of 2-hr. 1911 type, 13 sets of 2-hr. 1914 type, and 12 sets of $\frac{1}{2}$ -hr. 1914 type, Draeger helmets. The apparatus, as well as all repair parts—of which a large stock is always on hand—is kept in zinc-lined lockers. The oxygen bottles are kept pumped to 130 atmospheres. A large supply of oxygen and several hundred potash cartridges are kept in stock. At each station there are hand- and motor-driven oxygen pumps. In the buildings there are a bath and toilet and a small smoke room. Outside, there is a timber gallery for apparatus practice. At the Anaconda station there are also three pulmotors and one lungmotor. Each station has a light automobile truck which is used for the transport of helmets or supplies to the mines where needed.

³ Since the above was written, all the Anaconda Copper Mining Co.'s mines have been equipped with special oil switches on the lighting lines, by means of which station tenders or other authorized persons may give an "all out of the mine" signal. This consists of nine flashes repeated three times, followed by the signal for the level on which the danger exists.

An attendant is on duty at each station at all times. These attendants work 8-hr. shifts. If the attendant has to leave the station at any time, he leaves word as to his whereabouts with the telephone central.

All the fire work at this mine was conducted under the supervision of C. L. Berrien, assistant general superintendent of mines, H. R. Tunnell, mine foreman, and C. E. Nighman, fireboss.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Evidence of the Oklahoma Oil Fields on the Anticlinal Theory

BY DORSEY HAGER,* TULSA, OKLA.

(New York Meeting, February, 1917)

THE information given in the accompanying table is submitted as evidence confirming the application of the anticlinal theory and the value of geology in the Kansas and Oklahoma oil fields.

The term anticline has been used to define a particular type of arched fold. The anticlinal theory of oil accumulation should, however, be broadened to include any structural fold of an arched type even if not a distinct anticline. A terrace may very properly come under the heading of the anticlinal theory of accumulation, as the folding is closely related to that of the anticline; in fact, in many cases is an arrested anticline, though not completely closed in all directions. Stratigraphic forms due to lensing or irregular cementation of the sands certainly cannot be considered to fall within the anticlinal theory, though the laws of accumulation may be similar. A dome is similar to an anticline and belongs to the same class. A dome is broader in proportion to its length than an anticline. Many anticlines undulate along their axes to form domes. The anticlinal theory in this discussion will then be applied to anticlines, domes and terraces.

In the following list, the writer has tabulated the main oil fields of Oklahoma, and a few recently discovered fields in Kansas. The fields discovered within the past 4 years are listed with the date of their discovery. Under "How found," the writer puts "Wildcat" where a geologist has not been employed, "Geology" where it has been used, and "?" where there is doubt. In some places the geologists have claimed pools that may rightly belong to them, but their claims have been questioned, so that they belong in an unsettled category.

* Petroleum Geologist and Engineer.

Productive Field or Pool*	Type of Fold Domes and Anticlines	How Found	Date
Augusta, Kan.....	5 minor domes on 2 anticlines	Geology	1915
Bald Hill.....	Dome	Wildcat	
Beaumont, Kan.....	Dome	Geology	1915
Bigheart.....	Dome	Wildcat	
Billings.....	Dome	Geology	1916
Blackwell.....	Dome	Geology	1913
Boston Pool.....	Dome	Geology	1912
Boynton.....	Minor domes	?	1913
Cole Pool.....	Dome or lense	Wildcat	1913
Checotah.....	Dome on anticline	Geology	1914
Dexter, Kan.....	Dome on anticline	Geology	1914
Drumright, Cushing Pool.....	Dome on anticline	Wildcat	1914
Dropright, Cushing Pool.....	Dome and terrace	Wildcat	1913
Eaton, Gas.....	Dome	Geology	1916
Eldorado, Kan.....	Domes on anticline	Geology	1915
Foxburg, Cushing Pool.....	Dome on anticline	Geology	1915
Fox, North of Healdton.....	Dome on anticline	Geology	1915
Ft. Smith, Gas Pool.....	Dome on anticline	Geology	
Garber.....	Dome	Geology	1916
Healdton.....	Minor domes on anticline	?	1913
Hominy, Oklahoma Dome.....	Dome	Geology	1916
Ingalls.....	Dome	Geology	1915
Loco Gas Pool.....	Dome	Geology	1913
McMan Gas Pool, Haskell.....	Dome	Geology	1915
Morrison.....	Dome	Geology	1915
Muskogee.....	Domes on anticline	Wildcat	
Newkirk.....	Dome on anticline	Geology	1913
Ponca City.....	Dome on anticline	Geology	1913
Paola, Kan.....	Dome	Wildcat	
Poteau Gas.....	Dome	?	
Quinton Gas.....	Dome on anticline	Geology	1915
Robinson Pool.....	Faulted dome	Geology	1915
Shamrock (Cushing).....	Dome on anticline	Geology	1914
Stone Bluff.....	Dome	Geology	1915
Town of Cushing.....	Dome and terrace	Geology	1914
Vera.....	Dome	?	1915
Virgil, Kan.....	Anticline	Geology	1916
Wagoner.....	Dome	Wildcat	
Wainwright.....	Dome on anticline	Geology	1914
Wheeler.....	Dome on anticline	Geology	
Winfield, Kan.....	Dome	Geology	1916
Yale.....	2 domes on anticlines	Geology	1914

* Pools in Oklahoma unless otherwise mentioned.

Productive Field or Pools (Continued)	Type of Fold Terraces	How Found	Date
Bixby (Franchot Pool).....	Terrace	Wildcat	1916
Catoosa.....	Terrace	Wildcat	1914
Dawson.....	Terrace	Wildcat	
Tulsa.....	Terrace	Wildcat	1913
Glen Pool.....	Double terraces.....	Wildcat	
Jenks.....	Terrace.....	Wildcat	
Jennings.....	Terrace.....	Wildcat	1915
Kellyville.....	Terrace.....	Wildcat	1913
Morris.....	Terrace.....	Wildcat	
Mounds.....	Terrace.....	Wildcat	
New York Pool.....	Terrace.....	Wildcat	1913
Owasso.....	Terrace	Wildcat	
Pumpkin Center.....	Terrace	Wildcat	
Ripley.....	Terrace	Geology	1914
Tiger Flat.....	Terrace	Wildcat	
	Lenses		
Avant.....	Lense	Wildcat	
Coweta.....	Lenses	Wildcat	
Misener Pool.....	Lense	Wildcat	1914
Wigton Pool.....	Lense	Wildcat	1916

Oklahoma Shallow-Pool District

Hard to Define without Definite Detailed Surveys

Productive Field or Pools (Continued)	Type of Fold Terraces	How Found	Date
Bartlesville.....	Terraces, small domes.....	Wildcat	
Bird Creek Pools.....	Terraces, small domes.....	Wildcat	
Childers.....	Terrace, domes	Wildcat	
Copan.....	Terrace, dome	Wildcat	
Dewey.....	Terraces, domes	Wildcat	
Flat Rock.....	Terraces, domes	Wildcat	
Lenepah.....	Domes, terraces.....	Wildcat	
Nowata.....	Domes, terraces.....	Wildcat	
Ochelata.....		Wildcat	
Owen.....		Wildcat	
Wann.....	Dome and terrace	Wildcat	

Unless one has mapped all the old pools, it is not possible to say just how far they are dominated by structure, but from our own work, and from discussions with others, it is safe to venture that nearly all the "shallow-pool" districts, Nowata, Bartlesville, and the Eastern Osage, are controlled by structure; domes, noses, and terraces are the main structural features. The "shallow pools" of Kansas show the same relation. At Peru and at Sedan, domes are evident, though gas occurs high on the domes, and oil well down the flanks, leading some to believe

that the oil occurs in the synclines. I have yet to hear of a well-defined Oklahoma or Kansas syncline carrying oil in the synclinal trough.

At Eldorado, Kan., oil does occur close to the synclinal trough, but it also occurs on top of the dome. The deformation or height of the Eldorado fold is 110 ft. Oil is found on top of the fold, and 80 ft. down from the top. The syncline, however, carries water. Structure influences the accumulation even here.

Conclusions

1. Of the 75 pools listed, which comprise the principal pools of Oklahoma and a few in Kansas, all but four are on well-defined structure.

2. It will be seen that by far the larger proportion of the representative fields listed are of the dome or anticlinal type. Terraces come next in order, while lenses form a small minority. Of the list 45 pools, or 60 per cent., are on domes and anticlines; 15 pools, or 20 per cent., are on terraces, and 11 pools, or 14.7 per cent., are undivided, while 4 pools, or 5.3 per cent., are on lenses. If those fields on undivided terraces and domes (14.7 per cent.) are split equally, which is fair enough, then 7.35 per cent. will be added to the dome, and to the terrace lists; so that 67.35 per cent. are on domes and anticlines, 27.35 per cent. are on terraces, and 5.3 per cent. on lenses, which is a fair ratio for the Mid-continent fields.

3. Also, it is interesting to note that of 45 pools found in 1913 and later, 30 pools, or $66\frac{2}{3}$ per cent., have been opened by the geologists—a truly remarkable record. Of these the geologists have credit for but one terrace.

4. The evidence shows that nearly all the known fields are on well-developed folds. Lensing of the sands and cementation have both been factors in favoring accumulation, but well-pronounced folding has influenced most of the fields.

The brief for the geologist is complete, and I must rest my case feeling secure that the evidence shows: (1) That the anticlinal theory holds in Oklahoma and in Kansas; and (2) that the geologist is playing a most important part in the development of those oil fields.

The writer so far has said little or nothing in this discussion regarding the use of geology in developing the fields discovered. In all the recent "wildcat" pools, the geologist has been of material assistance in guiding development after the first well. This was especially true at Healdton, at Cushing, at Boynton, and other recent strikes. The services of the geologists have grown so well recognized and known, that all the large oil companies of the Mid-continent employ them in locating wells and in examining territory.

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The Significance of Manganese in American Steel Metallurgy*

BY F. H. WILLCOX,† PITTSBURGH, PA.

(New York Meeting, February, 1917)

In Bessemer-steel practice, air is blown through a bath of iron, or projected strongly upon its surface to burn out silicon, manganese, and carbon. Toward the end of the blow, when the iron is not protected from oxidation by these elements and the excess of air becomes great, there is oxidation of the iron. In open-hearth practice, the same difficulty is experienced with dissolved gas and oxides in the product, though not to as great or severe a degree as in the Bessemer process because the oxidizing conditions grow less severe as the end of the heat is approached, and if the bath is allowed to boil for a time it undoubtedly frees itself from a large part of the gas and oxides which have been absorbed in the earlier part of the process. Basic open-hearth steel is usually assumed to contain a larger proportion of oxides than acid open-hearth steel, but it may be said that all steel, Bessemer and open-hearth, contains more or less oxide of iron dissolved in the fluid metal at the end of the "blow" or "heat."

That certain metals are solvents for their respective oxides has been shown, in the case of copper, by Heyn, and, in the case of iron, by Law. It is indicated that the solubility of iron oxide in iron increases with the purity and temperature of the metal. This confirms what is observed and known by operating men, and these observations may be summarized as follows:

1. Cast iron is rarely subject to oxidation, as is logical from its composition.
2. Normal steel, unless deoxidizers are added, contains blow holes.
3. Steel known to be oxidized is full of blow holes.
4. The addition of deoxidizers prevents, to a greater or less degree, the formation of blow holes in normal steel.

With these observations in mind, it is logical to deduce that the cause of blow holes lies in part in the presence of dissolved oxide of iron in the

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† Metallurgical Engineer, U. S. Bureau of Mines.

molten steel and its consequent reaction with the carbon of recarburizers, when these are added, to form gases. (Another factor in the origin of blow holes is, of course, the solubility of gases themselves in the steel and their consequent separation when the metal solidifies in cooling.) In addition to the propensity of dissolved oxides to form blow holes in the steel, they are harmful in that they produce brittleness in the steel. Also, oxides are at present under suspicion in that they are believed to promote corrosion of the metal.

To produce a dense homogeneous steel, free from honeycombing, there is required above all the removal or suppression of gases; occluded, dissolved or mechanically held. To prevent dissolved gases, or gases formed as a reaction product from separating out at solidification, fluid compression and hot tops have been employed. But average practice is more concerned with the removal or suppression of gases before solidification. The wilder the metal (*i.e.*, the more gases evolved) the more boiling there is and, consequently, the more segregation and blow holes, so that, in extreme cases, the ingot is absolutely unfit for use. Hand in hand with the necessity for removal or solution of dissolved gases is the necessity for reducing the iron oxide dissolved in the steel, because, apart from any probable detrimental effect of the oxide as such, there is the probability of the formation of carbon monoxide or dioxide by the reaction of the iron oxide with carbon. Whether these ends are met by the direct elimination of the gases and oxides, or by their elimination as compounds of the deoxidizer used, or by increased solvent power of the steel for gases, is perhaps immaterial solely from the viewpoint of structural density of the solid product, however important it may be from the viewpoint of immunity from corrosion or gain in dynamic properties. These last two points must be considered when the matter of deoxidizer additions is under consideration, but the purpose of this paper and its length permit only consideration of the value of deoxidizers from the standpoint of removal of gases and oxides.

MATERIALS USED AS DEOXIDIZERS

Manganese

It may be fairly said that the basis of modern steel making lies in the several functions of manganese alloys when added to a steel as carburizers or deoxidizers. These functions may be classified as follows:

A. In foundry practice:

1. Its use as a deoxidizer and desulphurizer.
2. Its use to alter the constitution of grain in the metal.

B. In steel-works practice:

1. Its use as a deoxidizer.
2. Its use to impart certain static properties to steel.

Ferromanganese is the most prominent and widely used of deoxidizers. Added in the form of ferromanganese or speigeleisen, or intermediate products, manganese is readily oxidized and seizes with avidity any oxygen dissolved in the steel as oxide, this property being strong at the temperature of molten steel. For this purpose it is added in proportions varying between 0.3 or 0.4 to 1.5 per cent.; below 0.3 per cent. the steel is liable to be harmfully charged with oxides. The deoxidizing action is usually considered to lie in the greater affinity of manganese than of iron for oxygen, and the consequent seizing by the manganese of the greater part of the oxygen existing as dissolved oxide of iron. The manganese oxide thus formed is insoluble in iron and goes almost entirely into the slag, returning iron to the bath as metal. The usual additions of manganese do not entirely eliminate oxygen from the steel, and probably have a small effect upon dissolved gases. Traces of oxygen are commonly taken care of by the addition of aluminum after the ferromanganese addition, but the scavenging effect of manganese is nevertheless very strong, and it is this action, in addition to the other functions, that makes its use so indispensable. Another action of manganese, to which is due in part its value, is its behavior with sulphur. As steel exists at the end of a "blow" or "heat," the sulphur is present almost entirely as sulphide of iron. Microscopic examination shows iron sulphide to exist in steel largely as films between the metallic crystals, a condition which gives rise to weakness and brittleness of the metal when hot—"red shortness," due to the impairing of the closeness of texture and cohesion of the iron molecules. Manganese additions convert the iron sulphide to mixed sulphides. With the present low sulphur specifications and fast practice, it may be doubted whether very much manganese sulphide separates out from the metal into the slag. It probably remains in very large part in the metal, but is found to exist as sharp or rounded inclusions instead of films. A third property of manganese, which is lacking in the more common deoxidizers to be mentioned, is that it raises the critical temperature to which it is safe to heat the steel, preventing coarse crystallization at high temperatures.

Aluminum

Aluminum possesses to a high degree the ability to remove traces of oxygen from steel, very small quantities, about 0.05 per cent., "killing" the steel. In itself, it is a more efficient deoxidizer than manganese or silicon, but when considered as the sole means of deoxidizing the steel rather than as an ingredient to be used for removing the last traces of gases, there are objections to its use. First, the product of reaction, alumina, is a relatively infusible product, its melting point being higher than the usual temperature of steel, and it may therefore chill and remain

suspended in the metal instead of coalescing and floating or separating out into the slag. If used in a large amount, therefore, it may give rise to a dirty steel and render the mechanical properties weaker on account of the presence of non-metallic impurities. Second, alumina is incapable of eliminating or changing the condition of sulphur, as does manganese. Third, the use of too much aluminum introduces a tendency to form a larger pipe. Aluminum is without question beneficial when used rationally, for several reasons: The heat of reaction is appreciable and possibly renders the steel more fluid momentarily; by "killing" the steel, the metal is quiet and crystallization of the type described as "land locking" or "pine tree" is encouraged, thus decreasing segregation; and because aluminum seems to increase the solvent power of steel for gases, its use prevents blow holes to a marked degree. The point to be carried in mind is the natural limitation of aluminum as a total substitute for ferromanganese.

Silicon

Ferrosilicon is effective for removing dissolved oxygen, even when added in proportions as small as from 0.1 to 0.2 per cent. by weight of the steel. It reduces honeycombing, is said to diminish segregation and to increase the solvent power of the metal for gases. A large pipe is formed in ingots when ferrosilicon is used in excess. Another drawback is that the products of oxidation may remain in the steel should the reaction end product be silica rather than a silicate. Analyses do not show whether the silicon is present in the metal, as a result of additions, as a silicate, silica, or a silicide. Silicon additions have no effect in eliminating red shortness caused by sulphur, or molecular derangements of the metal, to compare with the effectiveness of manganese. An experiment is recorded where ferromanganese was reduced by 50 per cent. and ferrosilicon increased in varying percentages in an endeavor to economize in the cost of manganese additions, but with the result that the metal was honeycombed and permeated with inclusions of slag. Ferrosilicon finds its most effective use in the foundry rather than in the steel works.

Titanium

Ferrotitanium is given an excellent rating as a deoxidizer by some authorities, but is far from recommended by others. It is said to increase ductility, to prevent honeycombing and segregation, and to remove both dissolved oxygen and nitrogen. The removal of nitrogen is probably without question, and since the removal of all gases is necessary to obtain sound steel, there is no doubt of the worth of an element that will remove nitrogen. Titanium does not freely alloy with iron, but is more

likely to be found as microscopic inclusions in the form of nitride or carbide. When used alone, it is not as effective a deoxidizer, as when used with ferromanganese, to replace appreciable quantities of the latter. With ferrotitanium, as with ferrosilicon and aluminum, the use of ferromanganese is usually essential to obtain a dense workable product.

Other Deoxidizers

In addition to the above materials, many other deoxidizers have been suggested and some have been used. Among these are magnesium, calcium, sodium, vanadium, uranium, boron, and alloys of these with manganese, aluminum, silicon, and titanium. These workers have as a general base the idea that the reaction product should be a slag of great fluidity so that it will coalesce into globules of sufficient size to float out of the metal. Among these deoxidizers may be mentioned calcium-silicide, ferroaluminum-silicide, ferrocadium-silicide, ferromanganese-aluminum-silicide, and ferrotitanium-aluminum-silicide. In most of these alloys the various elements reinforce each other and should make powerful deoxidizers, and be preëminently suitable for producing homogeneous steel. Whether they actually are suitable is not known; they are apparently playing little part in present-day practice. The extent to which thorough study of these and similar possible deoxidizers has been carried is apparently small. The importance, interest and value of experiments along these lines should be evident in view of the manganese situation in this country.

Present Scarcity of Manganese in Germany

Germany has relatively scanty resources of manganese within its borders. For that reason it may be of interest to discuss briefly the apparent situation there when the Central Empires are cut off from former sources of supply. From time to time, reports have come from Germany which seem to have great technical importance. For instance, about a year ago a Dr. Schroedter mentioned, as a metallurgical experiment and secret, means by which Germany was able to get along with less manganese than formerly in steel making. Substitute alloys have been mentioned and also a method by which metallic oxides of iron are eliminated from the slag in open-hearth practice before the heat is tapped, thereby freeing the steel itself of oxides. The facts in the case seem to be much as follows: In the year before the war almost twice the normal amount of high-grade manganese ore had been imported from India and Russia. Also, in the spring of 1915, it was reported that stocks of manganese ore had been confiscated at French and Belgian plants. During the last year, manganese has been reported scanty and prices have been high,

though not as high as speculative dealings and scarcity have forced the price in this country. During the past year, piles of slag from old ferromanganese furnaces in Westphalia running from 5 to 14 per cent. manganese, have been drawn upon, and U. S. Consul Albert, of Brunswick, Germany, reports that the village of Adenslidt has been demolished to secure manganese ore running about 22 per cent. manganese. Lately, complaint is reported from Dutch sources in regard to the quality of German steel. It is said that the steel is daily proving worse and is becoming hard and brittle. The deterioration is attributed to lack of skilled workmen and to the lack of manganese.

Present Situation in America

Before the European war, little ferromanganese was produced in the United States except by one large company. The apparent reason was that supplies from abroad had always been abundant and had been offered at reasonable, and sometimes remarkably low prices, \$40 or \$45 seaboard. At the beginning of the war the price was \$68 seaboard for British ferromanganese, and since the war it has touched over \$400 per ton. At present the price is in the neighborhood of \$165. Before the war, independent steel companies depended almost entirely upon importations and upon one large producer in America, the latter having adopted a liberal policy in sales and in tiding competitors over periods of temporary stringency. Within the last 18 months, several new producers have entered the field.

TABLE 1.—*Production and Imports of Manganese Material*

	Production			Imports	Apparent supply
	Ferromanganese, Tons	Spiegel, Tons	Total, Tons	Ferromanganese, Tons	Ferromanganese, Tons
1912	125,378	119,506	244,884	99,137	224,515
1913	119,495	126,081	245,578	128,070	247,565
1914	106,083	100,365	206,448	82,217	188,300
1915	146,542	93,282	239,824	55,201	201,743
1916 (estimated)	195,605	181,327	376,932	73,031	268,636

TABLE 2.—*Rate of Steel Production*

	Open Hearth	Bessemer	Other	Total
1912	20,780,000	10,327,000	142,600	31,251,300
1913	21,599,000	9,545,000	155,200	31,300,800
1914	17,174,000	6,222,000	117,400	23,513,000
1915	23,679,000	8,287,000	184,600	32,151,000
1916			 (Estimated)	42,000,000

From Tables 1 and 2, it may be estimated that the average consumption of ferromanganese for the years 1912, 1913, and 1914 runs about 19 lb. per ton of steel. This is deduced by estimating that 30 per cent. of the Bessemer production for these years is recarburized and deoxidized by spiegel instead of ferromanganese. Applying the same estimate to 1916, we have 360,000 tons as the indicated consumption of ferromanganese for 1916.

Such an estimate is an approximation and the indication of a deficit of 100,000 tons of ferromanganese is, of course, not borne out by the market price at the present moment. Inasmuch as the method of approximation is applicable to normal years, the apparent deficit shows that unusual economies are being exercised in the use of 80 per cent. ferromanganese. Intermediate products, between 20 and 80 per cent. manganese, ferrosilicon, silicospiegel and special deoxidizers are being used liberally, so that the apparent lack of ferromanganese shown in the above analysis is compensated for by these substitutes and economies. From all indications—*i.e.*, market prices, estimates, and quality of steel—there is no danger in the present ferromanganese situation in regard to a sufficient supply of the alloy. Notwithstanding this fact, the general situation regarding the supply of manganese for the country is not without disturbing features, inasmuch as we are dependent upon British alloy and upon foreign ore for this ingredient in steel making.

The production of manganese ore in the United States for the year 1915 was 9,651 tons. This was in the face of an unprecedented demand, a shortage in the supplies of foreign ore and alloy, a price per unit of manganese that reached 97 c. (the average price per unit for the preceding 5 years was 12½ c.) and following extensive exploration work. This tonnage was scattered through 10 States and came from 34 producers, and should be contrasted with our importations of foreign ores.

TABLE 3.—*Imports of Manganese Ores*

	Tons
1914.....	288,706
1915.....	206,859
1916.....	492,850 (apparent rate)

The bulk of this foreign ore in the present year is coming from Brazil. In 1913, one-third of the ore came from Brazil whereas this year nine-tenths is coming from that country.

It is evident that we do not produce this ore in quantities commensurate with our consumption. In fact, our production is negligible; how negligible, may be realized from the fact that based upon the amount of ore necessary to furnish the apparent supply of 260,000 tons of ferromanganese available this year, our production of ore amounts to 1½ per cent. Our contribution is equivalent to about 2 per cent. of the ore required to furnish the apparent production of 195,000 tons of ferromanganese in

this country. This is on the basis of last year's production. This may be increased this year, but it is not possible that it can be increased to a degree commensurate with our requirements. No deposits of rich manganese ore that hold any promise of furnishing the needed quantities of this material have been developed to date in this country. The situation can not be better expressed than in the words of an editorial in a recent number of an iron trade journal: "Insecurity lurks behind these facts (apparent supply of manganese). To supply this most necessary element in making steel, our own territory has furnished scarcely any ore. The iron industry is dependent absolutely upon foreign supplies and sources for manganese, without which thus far good steel is impossible. Were war to break out between us and a first-class power capable of controlling in part or entirely the high sea, our situation would be precarious, for no nation can defend itself without steel. Assiduous efforts to discover manganese ore in this country have been unavailing. We are dangerously dependent upon manganese. It is to be hoped that metallurgical science will discover a substitute."

This situation is not one that carries immediate danger. It is certainly comparable, however, to the potash and nitrate situation and the suggested production of these materials in this country. It may well be more serious than either of these two examples. In a national emergency we could economize and to some degree dispense with the one and manufacture the other, but we can not entirely dispense with nor manufacture manganese, nor can we economize in its use to the degree called for by our inadequate production and great demands.

CONCLUSION

These are two ways to meet such a situation as has been presented in the above paper. The first suggestion is an obvious one, and is that the manufacturers of the country accumulate a sufficient reserve of high-grade manganese ore to tide the steel industry over at least a year's stoppage in supply. The amount of capital that would be tied up by the investment in such a tonnage of ores at the present prices and scarcity of freight-carrying bottoms is the answer to this proposal at the present time, and may be a deterring factor for all time to come, because when the present war is over and peace established, the present very apparent need of some such precaution will naturally become less and less evident. The second suggestion is that an attempt be made to develop a substitute alloy which can be used in place of or, more probably, with ferromanganese in such proportions as to conserve the manganese.

As far as can be learned, substitute deoxidizers are not playing a part commensurate with that which they would be called upon to do in case of emergency, nor are our resources of manganese in such shape, nor our

means of utilizing them sufficiently developed, to enable us to count upon them in case of need. No thorough and independent study has been made of all alloy combinations as regards their efficiency in deoxidizing steel and rendering it homogeneous and workable, either as a whole or part substitute for manganese, or if this has been done, it is not generally known and available.*

Before undertaking work of this nature—which would involve preparation of many alloy combinations, coöperation with steel and foundry men, and chemical and metallographical examination of the steels treated with these alloys for inclusions of gas, oxides, and slag—it has seemed best to submit the situation and the need of investigations on substitute alloys, in order that an opinion may be expressed as to whether the matter is of sufficient importance and promise to warrant serious consideration.

ACKNOWLEDGMENTS

In the preparation of this paper, reference has been made to Howe, Hall, Campbell, and Stoughton. The current numbers of the *Iron Trade Review* and *The Iron Age* have been consulted. Advice has been given by operating steel men, and also by Professors Crabtree and Macintosh of the Carnegie Institute of Technology, the latter of whom has carried on work in the preparation of ferromanganese-silicon alloys at the laboratories of the Carnegie Institute of Technology.

H. M. Boylston's "Investigation of the Relative Merits of Various Agents for the Deoxidation of steel," *Carnegie Scholarship Memoirs, Iron and Steel Institute*, vol. 7, p. 102 (1916) became available and known to the writer after this paper was submitted.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Dry-Hot versus Cold-Wet Blast-Furnace Gas Cleaning

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(New York Meeting, February, 1917)

Introduction

MARKED differences of opinion have been expressed by engineers interested in cleaning iron blast-furnace gases for use in hot-blast stoves and under boilers, in reference to the advantages of a hot-dry method over a cold-wet method. One point at issue involves the sensible heat energy in the moisture contained in the gas. Some advocates of the cold-wet methods claim that the condensation and resultant removal of the greater portion of this contained moisture by wet scrubbing, spraying or similar methods, results in a saving of some of this sensible heat energy, by reason of the fact that water vapor has a high capacity for sensible heat energy and may carry from the exit of a hot-blast stove, for example, more heat units than are sacrificed or lost when the gas is cleaned by this cold-wet method. Other advantages claimed for the latter method are: That gas burns more readily when it is free from moisture in any form; that because gas is made denser by cooling and removing the moisture it has a higher calorific value than hot gas carrying moisture; and that higher flame temperatures are obtained when the gas is cleaned by the cold-wet method.

A search through the available literature fails to disclose any extensive calculations or records of conditions obtained in practice, bearing upon these phases of gas cleaning.

This paper deals with these problems from the standpoint of the economy of using a gas of higher flame temperatures, of improved stove design, and of economy in the gas-cleaning department. Calculations have been made which show that by cleaning the gases by a cold-wet method the sensible heat energy of the blast-furnace gases is greatly reduced; and, in general, this loss of heat energy is far greater than that lost from the stoves or boilers due to the sensible heat capacity of the water vapor if carried away by the exit gases. By making use of the

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latest values given for the mean specific heat of gases, by Kuzell and Wigton,¹ it will be shown that for a typical iron blast-furnace gas the sensible heat energy which is lost from the gas by the cold-wet cleaning process is in all cases much greater than the gain resulting from the removal of moisture, even when the moisture content of the furnace top gases is as great as 100 gr. per cubic foot of the gas, calculated at 32° F., due consideration being given to the quantity of sensible heat energy which such an amount of moisture is capable of carrying from a hot-blast stove or boiler throughout the usual range of exit gas temperatures.

Returning to the subject of the effect of water vapor on flame temperatures, the paper shows that it would be true, as claimed by some iron blast-furnace operators, that the removal of moisture from the gases prior to combustion results in a higher flame temperature, provided no sensible heat energy were removed from the gas along with the moisture, due to cooling the gas. It is shown, however, that for typical kinds of iron blast-furnace gas, the theoretical flame temperatures are considerably higher when the cleaning method allows both the sensible heat energy and the moisture to be kept in the gas. The ideal cleaning process would be that of removing the moisture without reducing the temperature of the gases, but as no process of this kind is available it will be obvious that, other things being equal, it is better to clean these gases hot and dry and leave the moisture in the gases than to cool the gases for the purpose of cleaning and of removing water.

The gain resulting from the conservation of the sensible heat energy of the gas entering the stoves also permits of other changes in practice that appear on the whole to be advantageous. By having hot gases enter the hot-blast stoves, it is possible to obtain higher flame temperatures in the stoves. This will result in a saving of coke in preparing the furnace charge, since a hotter blast will be obtainable in the same length of time with the consumption of the same quantity of dry top gas from the blast furnace. This hotter blast not only supplies additional heat energy, but it makes possible the attainment of higher temperatures within the furnace, and this in turn makes possible a reduction in the amount of coke charged. This effect is best understood by a reference to the excellent article by Walther Mathesius entitled "High Blast Heats in Mesaba Practice."²

Furthermore, by employing higher temperatures it should be possible to store up the same quantity of heat energy in the checker work of the stoves in less time than is now required. By carefully cleaning the gases hot-dry and by thoroughly mixing the air for combustion with the blast-furnace gases a minimum amount of excess air is required. To further increase the flame temperature, it may be feasible in some cases

¹ *Trans.*, vol. 49, p. 774 (1914).

² *Trans.*, vol. 51, p. 794 (1915).

to utilize the heat in the stove exit gases for preheating the combustion air. It is obvious that the hotter the gas and the preheated air for combustion, the higher will be the flame temperature; and the same will also be true, the smaller the amount of excess air. The effect of water vapor, temperature of the gas, temperature of the combustion air, and the amount of excess air upon flame temperature is graphically shown by curves.

The curves have been plotted from a set of tables prepared from calculations based on the data on specific heats given by Kuzell and Wigton in the paper previously referred to. These tables can be applied to a large number of problems such as those in connection with hot gases coming from Portland cement kilns, smelter furnaces, cupolas, etc., where data on the sensible heat energy in the gases are required. It is possible that these tables may be submitted to the Institute for publication in another paper.

There appears to be so much room for improvement in the design and structure of hot-blast stoves that the subject has been discussed in a separate paper.

The total heat energy in a gas includes the heat which may be developed by combustion (chemical energy), the heat due to the temperature of the gas (sensible heat), and the latent heat of vaporization. The first type of energy may be called "the heat energy of combustion" and the second type "the sensible heat energy" of the gas. The heat energy of combustion is a function of the composition of the gases. For a definite composition of gases it is practically constant and can be readily calculated. The sensible heat energy of a gas depends upon the quantity of gas, the volume and temperature of the gas and the mean specific heat of the gas.

The specific heat of a gas in turn depends upon the temperature of the gas and its chemical composition.

In the present discussion 1 lb. of a typical dry, clean top blast-furnace gas is taken as the unit, and it is assumed to have the following percentage composition by weight:

	Per Cent.
CO ₂	21.00
CO.....	24.00
H ₂	0.25
CH ₄	0.25
N ₂	54.50

The presence of moisture, of dust and of excess air is measured in terms of the quantity of this foreign material per pound of such dry, clean top gas. The sensible heat energy of moist, top gas is, therefore, according to our method of calculation, the sensible heat energy of 1 lb. of dry top gas plus the heat energy of the moisture which accompanies and is in addition to 1 lb. of the gases which constitute the dry top gas; *i.e.*, the heat energies are added together.

In practice, the amount of moisture or dust in a gas is frequently measured in grains per cubic foot of gas. This method of measuring the moisture or dust content of a gas usually assumes that the moisture or dust is computed at a certain temperature of the gas such as 32° F. Thus take 1 lb. of dry top gas with 20 gr. of dust per cubic foot of gas calculated at 32° F., and 50 gr. of moisture per cubic foot of gas calculated at 32° F. This unit of gas would contain 0.0349 lb. of dust and 0.0874 lb. of moisture. The unit of gas considered would consist of a total of 1.1224 lb. of matter, and a calculation of the sensible heat energy of 1 lb. of dry, clean, top gas with the above moisture and dust content would involve the calculation of the sensible heat energy of 1.1224 lb. of matter. The present paper, however, compares two methods of cleaning, the dry-hot and the cold-wet methods. Since both methods presuppose the removal of dust, it is not necessary to consider the sensible heat energy of the dust. In the above example the sensible heat energy would be calculated for 1.0874 lb. of matter, the dust being excluded.

In the above example the amount of dust and moisture per actual cubic foot of gas decreases with rise in temperature, for the reason that the gas expands. At atmospheric pressure 1 cu. ft. of gas at 32° F. will become 2 cu. ft. at 523° F. and in the example of moist, dusty gas given above, the dust content of 20 gr. per cubic foot of gas calculated at 32° F., will fall to 10 gr. per actual cubic foot of gas at 523° F., while the moisture content will fall from 50 gr. per cubic foot, calculated at 32° F., to 25 gr. per actual cubic foot at 523° F., though the percentage of dust and moisture per pound of dry, clean, top gas has remained the same. The measurement of the density of dust and moisture in a gas is, therefore, made by calculating how many grains of each 1 cu. ft. of gas would contain if reduced in temperature to 32° F., the pressure being standard at 760 mm. of mercury.

For purposes of combustion it is necessary to add a certain minimum weight of air per pound of the clean, dry top gas. After combustion the chemical composition and the specific heats of the gases have been completely changed. The datum point will be taken as 60° F., and the sensible heat energy of the exit stove gases will be the amount of heat energy that the products of combustion of 1 lb. of dry top gas plus the specified excess air plus the specified moisture entering the stove with the gas and air would emit when cooled from the specified temperature of the exit gases down to 60° F. In these calculations it will be assumed that the moisture content of the stove exit gases is not great enough at any time to result in condensation of any of the water vapor at the temperature at which the mixed gases actually leave the stoves. In practice, the moisture content would seldom if ever reach such an amount.

The latent heat of the water vapor need not be considered because it is lost in any cleaning method which can be adopted. In the cold-wet

method the latent heat of the water vapor is absorbed by the water used during the washing process and is thus carried away by it; in the dry-hot method the latent heat of the water vapor is carried out with the stove exit gases and thereby lost.

Further, in using 60° F. as the datum point for calculations of the sensible heat energy of the exit stove gases or of the blast-furnace top gas after coming from the cleaner, it will be assumed that the gas contains 5 gr. of moisture per cubic foot of gas calculated at 32° F. This means that the sensible heat energy is referred to that of the same gas practically saturated with moisture at 60° F. Such a datum point is convenient because any kind of a gas cleaned by the cold-wet method usually comes out at about 60° F., and is practically saturated with moisture. As our comparison is made between a dry-hot and a cold-wet method of cleaning it is natural to assume conditions prevalent in the cold-wet method as the datum point.

Consider 1 lb. of dry, clean, top gas at 700° F., containing 20 gr. of dust and 25 gr. of moisture, both calculated per cubic foot of gas at 32° F. Our unit of dusty and moist gas weighs 1.0787 lb. It is found that the sensible heat energy in the 1 lb. of top gas and that in the moisture (which would be lost if the gas passed through a cold-wet cleaning apparatus, thus reducing its temperature to 60° F.) would be 174.69 B.t.u. = Q_t . By the cold-wet method of cleaning that cools the gas to 60° F., the above Q_t units of heat energy are lost for every pound of dry top gas, plus a specified moisture density.

In the dry-hot method of cleaning, no material lowering of temperature of the combustible gases for the stoves need take place. This condition is practically feasible when the electrical method of cleaning is used because the electrical precipitators need not be more than 15 or 20 ft. in length. The length of the gas mains and connections need not, therefore, be greatly increased, and, furthermore, they may be insulated so as to conserve the heat energy of the gases.

In the dry-hot method of cleaning the 25 gr. of moisture per cubic foot of gas, mentioned above, remain in the gas, and are carried into the stoves and then out with the products of combustion. The only difference between the exit gases from dry-hot cleaning and cold-wet cleaning in the present example is that in the one case there are 20 gr. of moisture per cubic foot of gas standard more than in the other case. Let us assume that the exit gases leave the hot-blast stoves at 600° F., which is a fair average. With the hot-dry method of cleaning these gases will carry away 8.83 B.t.u. of sensible heat energy for every pound of dry top gas over and above what the same gases would have carried out had they been cleaned by a cold-wet method, due to the greater amount of moisture left in the gas when cleaned by the hot-dry method. Let this energy be Q_e . The saving in sensible heat energy by the hot-dry method of cleaning as compared to the cold-wet method is:

$$Q_t - Q_e = 174.69 - 8.83 = 165.86 \text{ B.t.u. per pound of dry top gas.}$$

In the above comparison any energy changes due to expansion or contraction of the gases can be neglected because the exit gases are under practically the same condition of pressure and temperature for both methods of cleaning.

Assuming a ton of iron to represent a production of 12,000 lb. of such typical top gas, a hot-dry method of cleaning the gases would conserve: $12,000(Q_t - Q_e) = 1,990,320 \text{ B.t.u. per ton of iron in the above example.}$

Data and Theory of the Conservation of the Sensible Heat Energy of Gases

In the present article the term "grains of dust per cubic foot of gas calculated at 32° F., or 0° C." will be spoken of as "grains of dust" or as "grains of dust per cubic foot." "Grains of moisture" will mean grains of moisture per cubic foot of gas when the temperature of the gas is calculated at 32° F., and the pressure is 760 mm. of mercury. As previously stated, the moisture content is separate and in addition to the typical gas.

C_p is the specific heat of a gas at constant pressure and represents the ratio of the amount of the heat energy required to raise the temperature of a unit mass of the gas one degree compared with that required to raise the temperature of the same mass of water one degree (from the temperature of maximum density).

C_m is the mean value of C_p between the temperature t_2 and t_1 .

The equations used for calculating the sensible heat energy of various gases are obtained from the following equations, where t is the temperature Fahrenheit. These equations were used by Kuzell and Wigton. For references as to the experimental determination of the constants in these equations the reader is referred to that part of the bibliography pertaining to specific heats.

Mean Specific Heats of Gases under Constant Pressure

(To 2,000° C. or 3,600° F.)

	$C_m, (0 \text{ to } t) \text{ for}$ 1 lb. in B.t.u.
N_2	$0.2411 + 0.0000099t$
O_2	$0.2111 + 0.0000087t$
H_2O	$0.4701 - 0.0000118t + 0.0000000127t^2$
Air.....	$0.2342 + 0.0000096t$
CO	$0.2413 + 0.0000099t$
H_2	$3.3381 + 0.000124t$
CO_2	$0.1975 + 0.0000388t - 0.0000000043t^2$
CH_4	$0.5904 + 0.0000411t$

The term "flame temperature" in this paper will refer to the temperature that a gas will attain when one assumes that the energy of combustion is added to the sensible heat energy of the gas; all the resultant heat energy being used to raise the temperature of the gas, calculated on the basis of the values of the mean specific heat as given by the above

equations. The actual flame temperature will be lower because some of this heat energy is conducted or radiated away before complete combustion takes place. The pressure of the gases is considered to be constant and standard. Slight changes in pressure would make only very small differences in the calculations and conclusions.

Some Conclusions Derived from Sensible Heat Data

1. The mean specific heat at constant pressure increases for all gases so far as known, as the temperature is increased. In some instances, values of the specific heats at constant pressure are assumed for temperatures as high as 3,600° F. Undoubtedly, the measurements of specific heats at these temperatures are not as yet very exact, but it is probable that the results are of the right order of magnitude and therefore do not invalidate any conclusions drawn from them which are essentially qualitative. The conclusions are also applicable to gases of quite a range of composition, the results being worked out in a similar manner to that used here in the typical example.

2. In general, the sensible heat energy lost in cooling iron blast-furnace gas by a cold-wet method of cleaning is much greater than that carried out with products of combustion by the moisture left in the gases in a hot-dry method of cleaning. Consider the curves in Fig. 1. In *E* and *E'* the typical blast-furnace top gas has a moisture content of 100 gr. From curve *E* it will be seen that if the top gas enters the wet cleaner at 300° F., it will lose 78 B.t.u. per pound typical top gas when cooled to 60° F. From curve *E'* it will be seen that if the same gas is burned without reduction of moisture content and the product of combustion leaves the stove at 1,000° F., the accompanying moisture will carry away 74 B.t.u. per pound of typical top gas. Even under these very exceptional conditions, a hot-dry method of cleaning would result in a saving of 4 B.t.u. per pound of typical top gas.

Consider the other extreme condition. Assume the blast-furnace gas to enter a hot-blast stove at 900° F. From curve *E* it will be seen that on cold-wet cleaning 284 B.t.u. would be lost per pound of typical top gas. Also assume that the gas leaves the stove at 1,000° F. Then from curve *E'* it will be seen that the moisture accompanying 1 lb. of typical top gas will carry away from the stove 74 B.t.u. In this case there will be a saving of 210 B.t.u. per pound of typical top gas. The other curves show results more favorable to the hot-dry method of cleaning, since the moisture contents are less. In practice the average moisture content seldom exceeds 35 gr. per standard cubic foot of typical top gas, even when the stock in the furnace has been watered to reduce dust losses.

3. The effect of moisture on flame temperatures is given in the curves in Fig. 2.* The effect of increasing the moisture content from 10 gr. to

* Based on calculations from the formulæ and curves of Kuzell and Wigton, *Trans.*, vol. 49 (1914).

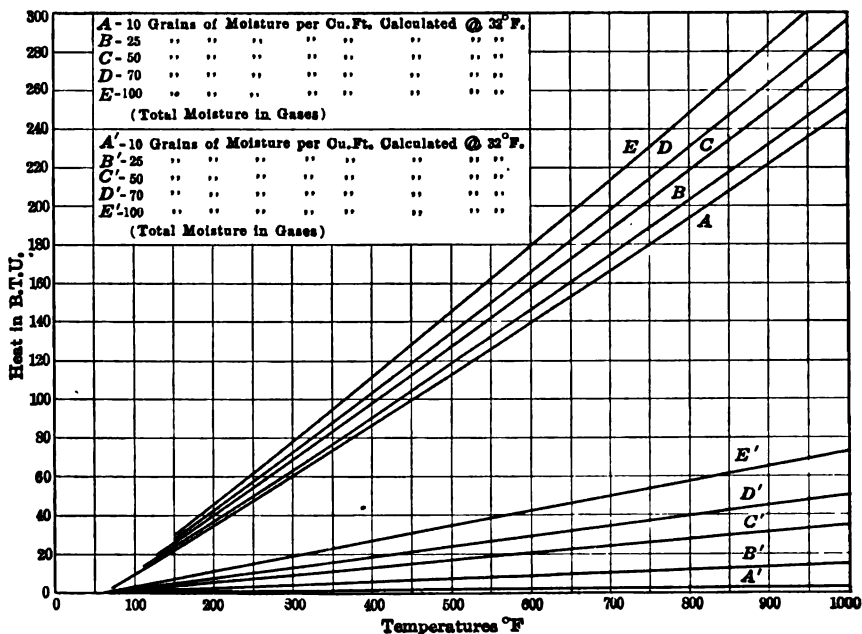


FIG. 1.—SENSIBLE HEAT CARRIED OUT BY MOISTURE IN STOVE EXIT GASES.

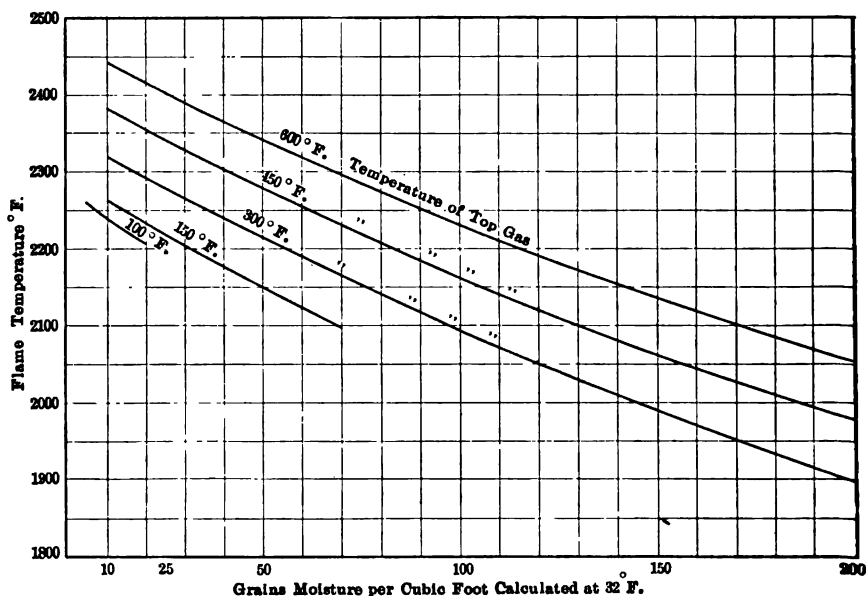


FIG. 2.—EFFECT OF MOISTURE IN FURNACE TOP GASES ON FLAME TEMPERATURE.

100 gr. would be to lower the flame temperature produced by burning the typical gas from 2,320° F., to 2,090° F., for the gas entering the burner at 300° F., and from 2,460° F. to 2,230° F., for the gas entering the burner at 600° F. Now suppose that these gases had been cooled to 60° F. and the moisture thus reduced to 5 gr. The flame temperature would then have been about 2,250° F. In other words, a cold-wet method of cleaning would have resulted in slightly higher flame temperatures in the above cases provided the moisture content was 100 gr. or greater. For a 50-gr. moisture content the cold-wet method would give a higher flame temperature if the top gases were originally at 300° F., whereas if they were at 450° F., or 600° F., or over, the hot-dry method of cleaning would

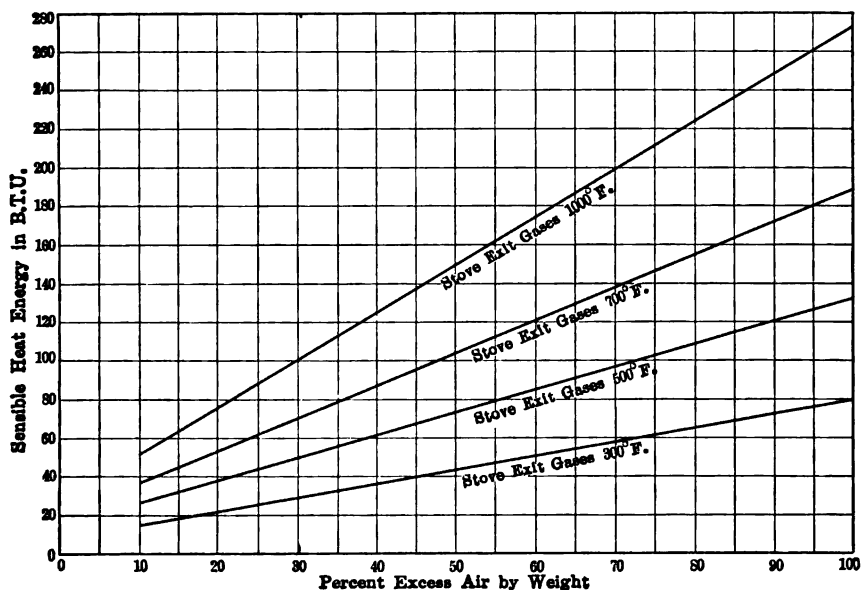


FIG. 3.—EFFECT OF EXCESS AIR ON AMOUNT OF SENSIBLE HEAT ENERGY ABOVE 0° F. CARRIED AWAY WITH STOVE EXIT GASES.

give the higher flame temperature. From the point of view of flame temperatures, there are certain temperature and moisture-content limits, on one side of which a hot-dry method of cleaning gives the higher flame temperature, while on the other side the cold-wet method of cleaning gives the higher flame temperatures. Operating conditions quite usually lie in the hot-dry region, as can be seen by a study of the curves.

4. The effect of excess air in increasing the amount of sensible heat energy carried out from the stoves and boilers by the exit gases at different temperatures is given in the curves of Fig. 3. These curves assume a moisture content for the top gas of 35 gr. (which is probably a fair average). The relative loss of heat energy due to excess air increases as the temperature of the exit gases is increased. The curves show how

very important it is to keep the excess air down to a minimum. Thorough mixing of air and gas, and having the mixed gases hot on entering the stoves and boilers, are important essentials, and some automatic device for regulating the air input for variations (due to furnace changes) in the quantity and composition of the blast-furnace top gases should be used.

5. The sensible heat energy in the constituents of the stove exit gases resulting from the burning of 1 lb. of dry top gas, carrying 35 gr. of moisture with 60 per cent. excess air, the air carrying 5 gr. of moisture, is given in the curves of Fig. 4 for various temperatures of the exit gases.

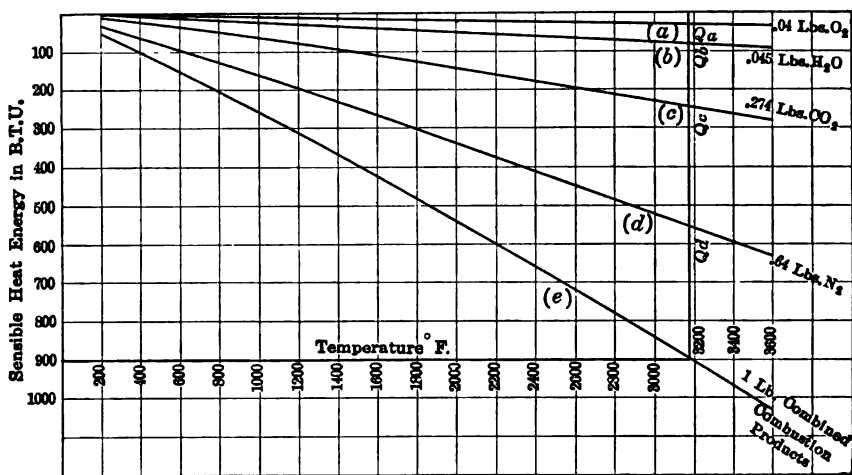


FIG. 4.—SENSIBLE HEAT ENERGY ABOVE 0° F. IN THE DIFFERENT PRODUCTS OF COMBUSTION FROM BURNING 1 LB. OF TYPICAL TOP GAS.

Although not plotted in that way, these curves could be made to show the sensible heat energy carried from the stoves and boilers by the moisture present in the original top gas and by the excess air. Such curves would give at various temperatures the loss of heat energy due to moisture and to excess air.

6. The curves of Fig. 5 show the effect of excess air on flame temperatures. A range of excess air from 20 to 60 per cent. lowers the flame temperature more than 300° F. As the amount of excess air is increased, the flame temperature depends less and less upon the temperature of the blast-furnace top gas. The great importance of reducing the amount of excess air, both as regards the conservation of sensible heat energy and the maintenance of a high flame temperature, is shown very positively by these curves.

7. To sum up, the statement frequently made, that the cold-wet method of cleaning gases is advisable because moisture is removed from the top gases, thereby permitting higher flame temperatures and obtain-

ing a decrease in the loss of sensible heat energy, is not true under the conditions of operation assumed in the paper. It is shown herein that these conditions are fairly typical and representative. Unless the gas has a moisture content exceeding 100 gr. per cubic foot, or its temperature is extremely low, sensible heat energy is conserved by using a hot-dry method of cleaning. The conservation of this sensible heat energy by a hot-dry method of cleaning also permits of a higher flame temperature, if the moisture content is not too high.³ The hot-dry method also makes it possible to operate with a minimum amount of excess air for combustion, this in turn promoting higher flame temperatures and conservation of gas.

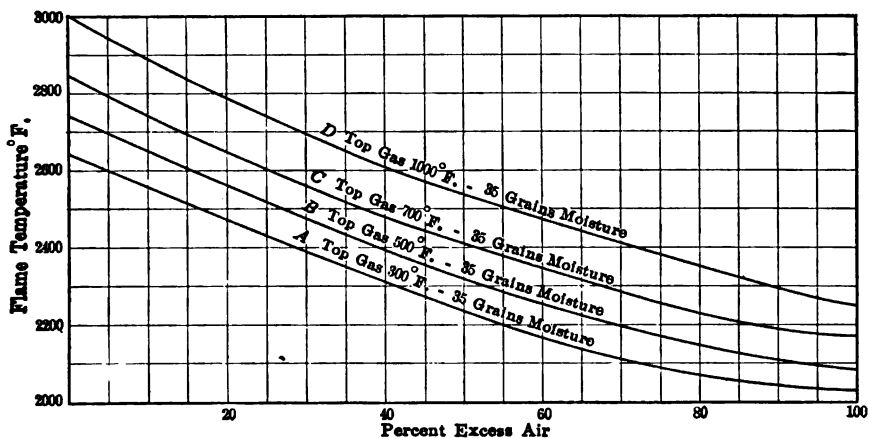


FIG. 5.—EFFECT OF EXCESS AIR ON FLAME TEMPERATURES

In some ores a considerable amount of compounds of potash, zinc, lead, arsenic, antimony, etc., may accompany the compounds of iron, copper, etc., for which the ore is being smelted. Under present conditions this more volatile part of the ore may be carried away in the top gas. A dry method of cleaning may allow the recovery of certain of these volatile compounds, thus making commercially possible the treatment of a greater variety of ores. At the present time the application of the electrical method of cleaning blast-furnace gas from iron ore containing zinc is being developed and other problems similar to this are also under consideration.

³ Dissociation of water vapor into hydrogen and oxygen is considered to be negligible under the above conditions. This would practically be the case if there was dissociation because under proper conditions the hydrogen and oxygen would again recombine. See Mallard and LeChatelier, *Annales des Mines*, 1883, vol. 4, p. 379; Hofman, *General Metallurgy*, p. 106. New York, McGraw-Hill Book Co., 1913.

Hot-Dry Cleaning by Electrical Precipitation

Under practical operating conditions, many other conditions relating to the cleaning process must be considered. A few of these may be briefly considered. The cold-wet process of cleaning is usually more or less automatic in operation and requires comparatively little attention. Rather large quantities of water are used and considerable power is consumed in handling this water, and in forcing the gases through the system. The washers are comparatively large and the initial expense of installation is large. In some instances, a problem arises as to the disposal of the muddy water, it often being illegal to allow this contaminated water to run into the streams, while in other instances water is not abundant and its use for gas cleaning may be prohibitive.

There is a hot-dry method of cleaning that promises to be very advantageous for the purpose of cleaning these gases. The electrical precipitation processes make use of a high-tension electrical discharge which sweeps out the suspended matter from the surrounding gas. The electrical method does not cool the gas, it precipitates the suspended dust in a dry state, thus making it easy to reclaim this material. The operation is practically automatic and the energy consumption is small.

The cleaning power of the electrical method is practically complete, in many instances being as high as 99 to 100 per cent.

The degree of cleaning is greater than that usually obtained by wet methods in general and is ample for stoves and boilers. Indeed the results of recent tests have indicated the probability that even with a single pass precipitator the gases would be cleaned to the degree required by internal-combustion engines.

Summary

1. In general the sensible heat energy of the top gas from iron blast furnaces above 60° F., is much greater than the sensible heat energy of the moisture carried away from the stoves. A hot-dry method of cleaning the gases that conserves the sensible heat energy of the top gases is therefore more efficient as an energy saver than a wet method of cleaning that reduces the moisture content to 5 gr. (per cubic foot calculated at 32° F.), thus decreasing the sensible heat energy carried out by the moisture leaving the stoves.

2. By conserving the sensible heat energy of the top gas a dry-hot method of cleaning the gas (having a moisture content of not more than about 60 gr. of moisture) generally permits of a higher flame temperature than would be obtained by a wet process of cleaning that reduces the moisture content to 5 gr., excess air being considered constant.

3. Excess air lowers the flame temperatures. Curves are given that show this effect for various amounts of excess air.

4. A dry-hot method of cleaning permits of the immediate recovery of the precipitated fume, dust or smoke in a readily usable condition.

5. Curves are given showing the flame temperatures with the typical top gases entering the furnace at 300°, 450° and 600° F., the gases containing variable quantities of moisture ranging from 10 to 200 gr. per cubic foot. Taking the top gas with a moisture content of 100 gr. and entering the stove at 600° F., the curve shows a lowering of the flame temperature of 220° F., from about 2,480° F. The wet method of cleaning would have cooled the entering gases from 600° F., to 60° F., thereby resulting in even a lower flame temperature.

6. Assuming a furnace to give 6,000,000 lb. of gas per 24 hr., the saving by the dry-hot method of cleaning of 140 B.t.u. per pound of gas (a representative case in practice) compared with the wet method would result in a saving of \$69.67 per day, assuming coal to cost \$2.50 net per ton and to contain 15,000 B.t.u. per pound. This equals about \$25,000 per year.

7. The electrical method is unique in that it is the only known method that operates very efficiently for removing small suspended particles. For this reason the electrical method should be especially well adapted for Mesaba and similar ores, where the dust in the top gas is very fine.

8. The electrical method of cleaning the top gas should prevent the troubles of slow ignition and imperfect combustion often encountered when wet cleaning is used. These ignition troubles are due primarily to the loss of sensible heat energy.

9. The use of clean gas permits of a much more efficient type of stove construction since it eliminates cleaning. For a similar reason, boilers will operate more satisfactorily.

10. The dry method of cleaning furnace gases permits of the collection of such volatile constituents of the ore as potash, zinc, lead, tin, mercury, arsenic and antimony compounds. Ores that are lean in iron or copper, for instance, might contain enough zinc, for example, to make their smelting profitable, as the zinc might be recovered in a highly concentrated condition.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Some Suggestions Regarding Construction of Hot-Blast Stoves

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(New York Meeting, February, 1917)

A HOT-DRY method of cleaning the gas from blast furnaces has been shown¹ to conserve the sensible heat energy of the gas, and in general it thus permits of a higher flame temperature. The electrical method of cleaning gases removes from 98 to 99 per cent. of the suspended matter in one operation when operating properly, thus making possible the use of a very clean gas for burning in the stoves. The use of such clean gas thus makes feasible a different type of hot-blast stove construction from that being used at present. It therefore seems advisable to construct the stoves from the viewpoint of getting the maximum amount of heat energy into the products of combustion.

By preheating the air added to the top gas and obtaining a better mixture of the air and top gas, less excess air will be required and a higher flame temperature obtained from the combustion of the top gas.

The interior of the stove should be provided with a kind of brick that will withstand a high flame temperature, will conduct the heat energy along certain directions, insulate the flow of heat along other directions and at the same time store heat energy so as to make the flow of such energy more uniform.

It is quite likely that these results can be best obtained by using several kinds of bricks: one type which is a good insulator of heat energy to be used for lining the stove and for making partitions through the stove to restrain the flow of heat; another type of brick or material with high refractory qualities; and a third type which will be a good conductor of heat energy.

The surface of the stove exposed to the atmosphere should be a minimum in comparison to the mass of checkerwork which it is possible to obtain, due regard being paid to other conditions.

With clean and hot furnace top gas it will be advisable to consider a rearrangement of the checker work in the stoves. One reason for the present size and shape of checker openings is to provide facility for

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¹ Hot-dry vs. Cold-wet Blast-furnace Gas, *This Bulletin*.

periodical removal of coal dust. With clean gas the present wide spacing would not be required.

The mass of checker bricks should be as great as possible and still expose sufficient surface to the passing hot gases to absorb the greatest amount of heat, leaving the exit gases as cool as possible.

The checker openings should be small so as to bring the brick surfaces into intimate contact with the hot gases. The old rule that checkerwork openings must be 9 in. (due primarily to the necessity of cleaning the stoves of dust) can now be changed, and in fact has been changed in several plants.

The hot gases should be uniformly distributed across the cross-section of the passage and so maintained during their travel through the stove.

The hot gases should pass through the checker openings at a fairly high speed, thus increasing the heat transfer by the scouring action against the faces of the bricks.

The bricks should be so shaped and spaced as to insure the results mentioned above. It might even pay to have the bricks so close together as to require the installation of a fan above the stove to compensate for and overcome the additional friction of the gases through the smaller openings. This greater resistance to gas flow in the stoves could be taken advantage of in insuring the best distribution, high gas speed adjacent to brick surfaces and maximum heat absorption.

The bricks should contain the greatest mass or weight of material per unit of volume, combined with the highest mean specific heat and heat conductivity; i.e., should be capable of holding the maximum heat energy consistent with speed of absorbing, storing and delivering such heat energy, and still possess the other qualities required of bricks for this purpose. Some experiments should be conducted and calculations made to determine the proper size, shape and composition of checker bricks, in the light of the points brought out in this paper.

Through improvements in design and the use of a hot-dry method of cleaning the top gas, it may be possible materially to lower the installation and the operating costs of the stoves, and obtain results equal to those obtained in present practice.

Mr. Mathesius'² paper has shown several advantages resulting from increasing the temperature of the blast. It would seem that effort along the lines indicated above should be made to increase the temperature of the blast by obtaining hotter stoves through the use of higher flame temperatures. Success by this method should show a further marked reduction in coke consumption in the furnace, an increased output of iron, smoother running conditions, and an increase in the calorific value of the gas from the furnace.

² High Blast Heats in Mesaba Practice, *Trans.*, vol. 51, p. 794, (1915).

As for preheating the combustion air for the stoves, this is somewhat analogous to preheating the blast for the furnace, but it is unnecessary to employ so expensive a method. The combustion air could be preheated by the stove exit gases by means of suitably designed heat-exchanging apparatus placed above the stove, the preheated air being delivered to the stove burners by means of fan and duct. Calculations readily show the importance of adopting this suggestion and should be proven in practice.

It should be noted that a higher efficiency of absorption can be obtained in the stove when the flame temperature is highest; the exit gases will be only slightly, if any, higher in the one case than in the other. Furthermore, perhaps the greatest advantage to be obtained by preheating the combustion air and by hot-dry cleaning will be the ability to get a hotter blast in the same length of time, thus improving the ratio of "time on wind" to "time on gas."

A burner for intimately mixing the gas and air before burning (a modified Bunsen type) will aid greatly in giving a high flame temperature and the gas will be burned with a much less excess of air. Forced combustion, consisting of properly blowing the air into a gas burner with sufficient pressure, insures quicker combustion and a larger stove capacity. Radiation losses might thus be reduced, since a smaller number of stoves would be required, provided the efficiency of heat absorption by the checkerwork is not impaired. The question of long flames versus flameless combustion should be answered by thorough practical tests.

Application of Heat Data to Boilers

For use in boilers the importance of thorough cleaning may in some cases be more important even than it is in the case of the stoves. In using raw gas it is customary to blow the dust off the tubes every day by means of compressed air or steam. During blowing, a large amount of excess air enters the combustion chamber. Immediately after cleaning, the dust begins to deposit again and the efficiency of the boiler immediately decreases. The cleaner the gas, the more valuable it is for boiler purposes. The hot-dry method of cleaning the top gas for burning in boilers is therefore more efficient than the wet method for the following reasons: (1) the top gas is made cleaner than when the wet method of cleaning is used; (2) the sensible heat energy of the top gas is conserved; (3) the temperature of the gases in the boiler is increased, making possible the use of a gas of lower calorific value; (4) no slags or corrosion results from the dust settling on the furnace bricks or the boiler tubes.

The curves described in our other paper in this *Bulletin* can be applied directly to calculations on the saving of heat energy in the boilers. In boiler operation there is the added advantage that the gas carried from the boilers is usually at a comparatively low temperature. For this

reason the sensible heat energy of the unremoved moisture, carried away in the exit gases from the boiler, is even smaller than that from stoves.

The data on sensible heat energy given in the charts can be applied to a large number of problems: The hot gases coming from Portland cement kilns, smelter furnaces, cupolas, furnaces used to heat retorts for making illuminating, water and producer gas, coke ovens, etc. The electrical method of cleaning is now used in some of the above cases for removing the dust and fumes at high temperatures. At the Riverside cement plant and in sulphuric acid plants the electrical method is now being used to clean gases at about 900° F. Several other gases have been cleaned at this temperature. A cheap and efficient method of cleaning gases at a high temperature will thus introduce a new epoch in the saving of the sensible heat energy that is now being wasted in so many instances.

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The Seasoning of Castings

BY RICHARD MOLDENKE,* E. M., PH.D., WATCHUNG, N. J.

(New York Meeting, February, 1917)

ONE of the little-known characteristics of cast iron, which nevertheless has an important bearing on results where accuracy in machining is essential, is the ability of this material to ease up internal strains when allowed to remain quiescent for a more or less extended period of time. It seems as if the molecules in such a casting, by virtue of their "mobility," can adjust their relative positions to an extent sufficient to overcome some of the existing stresses.

The following instance will perhaps give a fair idea of the condition a casting may be in when just shaken out of the sand. A very large sheave-wheel, after shaking out, was taken outdoors to be cleaned and made ready for turning up. It was leaned against the side of the building, but before much could be done an arm tore apart with a loud report. Investigation showed that the sun had been shining on the upper rim, thus adding a slight strain to those already existing within the arm and thus overbalancing the strength of the metal in tension. Had this sheave been kept under cover for a while, or at least until machined, the strains would have eased off sufficiently to allow the sun to look upon it without disaster.

It will not be necessary to multiply examples. Every engineer knows the danger of water-hammer in pipe lines, particularly if the latter are of cast iron. Every mechanic knows, or should know, that it is not good to strike a fitting that is under steam pressure.

It will be necessary to say a few words on the "internal strains" in castings—the so-called "casting strains" we hear so much about. We all know that to get a casting reasonably true to the dimensions wanted requires a slightly larger pattern. The usual allowance for gray iron is $\frac{1}{8}$ in. to the foot (1 cm. per meter) and $\frac{1}{4}$ in. to the foot (2 cm. per meter) for white iron (all dimensions). This reduction in length, breadth and thickness in a casting is erroneously called "shrinkage." It should be called "contraction," as for practical purposes it is simply the difference in dimensions of the casting red-hot and cold.

The real "shrinkage" covers an entirely different situation. When a

*Consulting Metallurgist.

casting is poorly designed—thick and thin parts adjoining without special need—and in pouring the mold it is impossible to feed properly, the thinner sections set more quickly than the thick ones and may leave the latter without means of drawing in liquid metal to compensate for the reduction in volume in the act of setting. As the metal sets against the mold walls first, and gradually thickens from the surface inward, when the influx of fresh supplies is stopped there results a void in the center, or at least a spongy portion. This is, "shrinkage," and can be seen more particularly in white iron, by reason of its greater reduction in volume from liquid to solid form, apart from the final contraction from red heat to ordinary temperatures. The favorite places for such "shrinkage" are at abrupt angles, in thick parts adjoining thin ones, in the rims of flywheels, hubs of pulleys, at the flanges of cylinders, etc. The correction of the trouble is not germane to this article.

It will be seen from the above that there are really two kinds of reduction in volume to be reckoned with: First, that due to the change from the liquid to the solid state; second, the reduction in volume after setting until ordinary temperatures have been reached. The first, often called "interior shrinkage," is a rather serious thing. The specific gravity of molten iron is about 6.65, and does not vary widely from this figure whether the metal on setting is gray or white in fracture—all the carbon being combined when in the molten state. On setting, however, if gray iron results the specific gravity will be over 6.8, and if white iron, up to 7.8—the formation of graphite in the structure accounting for the comparatively moderate increase in the case of gray iron. In an average cast iron, with 7.3 specific gravity, the increase in density is 0.65, or 9 per cent.—which means a very big decrease in volume for equal weights of molten and solid metal. This situation accounts for the quantities of molten metal that have to be added to a mold after pouring it full in the first place, and in the case of small castings—particularly when of white iron—for the funnel-shaped sprue left in the pouring basin or gate.

Contrast this with the eventual reduction in volume after setting. Here we have a linear reduction of about 1 per cent. in every direction, or say a 1-in. cube taken from a cube the sides of which are 100 in. each; an infinitesimal accomplishment as against the real metal shrinkage.

It stands to reason that if the metal in setting has the power to pull apart whatever liquid material may remain after feeding has stopped, and thus give large spongy parts in the interior of a casting, there must have been set up powerful strains which affect the strength injuriously. This is apart from the reduction of strength in the material for the section itself. In other words, not only will the metal have a smaller tensile strength because of the spongy nature of part of the section, but the interior strains counterbalance part of the tensile strength that is available.

This situation is intensified by the fact that the metal in setting does so far more quickly at the mold surface than in the interior—the cold sand walls drawing away the heat from the molten iron more quickly at the beginning of the setting process than later when this heat has to travel through a more or less thick shell of metal already set. The consequence is a higher percentage of combined carbon at the surface than in the interior of the casting. In the extreme case—that of chilling the surface—we have a white iron surface and a gray iron interior. The relative change in the specific gravities of the same molten iron turned into two extreme forms of iron as cast will indicate what strains there must be within the casting in question due to the differences in volume which the two metals want to occupy when set but cannot properly occupy on account of the quickness of the setting action.

Finally come the strains due to the contraction in the set material until ordinary temperatures have been reached. This has been stated as $\frac{1}{8}$ in. to the foot in gray iron (about 1 per cent.) and $\frac{1}{4}$ in. to the foot in white iron (about 2 per cent.). In large castings this is very serious. Suppose, in the case of a big flywheel, the rim sets fast enough to hold the much cooler arm as in one set of jaws of a testing machine, the hub—held by the arms on the other side of the wheel—being the other set of jaws. Surely the arm in wanting to reduce in length $\frac{1}{8}$ in. to the foot (1 cm. per meter) must be under a terrific strain if not allowed to do so. In the case of white iron the situation is much worse, such castings as hand-brake wheels (subsequently annealed for malleable castings) snapping apart when allowed to cool in the sand in the ordinary way. Such work must be shaken out as quickly as set, taken to special ovens and allowed to cool down very gradually.

Sufficient has been said to make the case of cast iron look very weak. Fortunately, there are two phenomena which help to overcome some of the injurious strains set up. The first is the fact that cast iron—particularly gray cast iron—in the act of setting (between liquid and solid) can be stretched. The second is the before-mentioned “seasoning” or easing up of the remaining strains after the final contraction—through the mobility of the molecules. It is the stretching of gray iron during the setting that saves the flywheel arm from rupture before the new strains due to final contraction are introduced. It is the inability of white iron to stretch very much which makes for so many cracked castings in the malleable process which would not be seriously affected by the final contraction.

The above discussion of the actual situation in making castings has another bearing. Purchasers of castings may wonder why foundrymen who really know something about their basic material are so uncompromisingly opposed to test coupons on their product. The arguments given above should convince the man who is at least somewhat familiar

with cast iron that it is unfair to the maker of the casting to judge its value by a test piece subject to a variety of strains introduced as the result of position, manner of attachment, method of pouring of the metal, etc., of the mold. It is further unfair to the purchaser—if he only knew it—to judge by coupons, as there are many ways of artificially strengthening such test pieces. There is only one way of testing a casting properly, and that is to break it. Obviously this will not do, and hence for repetition work a given percentage of castings can be thus tested. For all other cases the only method of obtaining a reasonable assurance on the subject is the making of standard test bars, entirely apart from the castings but of the same iron, and making these test bars under conditions that give the iron the best possible chance to show just what it is, neither artificially strengthened, nor filled with strains and thus deliberately weakened.

This brings us to the real object of this note. Every mechanic knows that in planing up a slab of cast iron on both sides, to get a true job it is necessary to take a light cut, reverse, and take a cut on the other side, then reverse again for the finishing cut, finally reversing for the last cut. If this is not done there will be warped surfaces to deal with on account of the internal strains. Again, it is well known that to get a true piston is a rather difficult matter. Even after grinding to a finish it is apt to get out of true. It is not so generally known, however, that if such a cast-iron plate or piston is allowed to remain in storage for a long period, the results will be much more satisfactory. The castings have "seasoned." Where establishments are familiar with the situation, orders for castings are placed far ahead of requirements. Since, however, on getting to the bottom of a big pile the difficulty of tracing defective work becomes correspondingly harder, only shops having their own foundries are likely to do much storing.

The present demand for very high-class machined castings, as evinced by automobile cylinders, pistons, engine and compressor cylinders, etc., should bring this question of "seasoning" out very prominently. Inquiry by the writer has shown but little knowledge on the subject in the trade generally, though where first-class foundrymen were connected with the industries involved, these were very much alive to the matter. In general, the difficulty seems to be the inability of storing up ahead, or when so doing of striking defective product when least expected, particularly in such cases as castings for piston rings, etc.

For what seems the best indirect explanation of the "seasoning" action, we are indebted to the well-known metallurgist, A. E. Outerbridge, Jr., who in his famous experiments on "tumbling" castings to increase their strength, found that by the action of light blows, often repeated, the internal strains were relieved to such an extent that the real value of the metal came into play. The "mobility" of the mole-

cules was aided by artificial means. Incidentally, however, the tests establish the "mobility" of the molecules in cast iron very satisfactorily. Replace half-an-hour's tumbling by 6 months' quiescence and the molecules will have done their work with somewhat the same results.

In view of the possible depression scheduled for this country on the close of hostilities in Europe, would it not be well to ease up operations slowly instead of shutting down tight? This would help the industrial situation adjust itself more safely and at the same time stock up supplies of castings which will be all the better for having seasoned. Inasmuch as similar improvement has been noticed in the case of ingots, billets and other iron-base products, the same argument would hold.

The writer presents this memorandum in the hope that a discussion may bring out present practice and possible applications with respect to an almost unknown but highly valuable characteristic of cast iron and allied materials.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

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Portable Miners' Lamps

BY EDWIN M. CHANCE,* WILKES-BARRE, PA.

(New York Meeting, February, 1917)

DURING the past 10 years, the safe and efficient lighting of the coal mines of this country has received an ever-increasing amount of attention. Several States have passed laws attempting to regulate the type of lamp to be used and the nature of the fuel to be burned, and the mining departments of coal-mining States have generally shown a keen and intelligent interest in this subject. The passage of the recent Employers' Liability Act in Pennsylvania has made it necessary for many coal-mining companies to take out liability insurance, and the companies underwriting such insurance have made it desirable for the insured to permit the use of none but illuminants of established worth. While these conditions have not obtained for a sufficient length of time to permit the statement that the illumination of coal mines by portable lamps has been standardized, still, considerable progress has been made and the direction of future practice in this branch of coal-mining technology is very evident. Under these conditions it seemed that a review of the methods now used in coal-mine illumination, together with a brief consideration of the principles underlying these methods, might be of some interest.

Miners' lamps may be divided into three classes: The open light, the electric cap lamp and the flame safety lamp. It will be desirable to consider each of these classes separately, as each has properties peculiar to itself and one class is hardly comparable with another.

In this country, without question, the open light is the most generally used of all miners' lamps. This fact is explained by the relative freedom of a large proportion of our mines from gaseous conditions, and by the admirable systems of ventilation installed in those that show a tendency toward such conditions. In the metal mines, the miner's candle has had and still has considerable vogue. Because of the small amount of ventilation usually supplied in metal mines, the freedom of the candle from any tendency to produce noxious gases or offensive odors, and the small amount of air it consumes, are valuable assets, while, because

* Consulting Chemist and Engineer.

of the light color of the rocks in which workings are driven, the meager light is not a serious handicap.

In coal mines, however, conditions are very different. Because of the considerable volume of air passed through the workings, the small air consumption of the candle is of little moment, and its feeble rays are so completely absorbed by the dusky background that its light is entirely inadequate. Moreover, the cost of the candle is relatively high. For these and other less obvious reasons the metal miner's candle is practically unknown to the coal miner.

Up to a few years ago the open oil lamp had no rival in non-gaseous mines and it is still very largely used, though its use is becoming limited, as will be shown later. The oil lamp has many disadvantages and the legislatures of a number of States have endeavored from time to time to remedy these defects by law. As the service given by an oil lamp varies largely according to the character of the oil, efforts have been made to fix by law the quality of the oil that may be sold to miners for use in these lamps. These efforts, unfortunately, have proven rather ineffectual, as the result has generally been to increase the cost of oil to the miner without increasing its quality in anything like the same proportion. These laws, therefore, may well be considered one of the most potent of the forces that have driven the oil lamp out of its once strong position.

An example of the manner in which a law, that was honestly intended to be beneficial to the miner, failed of its purpose is found in the Bituminous Mining Law of 1911 of the State of Pennsylvania. This law stipulated that oils sold for use in miners' lamps should not yield more than 0.11 per cent. of soot when burned in a miner's lamp under standard conditions. One of these conditions was that the flame of the lamp should be $1\frac{1}{2}$ in. long. Now, low-grade oils when burned under these conditions yield as much as 1 per cent. of soot, while high-grade oils will give as little as 0.03 per cent. Thus it would seem, at first glance, that this law would considerably better conditions in the mines.

Such is not the case, however. Oils to pass this test must be very largely composed of costly fatty oils and this so greatly increased the cost to the miner that he was obliged to look for some cheaper illuminant. Moreover, instead of a flame $1\frac{1}{2}$ in. long, the miner burns one of a maximum length because he wants as much light as he can get. The writer has found that while costly oils, containing high percentages of fatty ingredients, will produce much less soot than oils of medium price, and less fatty material, when burned under legal test conditions, these differences very largely disappear when these oils are burned under the conditions that obtain in the mines. With very long flames the high-priced oils still show a superiority to the medium grade, but the differential is so slight as to be of little real moment.

Indeed, the soot-forming propensities of both these oils under the

conditions of use are so great that it is idle to attempt to classify one as better than the other. They are both very bad. Thus with a legal requirement of 0.11 per cent. soot or lower, we find the oil passing this test will give about 8 per cent. of soot when burned as it would be in the mines—that is, with a flame 5 to 6 in. long—while the oil that will not pass the legal requirement, giving under test conditions, let us assume, 0.5 per cent. soot, will make under actual working conditions about 9 to 10 per cent. soot. Thus we may say that despite the greatly increased cost of the legal-test oil it is practically no better than many oils that may be secured at half or one-third the price. It is to be understood that many oils are of so low a grade as to be entirely unsuited for use in miners' lamps and, of course, these remarks do not apply to them. The point is that the tendency of many of the State laws is to increase the cost of oil very considerably to the miner and mine operator without proportionately improving its quality.

One of the drawbacks to the use of the open oil lamp is the greatly increased fire risk where such lamps are used in dry workings. It is necessary for the user of such a lamp to renew its wick or lamp cotton at frequent intervals, and it is customary to pull out the old cotton and insert the new while the old lies blazing on the ground. When the new cotton is in place and alight the miner places his heel on the blazing remains of the old, and perhaps extinguishes it; at any rate, he goes away and leaves it, to burn or not as may be. Another source of fire is the shower of sparks that is blown from the wick when the wearer of the open oil lamp is traveling against a strong ventilating current. Together these are the possible causes of mine fires that have disposed thoughtful mine operators to look with disfavor upon this source of illumination.

Of recent years a substitute for miner's oil, called "Miner's Wax" and a host of proprietary and brand names, has been placed upon the market. It is a paraffin wax obtained in the refining of petroleum and possesses the property of burning with a whiter flame than miner's oil and giving somewhat less soot. It must be used in a special lamp, however, as means must be provided for keeping it in a molten condition in the lamp fount. This is accomplished by conducting heat from the flame to the fount. Its use, though considerable, is decreasing because the fire hazard with this illuminant is as great as with miner's oil, and it is troublesome to handle.

Undoubtedly the greatest advance made in the illumination of non-gaseous mines is the acetylene or "carbide" miner's lamp. This lamp has come into general use during the past 7 years and is now probably the most widely used of miners' lights. The reasons for its popularity are not far to seek; in brief, it gives far more light than any other portable miner's lamp and costs less to operate. It gives a clear, white light in which objects have very much the same color value as in daylight. It

makes no smoke or soot and its demands on the oxygen of the mine air are moderate. It gives more reliable indications of the presence of dangerous proportions of black damp than the oil-fed flame. It gives off no sparks and hence decreases the fire hazard very considerably. It may thus be seen that this type of miners' lamp has benefits for the mine operator and the worker and is liked by both. The writer understands that insurance companies underwriting the insurance of many coal-mining companies under the new Pennsylvania Compensation Act have recognized the safety features of the acetylene miners' lamp by giving credits on the insurance rate where such lamps are used in non-gaseous mines.

Some years ago, considerable uneasiness was felt among mining men because it was thought that the acetylene lamp failed to give adequate warning of the presence of black damp. In the past, black damp had been believed, by many, to be an atmosphere in which a lamp would not burn, the reasoning being along these lines: If an oil lamp goes out, it is because there is not enough air (meaning oxygen). Now it is a fact that the acetylene lamp will burn where an oil lamp will not. Hence, if the oil lamp will not burn there is no air, and as the acetylene lamp continues to burn, this indicates that the acetylene lamp will burn without air. Therefore, a man may carry an acetylene lamp into an atmosphere containing so little air that he may be rendered unconscious, and still his lamp will give no indication of the dangerous condition of the atmosphere.

The facts of the matter are these: The oil-fed flame requires a minimum of about $17\frac{1}{2}$ per cent. of oxygen for its maintenance; the acetylene flame requires about $12\frac{1}{2}$ per cent.; and a man's life is endangered should the oxygen content fall much below 10 per cent. At about 14 per cent. of oxygen, however, the color of the acetylene flame changes markedly. It loses its brilliance and illuminating power, and becomes greatly elongated and unstable. From these data it will be seen that the miner is given obvious and adequate warning of the vitiation of the atmosphere through deficiency in oxygen. While this warning is not so peremptory as that given by the oil lamp, still it is of ample distinctness for men to appreciate and value, and above all, it is essentially a real danger warning.

On the other hand, the warning of the oil-fed flame is given with so high an oxygen content that miners have learned to disregard it, and will frequently go into workings containing air in which their oil lamps will not burn. They know that they can live in an atmosphere in which these lamps will not burn, but do not realize that, once in the dark, they have no further guide to the quality of the atmosphere and that in a few feet it may become lethal. Such is not the case with the acetylene lamp; its warnings are given so near to the danger point that men will have a wholesome respect for them, to the great increase of their own safety.

While the writer has heard of cases in which men after working with acetylene lamps in sections in which oil lamps would not burn became

sick when brought into fresh air, these always proved to be based upon a fallacy. Upon investigation, the fact has always developed that the disability of the men was due to other causes; too high a temperature of the workings or carbon monoxide being the most usual. In any case the presence of the disturbing agency would not have been detected by the use of oil lamps. Indeed, the writer has many times seen men fall like flies in a place they had considered safe, because their oil lamps gave no indication of anything abnormal in the composition of the atmosphere.

The supposed danger from the use of acetylene lamps where black damp may be encountered has caused their use in pillar work and in robbing to be discontinued to some extent, thus exchanging a danger of the imagination for a very real peril. Those familiar with the coal-mining industry know what a curse miner's asthma has been. If not caused, it is at least aggravated, by the greasy soot and oily emanations given off by the oil lamp. Now these foul vapors are at their worst in pillar work and in robbing where the ventilating current is apt to be at its lowest ebb and where black damp is obviously most likely to be encountered. Hence, as a result of trying to safeguard the miner against the hypothetical danger—of a combination of black damp, the acetylene lamp, and his own stupidity—we expose him to the very real and pressing danger of miner's asthma, against which the most careful and most thoughtful is helpless.

The deepening of our mines, and the increased length of airways, together with the greater attention now given to the danger of gas and dust explosions, have all tended to increase the number of safety lamps in use. Safety lamps may be divided into two classes: The electric cap lamp and the flame safety lamp.

The electric cap lamp is a development of the last 7 or 8 years and its growth has been very lucidly traced by J. T. Jennings in a paper read at the 1916 meeting of the Coal Mining Institute of America. It will, therefore, be unnecessary for the writer to go deeply into this matter. European practice has tended toward the development of an electric hand lantern. Because of the more general use of flame safety lamps there, their mine workers were satisfied with this type, being accustomed to the inconvenience of hand lamps.

In this country, however, conditions are radically different. The miners are, as a general rule, accustomed to open lights worn on the cap and rebel at the inconvenience and inefficiency accompanying the use of the electric hand lantern. Hence, when an electric miners' lamp was developed it became essential that it should be such that the efficiency and convenience of the cap lamp would be retained. Progress was made along these lines with the result that the present very convenient equipment has been developed.

It is remarkable that so complete a standardization in general design

as is now found among the product of the numerous manufacturers of this type of lamp should have been possible. The credit for this should be given without stint to the Bureau of Mines. This bureau has worked hard and faithfully with the manufacturers of electric cap lamps for the past 4 years or so, and has had a very definite vision of what such a lamp should be. The result of this pre-natal influence is a startling similarity in the various equipments offered to the mining industry. While this method has perhaps sacrificed a little individuality, it has undoubtedly increased the average excellence of the product, and made the whole industry more robust by weeding out the abnormalities.

Technical literature has been so full of descriptions of various types of miners' electric cap lamps that it will not be necessary to describe any in detail; it will suffice to touch upon the general advantages and disadvantages of this type of lamp.

Many of the underwriters of insurance under the Employers' Liability Act have shown a marked preference for the electric cap lamp when compared with the flame safety lamp. This preference has led to the penalization of companies using the flame safety lamp, to the extent of 11 c. for each \$100 of pay roll, whereas, were electric cap lamps installed, this penalty would be wholly removed. This premium has led to the installation of many electric cap lamps.

This lamp throws its light into the plane of vision of the wearer so that its light is efficiently utilized. It leaves the miner's hands free and the light requires no attention; indeed, the outfits are so arranged and locked that it is impossible for the wearer or any unauthorized person to tamper with them. There is no fire risk with these lamps and we are assured by the Bureau of Mines that the danger of their originating a gas explosion is practically nil. These are also the safest of all lamps with which to handle explosives. They have three chief disadvantages: Their upkeep is high, the flux of light they furnish is not so great as it should be, and their wearer is in absolute ignorance, so far as the lamp enters into the matter, of the nature of the atmosphere surrounding him. It is very probable that the next few years will see these faults abated to a considerable extent.

That the flame safety lamp should be so old, so widely used, and so little improved in all its years of service reflects but scant credit upon the human mind. To all intents and purposes we still have Sir Humphrey Davy's invention in actual use, development having practically stopped after introducing the use of a cylinder glass to surround the flame. In this country the bonneted Clanny and Wolf type are practically standard except where heavily pitching veins are encountered, as in the Schuylkill anthracite district. Here, because of its lightness and the convenience with which it is handled on the pitches, the old Newcastle Davy is still supreme.

The flame safety lamp is principally handicapped by the meager light it gives. Its use is absolutely essential as an indicator of the quality of the mine air in all gaseous mines, and the writer would suggest that even in non-gaseous mines, where electric cap lamps have superseded open lights, the installation of a number of flame safety lamps would be an added safeguard. The principal difficulty in the use of the flame safety lamp as an indicator of atmospheric conditions, where electric cap lamps are relied upon for illumination, is that unless the flame lamp is the source of light it will come to hang unnoticed on the miner's belt or be left neglected in a corner of his working place. In other words, unless his attention is automatically called to it from hour to hour, he will, in the long run, cease to note its warnings.

While the variations of the Davy principle on which flame safety lamps are constructed have been endless, these variations have been slight, and the principle has not been diverged from with any success. As a result, the flame safety lamps in actual use in this country are of two very similar types, the Clanny and the Wolf, as has been mentioned above, ignoring the Davy lamps used in the southern anthracite district of Pennsylvania, for these will soon disappear.

With the design of the lamp fixed, we have but one variable to consider and that is the fuel burned. Even this disappears in the case of the Wolf lamps, as these will only function with naphtha whose composition may vary within but narrow limits. With the Clanny type lamp, however, a wide variation in the nature of the fuels is possible. Among these are sperm, peanut, lard, rape, seal, cotton seed and mixtures of these with mineral burning oils. The writer has spent much time in investigating the question of improving the quality of safety-lamp oils and has met with some success in this direction. It has been found that some of the most costly and highly prized safety-lamp oils were really inferior to mixtures containing high proportions of high-grade mineral burning oils. These mixtures burn with a whiter flame, give appreciably more light, do not crust the wick, and are much cheaper than the standard safety-lamp oils.

There is another type of lamp that holds out promise for the future. This is the acetylene safety lamp invented by T. M. Chance of Philadelphia and described in the discussion of R. P. Burrows' paper on coalmine illumination.¹ As this lamp is not yet a commercial fact, but little can be said of it definitely. It would seem, however, that it combines the safety and indispensable gas-detecting properties of the flame safety lamp with many times the illuminating power of the electric cap lamp.

A table is appended containing data that may make more intelligible some of the statements in this paper. These data have been accumulated

¹ *Trans.*, vol. 54, p. 34 (1916).

during the past eight years and are general averages. The photometric determinations were made upon a United Gas Improvement Co. 60-in. bar photometer. The photometric standards used were 10-volt tungsten lamps, prepared and calibrated by the National Lamp Works, and standard sperm candles. At times a secondary standard was used, consisting of a long-time kerosene burner, similar to that used in railway signal practice, standardized against one of the primary standards noted above.

It will be noted that no estimate of the cost per day of electric cap or flame safety lamps is given in the table. In the writer's opinion, the modern electric cap lamp has not been in use long enough for an intelligent opinion of its upkeep cost to be formed. Moreover, the labor charge on both the electric cap and flame safety lamps is so large and varies so much with the size of the installation that such figures as could be given would have but little meaning.

Conclusions

1. The open oil lamp has outlived its general usefulness. It still has a field, however, in special cases, such as those of drivers, motormen, trip runners and the like, who are obliged to work in swift air currents.

2. The use of the open acetylene lamp is growing in all non-gaseous mines because of its cheapness, the powerful light it gives, its reliability in the presence of black damp, and its freedom from soot and noxious vapors.

3. The electric cap lamp is best adapted to use in gaseous mines. Its flux of light is superior to that of the flame safety lamps now in use. Moreover, as it leaves the hands free it is more convenient than the flame safety lamp. Under especially drastic conditions in non-gaseous mines, where the fire risk is unreasonably high due to peculiar local conditions, its freedom from fire hazard recommends its use. In all gaseous mines its use must be accompanied by that of flame safety lamps, in order that the condition of the atmosphere may at all times be known in all parts of the mine. Where it entirely replaces open lights in non-gaseous mines it is imperative that a few flame safety lamps be supplied along with the electric cap lamps for the same purpose.

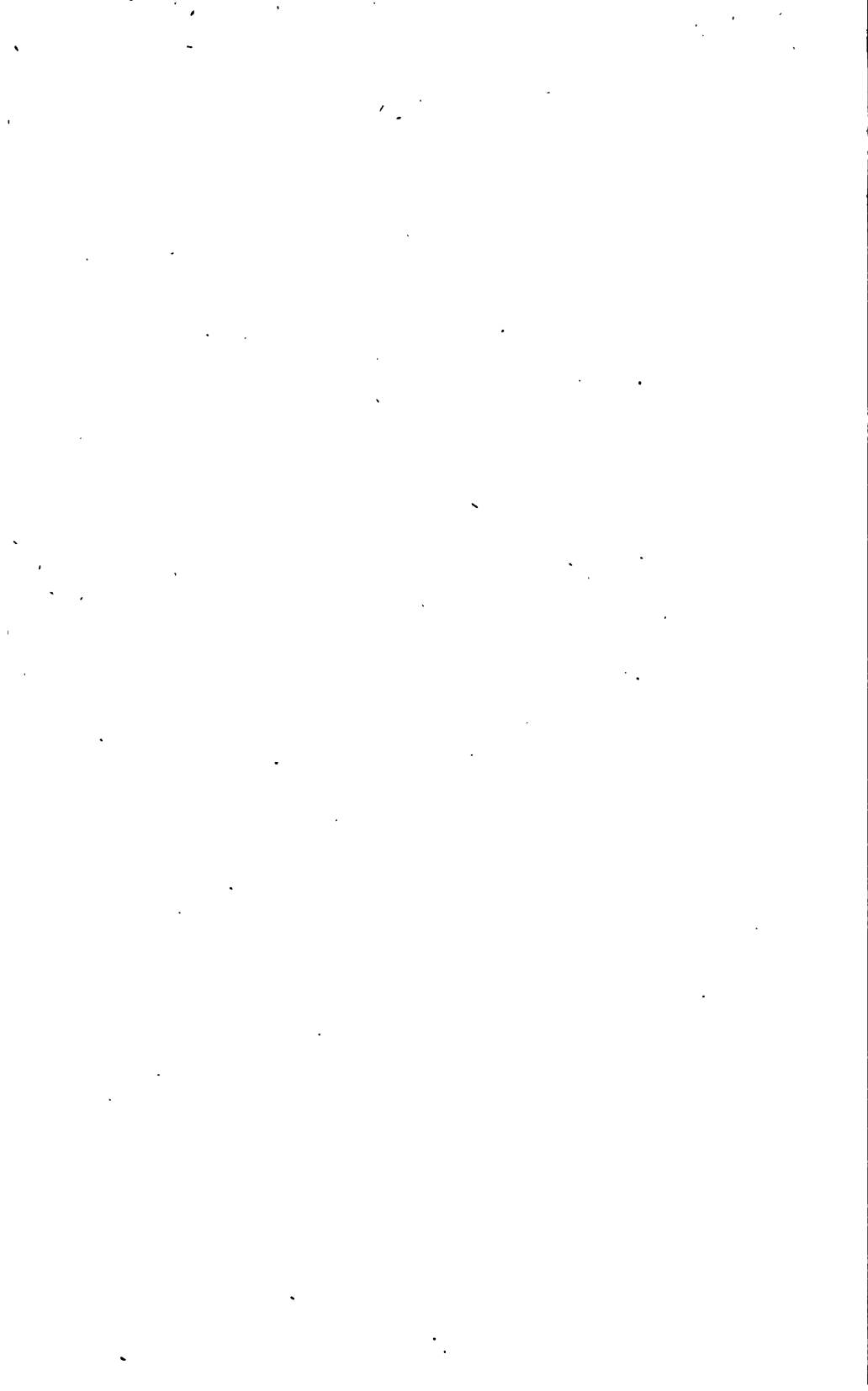
4. The flame safety lamp should be used in all gaseous mines irrespective of the use of electric cap lamps. These lamps form in themselves the best and most trustworthy gas detector yet devised and their presence is absolutely essential to the safe operation of such a mine. Even in non-gaseous mines, where electric cap lamps have replaced open lights completely, it is good policy to have a liberal proportion of flame safety lamps. The flame safety lamp is doubtless an unsatisfactory working light, but its rôle is twofold. It may be used as a working light, but it must be used as a gauge of the safety of the mine atmosphere.

5. There is a new acetylene safety lamp that gives promise of being an admirable working lamp as well as an excellent fire damp and black damp detector.

Candlepower of Various Types of Portable Miners' Lamps

	Candlepower	Cost per Shift, Cents
Miner's open oil cap lamp.....	1.50	2.4
Miner's open acetylene cap lamp.....	5.00	1.5
Electric cap lamp.....	1.10	
Davy safety lamp.....	0.12	
Clanny safety lamp.....	0.35	
Wolf-type safety lamp.....	0.65	
Akroyd and Best safety lamp.....	1.10	
T. M. Chance acetylene safety lamp.....	3.80	

NOTE.—The above candlepowers are in no sense maximum but are the average values over the field illuminated by the lamp in question and have been obtained from many determinations. These are the values that may be expected to be realized in practice under working conditions.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Notes on Flotation—1916*

BY J. M. CALLOW,† SALT LAKE CITY, UTAH

(New York Meeting, February, 1917)

THE results obtained by pneumatic flotation throughout the country on all classes of ore, and the tonnage now being treated by this particular method, speak for themselves. Its advantages over the so-called agitation process are now established by facts and data from practice. The comparative efficiency of a rapidly revolving paddle, as a means for aerating an ore pulp, and of a cycloidal or turbine blower for the same purpose, would seem to be so obvious as to leave no room for discussion.

Practitioners in this art are to be congratulated on the acquisition of the valuable literary contributions that have recently been made on the theory of flotation by Messrs. Ralston, Van Arsdale, Taggart and Beach, and others, and the very valuable and able articles by Messrs. Laist and Wiggin on "Flotation Concentration at Anaconda, Mont.," by Dr. Rudolf Gahl on the "History of Flotation at Inspiration," and by David Cole on "The Advent of Flotation in the Morenci District." It is not within the province of this article to attempt an answer to the various issues raised in regard to the theory, or to defend the one advanced by me a year ago. But since discussion usually benefits the profession, a reply will be made to some of the statements made by the last-named writers.

COMPARATIVE SENSITIVENESS AND ATTENDANCE

Concerning the article, "Flotation Concentration at Anaconda, Mont.," by Laist and Wiggin, a little explanation is in order. It should be stated that, with the exception of about 24 days at the end of the testing period, the Callow experimental plant at Anaconda was not run under my direction. This perhaps accounts for some of the conclusions drawn on page 562 of the descriptive paper. The conclusion that the use of acid seems to be of considerable advantage, calls for some comment. While the tests conducted by the Anaconda Copper Co. do show acid to be of considerable benefit, this is a consequence of using acid sludge as a flotation agent. My belief has always been that the Anaconda

* Originally presented at a meeting of the Utah Section on Dec. 16, 1916.

† Consulting Engineer.

ores (at least the reground sands) can be floated with coal and wood tar products in a neutral or alkaline circuit, but it was not until the 24-day period, referred to above, that an opportunity came to demonstrate it. Even in this short time, the results obtained very closely approached the best of those using an acid circuit. Still better results would probably have been obtained had sufficient time been available for the necessary experimenting, but the plant was shut down, because the grinding mill was required elsewhere.

The statement is made that "on account of utilizing the grinding mill as an agitator, the Callow machine requires less power than the Mineral Separation machine." By using a neutral circuit, the tube mill could have been used as an emulsifier, and the low-power consumption of the Callow scheme taken full advantage of, effecting a saving estimated at about 2,000 hp. for the eight operating sections, at the same time avoiding all the destructive effects of an acid circuit, which must amount to another appreciable sum. The grinding mill as an emulsifier is an advantage to a large extent thrown away with the Minerals Separation machine, but is taken full advantage of with the Callow, consequently it would appear that the conclusion that the Callow machine takes less power than the Minerals Separation could well have been made without the preliminary qualifying statement.

Referring to the statement that "the Callow machine is more sensitive and requires more attention than the Minerals Separation machine," the fact that at the Inspiration plant one man per shift handles four 800-ton sections of Callow machines, or 3,200 tons; while at the Miami Copper Co.'s plant two men per shift handle a like tonnage in 60 cells, or 1,600 tons per man, is sufficient evidence that Callow machines, at least at these plants, are neither sensitive, nor require much attention. At Anaconda on eight Minerals Separation machines, treating 3,000 tons per day, four men per shift are employed, each handling the equivalent of 800 tons; local opinion at Inspiration is that two 800-ton sections (consisting of two 10-compartment, and four 6-compartment Minerals Separation machines) is the limit that one man can properly attend. In other words, 1,600 tons per man with Minerals Separation machines compared with 3,200 tons per man in the Callow sections of the same plant is the limit of a day's work. It will be evident, from these comparisons, which machine is the more sensitive or requires the most attention.

Concerning the paper, "History of the Flotation Process at Inspiration," by Dr. Rudolf Gahl, a few comments are in order. It should not be overlooked that W. B. Thompson was the principal factor in its early application here. I had just completed the last of the gravity flow-sheet tests, when he brought the subject of flotation to the attention of his fellow directors, and it was immediately arranged that I should go to London with 20 tons of Inspiration ore, and watch tests by the Minerals

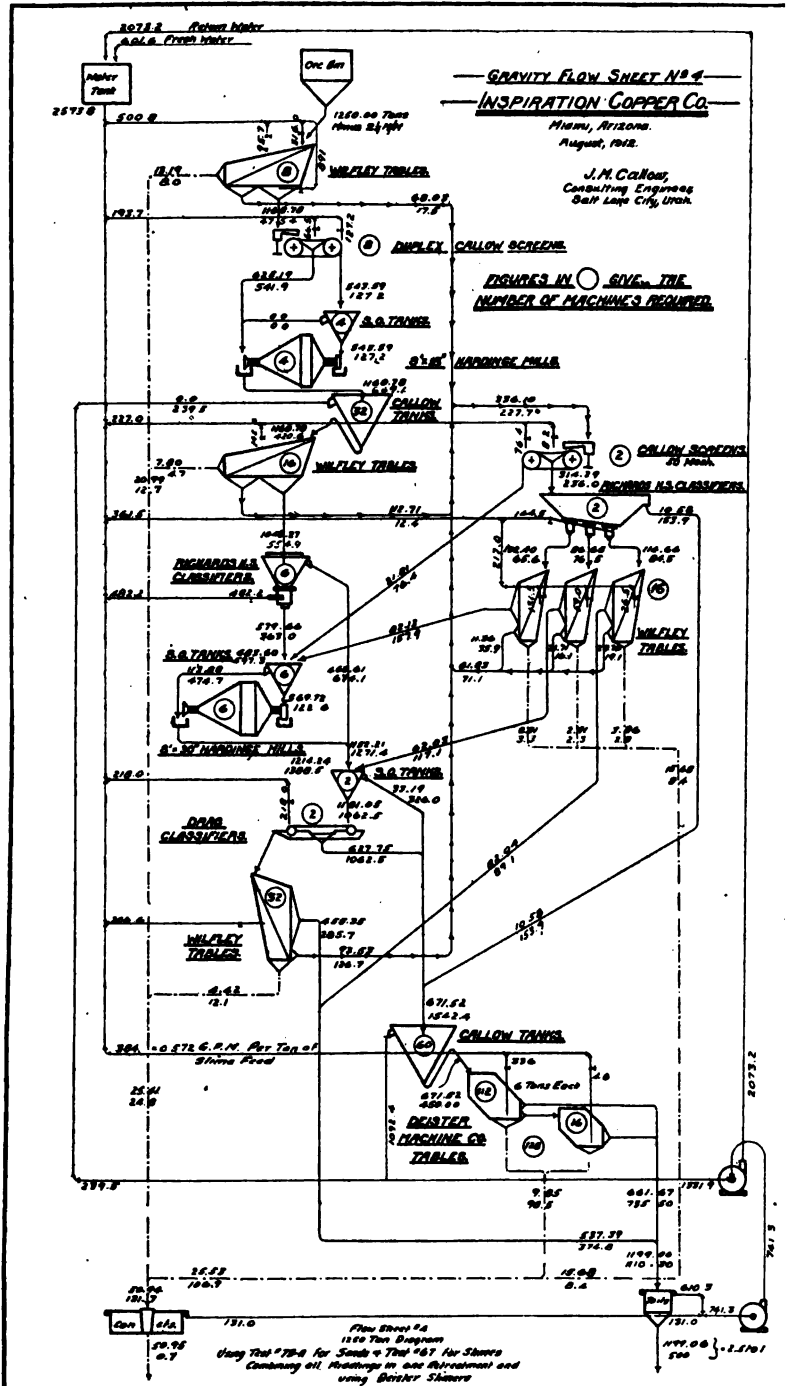


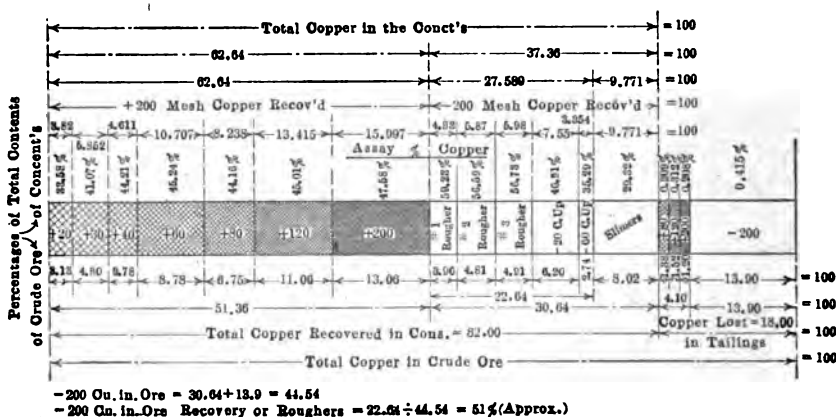
FIG. 1.

Separation Company men. This, however, was not to be, and neither the ore nor myself ever got farther than New York.

GRAVITY VS. FLOTATION RESULTS AT INSPIRATION

It might be interesting here to give some of the results obtained by this gravity flow sheet which the "all-roughing" system carried out to its logical conclusion. The results on three representative samples from the Joe Bush orebody are given in Table 1. The flow sheet eventually developed is given in Fig. 1.

The Wilfley tables employed had gradual terminating riffles, so that wherever clean mineral was available, it could be immediately removed. The middlings only were retailed. They were hydraulic classified, and re-treated on the secondary or cleanup tables. The chief virtue of the scheme was the gradual impoverishment of the slimes owing to the re-



1916, are given in Table 2. The comparison speaks well, I think, for the gravity work done, but it should perhaps be pointed out that the gravity results were on Joe Bush ore, which contained more chalcocite, less iron, and less oxide than the present regular mill feed. A composite screen analysis of the tailings from the three selected tests Nos. 68, 71, and 72, and those of the present practice, are compared in Table 3.

The gravity flow sheet called for 6 tons of water per ton of ore in circuit, and the present Inspiration practice takes about 5 tons so that neither the differences of recovery, crushing, or the water required, are so very conspicuous. The more self-evident benefits of the present practice are simplicity, and the much less space required. This latter cannot be better illustrated than by the fact that the present mill building, which was originally designed for 5,000 tons capacity when using the gravity flow sheet, is now treating within the same walls and roof, over 16,000 tons per day.

It will be evident from a study of all the published results, and also from Dr. Gahl's paper and the discussions which have followed it, that the most urgent problem facing the company is the recovery of the oxide copper in the tailings. The recovery of even 50 per cent. of the oxides lost (8 lb. per ton in August, 1916) would represent over \$2,500,000 a year on a 15-c. market, and raise the total copper recovery from 75 to 85 per cent. Further on in this article is described the oxide recovery method used by the Magma Copper Co., which should be of interest, and will be a partial answer to Dr. Gahl's remarks under this heading.

TABLE 1.—*Comparison of Gravity with Flotation Results, Inspiration Copper Co.—Gravity Flow Sheet*

	Heads		Tails	Concentrates	Recovery
	Total Cu	Oxide Cu	Total Cu	Total Cu	Total Cu
J. B. Dump test No. 68.....	1.84	0.10	0.47	36.89	75.91
J. B. R.O.M. test No. 72.....	1.96	0.10	0.30	38.25	80.95
Oxidized ore test No. 71.....	1.94	0.27	0.54	39.40	73.20

TABLE 2.—*Flotation Flow Sheet*

	Heads		Tails	Concentrates	Recovery
	Total, Cu	Oxide, Cu	Total, Cu	Total, Cu	Total, Cu
Callow results test plant, November-December, 1914.....	1.560	0.300	0.330	30.95	79.49
Inspiration Mill results, 2d half of 1915..	1.702	0.226	0.373	32.67	79.95
Inspiration Mill results, August, 1916...	1.564	0.392	0.417	30.22	74.36

TABLE 3.—*Comparison of Crushing, Gravity Experiments, and Present Inspiration Practice*

	Tailings Gravity Tests Nos. 68, 70, 71	Tailings Inspiration, Present Practice
Mesh	Per Cent.	Per Cent.
+ 48	3.00	2.90
+ 65	5.00	7.90
+100	9.00	12.60
+150	9.00	12.60
+200	14.00	5.80
-200	60.00	58.20
	100.00	100.00

DEVELOPMENT OF INSPIRATION FLOTATION MACHINE

It was on the invitation of the Inspiration Copper Co. that I installed the first pneumatic cell to be operated at their test plant in competition with Minerals Separation machines. At this time the tailings were assaying 0.5 to 0.6 per cent. copper, and the concentrates 23 to 25 per cent. copper, and were really inferior to those obtained by gravity. In the course of some 8 or 9 months of continuous competition, my results showed a final tailing of from 0.25 to 0.3 per cent. copper, and a concentrate of 30 per cent. and better. A summary of 2 months' results (November and December, 1914) showed a recovery of 2 lb. of copper per ton in excess of the best results of the other competitors, and a saving in power of some $2\frac{1}{2}$ c. per ton, or all told some 26 c. per ton, which on the present tonnage of 16,000 tons per day, and a 14-c. market, is equivalent to some \$1,500,000 per year, which, concisely stated, was my contribution to the history of flotation at Inspiration. A contract in the following January, for the use of my apparatus, was the result.

Concerning the so-called "Inspiration flotation machine" so fully illustrated and described in this paper, the development of which Dr. Gahl has explained, it can be fairly claimed that as all its essential features were anticipated and, moreover, reduced to practice prior to the installation of any flotation at Inspiration, it is a Callow machine in every essential. Neither metallurgically, nor mechanically, has this modification shown any advantage that I know of over my standard type; the blankets require more attention, and more frequent renewals, and on any feed containing a larger percentage of sand than the Inspiration feed now contains, they would be unworkable. The larger unit machine is a positive disadvantage, since when renewing blankets, 400 tons of capacity has to be lost or diverted, whereas on a smaller subdivision of units, as in the Callow cell, only 50 tons or less has to be diverted.

CALLOW CELLS ADOPTED BY ARIZONA COPPER CO.

Concerning the paper, "The Advent of Flotation in the Clifton-Morenci District," by David Cole, a word in passing. The Arizona Copper Co.'s mill, as the result of the competition referred to, is now operating with a full equipment of 63 Callow cells. It is believed that the tables of results in Mr. Cole's paper are apt to be misleading, though doubtless not so intended, since neither in May nor June was there any direct competition between machines, neither the feed, nor the contributory flow sheet, nor the crushing being the same to both machines. The actual competition commenced on July 10, and ended on July 28, and even then only on the last 10 days were the conditions truly parallel. The decision of the company in favor of the Callow machine was made for the following reasons: That it would make a higher recovery of from 1 to 1.2 lb. of copper per ton; it was simpler to operate, and the operating costs would be lower—two men as against eight; its power costs were lower—3.6 hp. against 5.9 hp. per ton.

In the statement as to the floor space required, it should be pointed out that the vertical mill height required for the C-B machine from the feed inlet to the tailings outlet is 12 ft., for the Callow it would be 6 ft. Machines have recently been designed by which this is easily reduced to 3 ft. Generally speaking, vertical mill height is more at a premium than floor space. However, this is a consideration quite secondary to recoveries, attendance, and power.

MAGMA COPPER CO. INSTALLATIONS

Three different wet milling methods on the ores from one mine is somewhat unique, and it is thought, therefore, that a description of them may be of interest, especially that relating to the flotation of copper oxides.

Description of Ores

The milling ores of this company have kindly been described for me by the manager, W. C. Browning, as follows:

"All ores occur in a porphyry-filled fissure vein. Three classes of milling ores are being mined, as follows:

"*First*.—A copper sulphide ore which contains varying amounts of bornite, chalcopryite, pyrite, and chalcocite, impregnated or ribboned through a gangue of altered, siliceous porphyry or altered diabase.

"*Second*.—An oxidized ore from the upper levels of the mine, which contains malachite, chrysocolla, and at times a small amount of cuprite, native copper, chalcocite and covellite. This ore is an oxidized product of the sulphide ores.

"*Third*.—A zinc sulphide ore, containing the black form sphalerite, galena, pyrite and small amount of chalcopryite. This ore occurs in separate shoots from the copper ores. The gangue is usually a very siliceous, altered porphyry. Both copper and zinc ores usually carry an ounce of silver to the per cent. copper or zinc."

Sulphide Copper Plant Flow Sheet

A flow sheet is given in Fig. 3, which is a combination of hand sorting, gravity, and flotation. The 6 by 4½ Marcy mill, here shown, was the first machine of this type installed commercially. Some early troubles were experienced from overspeed, and too much pulp dilution. Since then it has given excellent service. The pulp density is now 60 per cent. solids, and the revolutions 22 per minute.

As first installed, flotation followed gravity, with Dorr tanks for thickening the feed in the interval. The principal defect of this arrangement was the large amount of pulp storage ahead of flotation, so that adjustment of the oil, when fed in the tube mill, could not be followed in the flotation plant. This was changed and flotation was sandwiched in between the first and second tabling. The lower cells are still retained in operation as "gleaners." These gleaner cells, at the present time, contribute something less than 5 per cent. to the total recovery and take care of any irregularity in the operation of the main plant.

The distribution of labor is as follows:

		Total Men
Ore sorting and coarse crushing.....	1 shift of 12 men	12
Table floor.....	3 shifts of 2 men	6
Flotation.....	3 shifts of 1 man	3
Filter presses and concentrate settling tanks....	3 shifts of 2 men	6
Shift bosses.....	3 shifts of 1 man	3
Ore loaders.....	1 shift of 4 men	4
Watchmen.....	2 shifts of 1 man	2
Total operating.....		36
For repairs, general cleaning up, tailings dam upkeep:		
Carpenters.....		2
Head repairmen.....		1
Repairmen helpers.....		5
Roustabouts.....		16
Sampler.....		1
Total.....		61
which equals 3.6 tons per man.		

The distribution of power is as follows:

	Hp.	Shifts
Sorting plant.....	45	1
No. 64½ Marcy mill.....	45	3
5 by 12-in. tube mill.....	45	3
Concentrating machinery and miscellaneous.....	80	3
Flotation (15 cells).....	80	3
Pumping water.....	15	3

6,720¹ hp.-hr. per ton

which equals 22.8 kw.-hr. per ton.

¹ The above is "motor" horsepower-hours—not net horsepower-hours.

Part of the flotation concentrates are now settled by Dorr continuous thickeners, and part in intermittent decantation tanks.

Magma Zinc Plant Flow Sheet

This flow sheet, given in Fig. 3, is substantially the same as the one at the sulphide copper plant. The main difference is that a drag classifier is used in closed circuit with a Marcy mill instead of a Callow screen. This mill crushes to 48 mesh instead of 10 mesh, as in the sulphide mill. A 6-ft. Hardinge mill, used for the final crush, is loaded with steel pebbles. The final crushing is now to 150 mesh. The plant was designed for 50 tons a day, which it will handle with ease.

For commercial reasons, the plant is now temporarily treating about 75 to 100 tons per day of sulphide copper ore, with results that are fully up to the standard for the sulphide copper plant. The only change necessary was to change the launders to the tables handling the zinc flotation concentrates. With the addition of the two more flotation cells, it is expected to raise the tonnage to 150 tons per day.

This ore was first tested by the General Engineering Co. with a combination of roasting and magnetic separation, gravity, and flotation. This gave excellent results, but the process was complicated and costly. Persistent experimenting with flotation eventually led to the present flow sheet.

TABLE 4.—*Magma Copper Co. Results for Month of October, 1916*

Sulphide Mill

	Average Tons per Day	Assay in Copper		Recoveries, Per Cent.	
		Total	Oxides		
Hand-sorted ore.....	31.035	15.500		38.9	} = 92.7 per cent.
Mill feed.....	187.565	4.042		53.8	
Mill concentrates.....	50.670	13.165		7.3	} = 88 per cent. of mill feed
Mill tails.....	136.895	0.660	0.11	100.0	
Crude ore.....	218.600	5.672			

Month of September

	Average Tons per Day	Assay in Copper		Recoveries, Per Cent.	
		Total	Oxides		
Hand-sorted ore.....	17.830	18.000		28.97	} = 93.39 per cent.
Mill feed.....	188.590	4.173		64.42	
Mill concentrates.....	49.563	14.400		6.61	} = 90.02 per cent. of mill feed
Mill tails.....	139.027	0.529	0.14	100.00	
Crude ore.....	206.420	5.672			

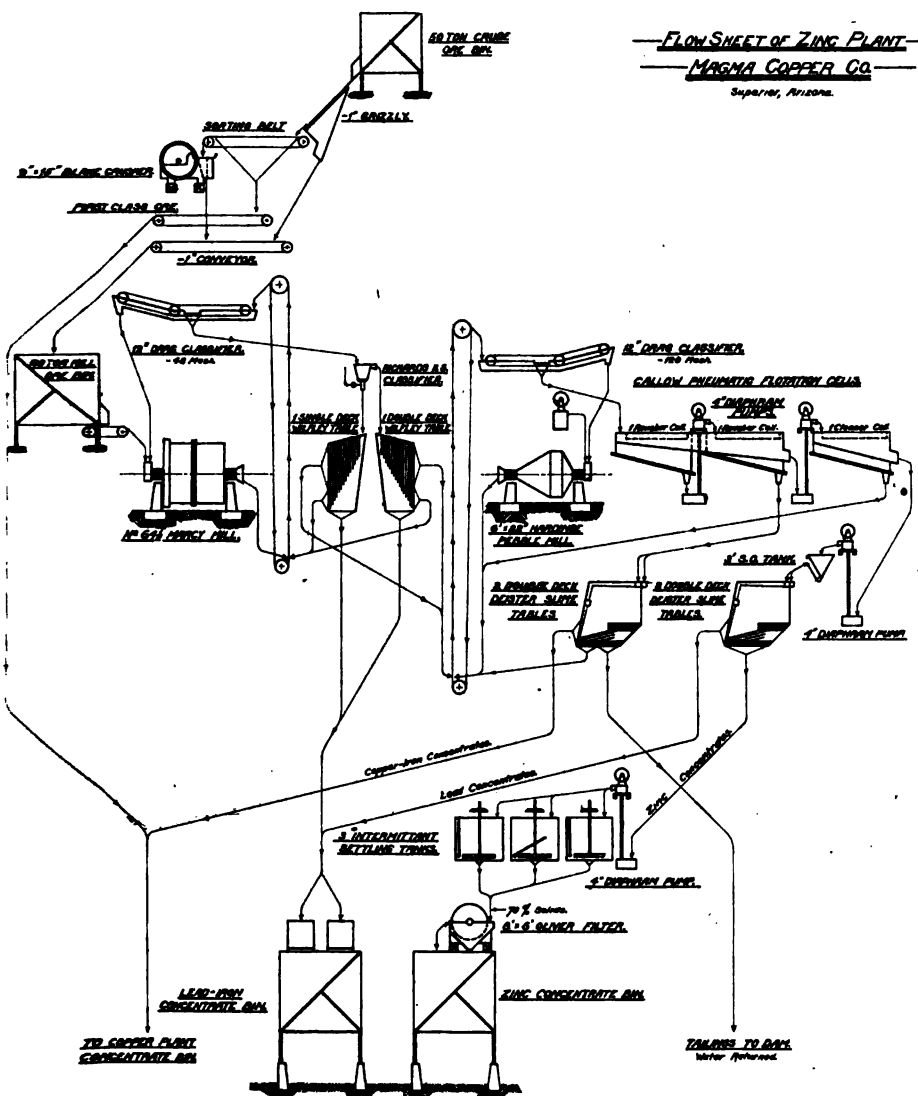


FIG. 4.

TABLE 5.—*Magma Copper Co., Sulphide Mill, Milling Costs for September and October, 1916*

	Costs per Ton Crude Ore	
	September	October
Ore sorting.....	\$0.2130	\$0.230
Coarse crushing.....	0.0879	0.150
Marcy mill.....	0.0790	0.087
Tube mill.....	0.0637	0.086
Callow screen.....	0.0143	0.010
Tables.....	0.1840	0.200
Flotation.....	0.3210	0.345
Filter pressing.....	0.0746	0.072
Total.....	\$1.04	\$1.18

The above costs include all items of direct labor, supplies, maintenance, and power, but no general expense or overhead charges.

It was found that No. 34 Gravity fuel oil in combination with General Naval Stores No. 17 oil had the property of floating the zinc in preference to the iron. These tests have been substantially duplicated in practice.

Some trouble was experienced at first in making both the desired grade of zinc and tailings. Frequent changes in the grade of ore, and the percentage of iron and zinc, owing to unavoidable conditions at the mine, made it difficult at first to get the desired results. It was also found that returned water from the sulphide mill pond contained sufficient coal-tar products to float the iron, thereby reducing the grade of zinc. Trouble was experienced at first with the oils which had lost some of their volatile constituents, due to hot weather. After correcting these conditions, results steadily improved.

The use of a small quantity ($\frac{1}{10}$ lb.) of copper sulphate was decidedly beneficial, reducing the tails from 2 to 3 per cent. zinc without materially affecting the grade of the concentrates. Prior to the use of copper sulphate, acid was tried and, while it lowered the tailing, it also lowered the concentrates. The dilution was 6 to 1, and was an important item in maintaining the grade of concentrates. The cells were sometimes run in parallel, and sometimes in series. No differences in results were noticed.

The mill proper was operated with two men per shift. As it was found necessary to operate the sorting and crushing plants only a few hours per day, no labor was here employed regularly; two men were taken from the crushing department of the sulphide plant for this work. Some iron and copper was hand-sorted at this plant, and, together with the iron concentrates from the —150-mesh tailing tables, was shipped with the regular product of the sulphide plant.

The intermittent settling system, as indicated in the flow sheet, was decidedly successful, delivering a 70 per cent. solid feed to the filter with a clear overflow from the tank.

A summary of results is given in Table 6, showing the difference in results with and without copper sulphate.

Magma Sulphide Filming Plant

The process in use in this plant is covered by Schwarz' U. S. patent No. 807501, which is the first disclosure of the use of a soluble sulphide for converting an oxide of a metal into a superficial sulphide, and afterward recovering it by a flotation process.

Our earliest experiments were made with H_2S gas as the filming agent. A plant of 25 tons daily capacity was built. In this the gas was applied to the pulp by introducing it into the bottom of an open tank, having a mixing agitator. The results were encouraging, but the consumption of gas prohibitive—as much as 8 or 10 lb. per ton. The ore treated was the tailings of the Magma sulphide mill, which at that time carried considerable oxides. Occasional recoveries of 60 per cent. (of the total copper) were made, but the results were erratic owing to the difficulty of getting uniform filming of the pulp with this method of applying gas. Then followed an interval of several months when we used sodium sulphide, calcium sulphide, and calcium sulphydrate in an endeavor to avoid the use of gas, on the assumption that it was objectionable owing to the danger of its poisoning the surrounding atmos,

TABLE 6.—*Magma Copper Co. Zinc Plant Results With and Without Copper Sulphate*

Without Copper Sulphate, Oct. 1 to 19, 1916	Zinc	Iron	Lead
Heads.....	15.38	7.80	3.89
Zinc concentrates.....	39.55	7.96	8.16
Lead concentrates.....	12.43	22.60	20.04
Tails.....	6.50	4.62	0.81
With Copper Sulphate, Oct. 20 to 31, 1916	Zinc	Iron	Lead
Heads.....	12.50	8.80	3.50
Zinc concentrates.....	38.50 ¹	9.16	6.57
Lead concentrates.....	15.93	24.01	12.60
Tails.....	3.23	4.56	0.78

¹ The lower zinc concentrates in the latter period are due to lower heads and not to use of copper sulphate.

phere. This was true when attempting to use it in an open tank, much of it being lost in the atmosphere. During this time a number of theories were advanced, exploited, and abandoned. One of these was that natural and artificial sulphides could not be floated together, and that H_2S interfered with the flotation of the natural sulphides. Our experience now is that H_2S , in the proper quantity really promotes flotation of natural sulphides in company with the filmed oxides, and also that it is immaterial whether the oiling is done before or after filming, and whichever plan is followed is merely a matter of convenience.

In treating Magma sulphide tails, in which the principal losses were sulphides, the introduction of the gas not only filmed the oxides present, a goodly percentage of which was recovered, but it also raised an entirely new crop of refractory sulphides. An all-sulphide sample of regular Magma ore was tested with and without H_2S . The results are given in Table 7.

The commercial plant was shut down and we again reverted to laboratory work, which resulted in our adherence to H_2S in preference to any other agent on this particular ore, and also to a complete change in our method of applying it. The plant was reconstructed, crushing machinery and additional cells added in accordance with the flow sheet shown in Fig. 5, for the purpose of treating Magma oxidized ores on a commercial scale.

The gasing method used at present introduces the gas into the suction of a centrifugal pump in the manner indicated in the flow sheet. This has proved effective, greatly reducing the gas consumption, giving more uniform recoveries, and has removed all danger from the poisoning of the atmosphere; in fact, the commercial results now being obtained date from the first use of this expedient.

TABLE 7.—*Effect of H_2S On Natural Sulphides*

Magma Sulphide Ore				
No H_2S Gas	Heads	Tails	Concentrates	Recovery
Test 1.....	2.45	0.22	11.82	92.70
Test 2.....	2.45	0.25	9.62	92.20
Average.....	2.45	0.235	10.72	92.45
With H_2S Gas to Excess				
Test 1.....	2.45	0.28	9.40	91.30
Test 2.....	2.45	0.20	8.41	94.10
Average.....	2.45	0.24	8.90	92.70

Manufacture of H_2S Gas

The present method of making H_2S gas is to heat sulphur and oil in a retort. Various proportions of sulphur and oil have been tried, our present proportions being 1 of sulphur to $2\frac{1}{2}$ of oil. The temperature in the retort is kept at a uniform $300^\circ C$. At times the making of gas

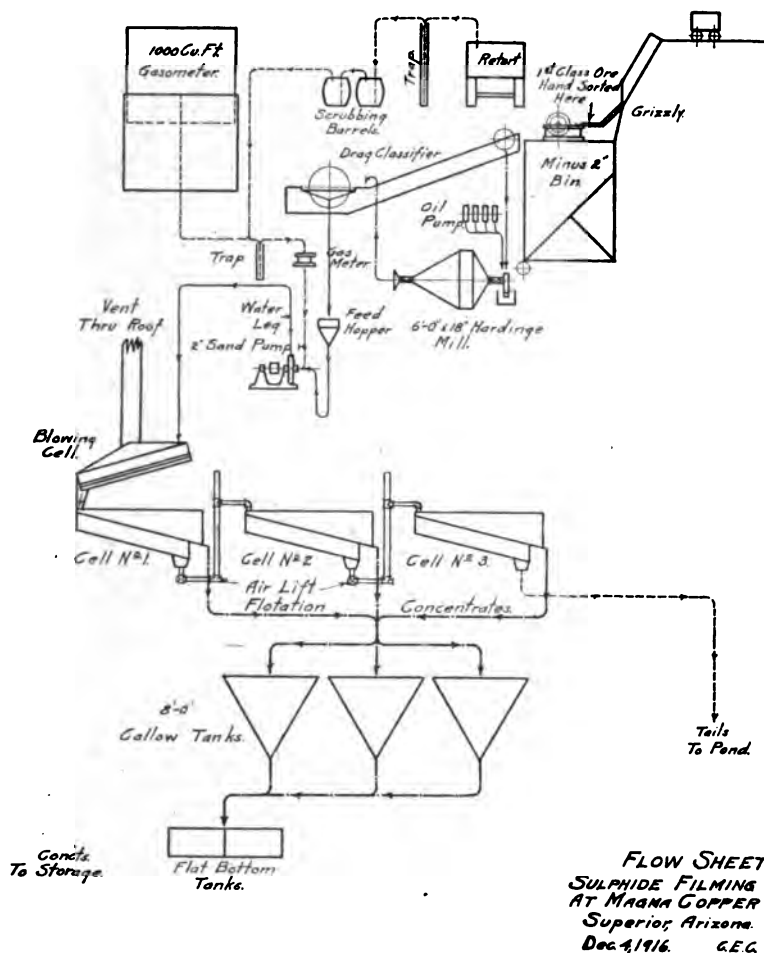


FIG. 5.

has given considerable trouble, owing almost entirely to changes from time to time in the quality of the oil used. Satisfactory results have been obtained at all times with California crude oil, but Texas oil always gives trouble, making a gas containing what we believe to be hydrogen persulphide, which interferes with flotation, and can always be identified by its eye-burning properties. This is in a measure overcome by careful

scrubbing, sulphur being precipitated in the scrubbers with free H_2S liberated.

The California crude oil has the following fractional analysis:

Specific gravity 0.9311 or 20.36 B $\acute{e}$. at 15° C.

Flash point—open dish—108° C.

Fractionation:

Temperatures	Distillate, C.c.
100° C.—150° C.	1.2
150° C.—175° C.	1.6
175° C.—200° C.	3.6
200° C.—225° C.	9.6
225° C.—250° C.	22.0
250° C.—275° C.	30.8
275° C.—300° C.	25.2

Free gas in a pulp is fatal to flotation, hence the use of the blowing cell at the head of the first flotation cell. Experiments indicate that heating the pulp slightly before gasing is beneficial.

The cost of manufacturing the gas by this method will, of course, vary greatly according to local conditions. Those at Magma are abnormal. Sulphur is costing nearly 3 c. today, and oil nearly 1 c. per pound, f.o.b. Superior, and costs on a basis of 30 tons per day, and maximum of 3 lb. per ton, stand as follows: 90 lb. of sulphur at 2.74 c., \$2.51; 225 lb. of oil at 0.914 c., \$2.16; total, \$4.77 = 15.25 c. per ton, or approximately 5 c. per ton for each pound of sulphur per ton required by the ore.

On Magma sulphide tails, the gas consumption varies from $\frac{1}{2}$ to $1\frac{1}{2}$ lb. sulphur per ton. On strictly carbonate ore, assaying 3 or 4 per cent. copper, 3 lb. is an average figure, and on the latest test with a mixed carbonate and silicate ore, assaying from 4 to 5 per cent. copper, 2 lb. of sulphur per ton. The fuel required for heating the retort is almost negligible. There are no cost figures, since so far we have been burning scrap lumber left over from construction. The labor item is unduly heavy since the retort is situated some 600 ft. away, and one man has to be held in reserve for the purpose. He could as easily make gas on one shift for 500 tons per day. With sulphur and oil at moderate prices on a 500-ton scale, using 2 lb. of sulphur per ton, my estimate of total gas cost is as follows:

1,000 lb. of sulphur at \$45.00 per ton = 2.25 c. per pound =	\$22.50
2,500 lb. of oil at \$1.75 per barrel = 0.436 c. per pound =	11.00
1 man at \$4.00	= 4.00
Extra fuel for heating retort and sundry repairs =	2.50
	\$40.00
	per day

Thus the probable cost is 8 c. per ton of ore, or 4 c. per ton for each pound of sulphur required for the ore.

Other methods were tried—one using powdered coal in a separate retort instead of the oil mixture, and another in which the oil instead of being mixed with the sulphur was dripped into the sulphur retort with a force feed lubricator. The usual iron matte and sulphuric acid method was also tried but none of these methods had anything to recommend them over the one adopted. Iron matte was never seriously considered on account of the recent high price of acid.

The flow sheet given in Fig. 5 is self-explanatory: The 6-ft. by 16-in. Hardinge ball mill has a capacity of 35 tons per day when loaded with balls requiring 35 hp., or 45 tons per day when loaded up to 50 hp. crushing from crusher run to 83 per cent. —150 mesh.

The power requirements are distributed as follows:

1 6-ft. by 16-in. Hardinge ball mill.....	35 hp.
1 8 ft. by 14 $\frac{1}{8}$ Root blower, No. 1 (400 cu. ft. at 5 lb.).....	} 20 hp.
1 7 by 14 Dodge crusher (1 shift only).....	
1 2-in. centrifugal gasing pump.....	
1 4-in. diaphragm pump.....	
1 10-in. belt drag classifier.....	
1 Oil feeder.....	} —
1 Ore feeder.....	
	55 hp.

This is the equivalent of 37.6 hp.-hr. per ton, = 28 kw.-hr. per ton.

The plant is operated by four men for the three shifts, the extra man on day shift crushing the ore and hand sorting out any first-class ore there may be at the same time.

Magma Results Summarized

Table 8 gives the results of gasing the sulphide mill tails, the feed in all these experiments being the feed to the lower cells or gleaner cells, as they are described on the flow sheet. The soluble copper in these tests varied from 0.3 to 0.45 per cent. and the average for the entire period was 0.35 per cent. Attention is directed to tests Nos. 63A and 67, giving results on this feed without gasing; further comparisons of more direct kind are given in Table 10.

The results on strictly carbonate ore are given in Table 11, and on a mixed carbonate-silicate-sulphide ore in Table 12. The latter tabulation gives a complete record, and shows the contribution that hand sorting makes to the total recovery. The carbonate results, for the most part, were on the rejections from the hand sorting done at the mine, but in all these results the heads, referred to, are actual mill feeds, and the effect of hand sorting is, therefore, not shown. On the mixed ore, gravity tables on the flotation tails would have added considerable to the total recovery but on the straight carbonate ores they would have served no purpose.

TABLE 8.—Summary of Results in Treating Sulphide Mill Tails by Filming, Magma Copper Co., Superior, Ariz., 1916

Test No.	Heads	Tails	Concentrates		Tonnage Oils, Per Cent. Weight						Retort Mixture		Remarks on Gas, Etc.	
			Rougher	Cleaner	Recovery ¹	Rate	C.T.	C.T.C.	F.O.	P.T.O.	C.P.O.	S		O
56	1.10	0.64	8.25	12.10	65.0		60	40	..	..	..	23	77	Excessive heat on old gas charge, test 57. Moderate heat, 1st firing, 0.77 lb. S per ton. No gas used. Same retort as 62C, 2d firing, 0.36 lb. S per ton. Same retort, 3d firing, poor gas. Gas from iron matte. No gas used. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. Slow continuous firing, trying effect of oils. To try out different oil mixtures. ² To try out different oil mixtures. To try out different oil mixtures. To try out different oil mixtures. Trouble with gas. Repetition under better conditions. Repetition under 1-1/2-ter conditions. Repetition under better conditions. But with overheated gas.
57	1.32	0.96	8.60	12.00	31.0		60	40	..	..	..	23	77	
62B	1.15	0.88	10.50	14.00	25.0		60	40	..	..	..	23	77	
C	1.15	0.69	10.50	14.00	43.0		60	40	..	..	..	23	77	
63A	1.40	1.21	11.90	18.30	15.1		60	40	..	..	..	23	77	
B	1.40	0.96	12.90	15.10	34.0		60	40	..	..	..	23	77	
C	0.74	0.58	5.10	8.40	24.4		60	40	..	..	..	23	77	
64	1.24	0.69	5.40	9.80	50.9		60	40	..	..	..	23	77	
66	0.91	0.63	7.00	8.20	33.2	29.40	60	40	..	..	..	23	77	
67	0.58	0.47	5.00		21.0	29.40	60	40	..	..	..	23	77	
68	0.83	0.49	6.70	12.50	44.2	31.20	60	40	..	..	..	23	77	
69	1.16	0.83	12.20		30.9		..	40	60	..	..	23	77	
70A	0.94	0.34	7.90	15.10	67.1	28.80	55	38	..	9	..	23	77	
B	0.66	0.37	5.00	15.70	53.1	21.60	60	40	..	..	..	23	77	
71B	0.91	0.44	7.90	15.70	54.5	34.00	55	38	..	9	..	23	77	
72	1.08	0.47	7.20	11.50	60.5	16.50	..	..	..	100	23	77	77	
73	0.88	0.44	7.40	17.10	53.3	27.20	55	36	..	9	..	23	77	
74	0.86	0.49	7.70		45.8	17.30	60	40	..	..	..	23	77	
75A	0.76	0.38	3.80	8.80	55.6	31.20	32	20	..	25	23	77	77	
76B	0.76	0.31	3.50	12.80	65.8	31.20	30	45	10	25	..	23	77	
75C	0.76	0.38	7.40		53.5	31.20	30	20	25	25	..	23	77	
76	0.80	0.47	4.10	17.20	46.6	31.20	30	45	..	25	..	23	77	
77	0.69	0.29	3.80	12.00	62.6	24.15	30	45	..	25	..	23	77	
78 ²	1.00	0.35	4.85	14.80	70.1	29.30	30	45	..	25	..	23	77	
79	1.23	0.42	7.40	11.25	69.9	30.90	30	45	..	25	..	23	77	
80	1.01	0.49	4.85	12.90	57.3	27.30	..	..	..	..	..	..	..	

¹ Based on rougher concentrates only. ² For complete analysis of heads and tails see Table 9. ³ Pulp was already oiled and additional oils had very little effect on results.

C.T. = coal tar. C.T.C. = coal-tar creosote. F.O. = fuel oil. P.T.O. = pine tar oil. C.P.O. = crude pine oil.

TABLE 9.—*Screen Analysis*

Test 78

	Heads				Tails			
	Per Cent. Weights		Assays, Per Cent. Cu		Per Cent. Weights		Assays, Per Cent. Cu	
Screen Size	Per Cent. Indi.	Total Cu	Oxide Cu	Sulphide Cu	Per Cent. Indi.	Total Cu	Oxide Cu	Sulphide Cu
+ 60	8.3	0.63	0.16	0.47	10.8	0.490	0.14	0.350
+100	25.6	0.77	0.29	0.48	25.6	0.420	0.21	0.210
+200	20.8	0.75	0.35	0.40	24.4	0.350	0.16	0.190
−200	44.8	1.32	0.67	0.65	39.2	0.310	0.17	0.140
Total...	100.0	1.00	0.44	0.56	100.0	0.368	0.17	0.198
Assay...		1.00	0.44			0.350	0.17	0.180

TABLE 10.—*Effect of H₂S on Mixed Sulphides and Oxides in Magma Sulphide Tails*

	Heads	Tails	Concentrates	Recovery
No H ₂ S	0.86	0.67	9.86	23.70
	0.78	0.60	9.63	24.70
Average	0.82	0.635	9.74	24.20
With H ₂ S	0.81	0.30	5.05	67.11
	0.78	0.29	5.51	66.38
Average.....	0.79	0.295	5.28	66.24

In all these tests pine tar oil was used, the tonnage rate was 25 per day and the sulphur consumption $1\frac{1}{2}$ lb. sulphur per ton of feed; the soluble copper was 0.3 per cent. The concentrates "with H₂S" are rougher concentrates. The arrangement of plant did not permit of cleaning the filmed concentrates. The concentrates "no H₂S" were cleaned in the regular way.

TABLE 11.—*Summary of Results in Treatment of Oxide Ore by Filming at Magma Copper Co., 1916, by Flow Sheet (Fig. 5)*

Carbonate Ores

Date	Test No.	Tonnage Rate	Heads	Tails	Concentrates	Ratio	Recovery
July 8.....	150	35.00	2.78	0.86	9.50	4.50	75.94
July 9.....	151	30.00	3.15	1.06	9.72	4.18	74.31
July 12.....	154	26.00	2.92	1.17	7.02	3.34	72.00
July 13.....	155	30.00	3.24	1.22	11.66	5.16	69.72
July 14.....	156	33.00	2.92	1.35	10.26	5.67	61.94
July 15.....	157		2.92	1.08	8.21	3.89	72.27
July 16.....	158	28.00	2.75	0.83	8.30	4.14	77.25
July 18.....	159	30.00	2.99	0.70	8.32	3.32	83.78
July 20.....	161	30.00	3.08	1.08	9.18	4.05	73.62
July 23.....	164	31.00	3.92	1.62	10.80	3.95	69.76
Average.....		30.33	3.06	1.09	9.34	4.22	73.08

NOTE.—Best results in July tests were obtained with a consumption of about 4 lb. sulphur and 10 lb. fuel oil per ton feed.

Date	Test No.	Tonnage Rate	Heads	Tails	Concentrates	Ratio	Recovery
Aug. 9.....	175	25.0	3.04	0.71	14.85	5.41	80.49
Aug. 10.....	176	30.0	3.63	0.81	16.07	5.40	81.82
Aug. 11.....	177	27.0	5.46	1.25	17.36	3.82	83.22
Aug. 12.....	178	25.0	5.46	1.87	22.74	5.72	72.81
Aug. 13.....	179	27.0	5.12	1.43	16.80	4.17	78.68
Aug. 14.....	180	25.0	5.04	1.85	16.91	4.72	71.09
Aug. 17.....	181	25.0	4.23	1.62	18.48	6.46	67.64
Aug. 18.....	182	25.0	11.66	2.65	26.50	2.65	85.76
Aug. 19.....	183	25.0	6.73	2.67	22.26	4.82	68.62
Aug. 20.....	184	25.0	8.35	2.45	20.67	3.09	80.12
Average.....		25.9	5.87	1.72 ?	19.26	4.63	77.025

NOTE.—Sulphur consumption on tests 181, 2, 3, 4 was 6 lb. per ton. In the other August tests, a minimum of 2.25 and maximum of 3 lb. per ton.

Date	Test No.	Tonnage Rate	Heads	Tails	Concentrates	Ratio	Recovery
Sept. 25.....	185	28	4.90	1.33	14.8	3.77	80.0
Sept. 27.....	186	27	5.50	0.79	17.8	3.61	90.0
Sept. 28.....	187	30	5.00	1.50	18.1	4.74	76.4
Sept. 29.....	188	25	3.10	0.61	13.9	5.30	84.7
Sept. 30.....	189	24	2.80	0.54	13.2	5.60	84.1
Oct. 1.....	190	23	2.30	0.58	15.1	8.40	77.8
Oct. 2.....	191	24	2.40	0.68	14.4	8.00	75.0
Oct. 3.....	192	27	2.60	0.41	14.3	6.30	86.0
Oct. 3.....	192	27	2.70	0.36	14.9	6.20	89.2
Oct. 4.....	193	..	3.60	0.61	10.3	3.60	89.6
Average.....		26	3.49	0.74	14.68	5.55	83.28

NOTE.—The September and October results were obtained with a sulphur consumption of 2.8 minimum and 3.5 maximum per ton.

TABLE 12.—*Summary of Results in Treatment of Oxide Ores by Filming at Magma Copper Co. by Flow Sheet (Fig. 4)*
Mixed Carbonate, Silicate and Sulphide Crude ore

Description	Oct. 14 to 27, 1916				Nov. 1 to 10, 1916			
	Tons	Assay, Per Cent. Cu	Contents	Recovery on O.F.	Tons	Assay, Per Cent. Cu	Contents	Recovery on O.F.
Crude ore received for period.....	173.300	5.61	972.0	100.0	183.10	5.19	1,002	100.0
High-grade sorted out.....	3.772	22.70	85.6	8.8	10.98	26.35	289	28.9
Mill feed.....	169.528	5.22	886.4	91.2	182.12	3.91	713	71.1
Mill concentrates.....	31.300	16.90	592.0	60.9	25.80	18.70	474	47.4
Mill tails.....	138.228	2.14	294.4	30.3	156.82	1.52	239	23.7
Sum of high-grade and concentrates.....	35.072	19.30	677.6	69.7	36.28	21.06	763	76.3
Mill ratio.....			5.4 to 1				5.02 to 1	
Mill recovery.....			66.5 per cent.				66.5 per cent.	
Sulphur consumption per ton crude.....			4.66 lb. per ton. High consumption due to				2.06 lb., including all line losses and leaks.	
Oil consumption per ton crude.....			faulty oil used. Figure therefore of little value.				5.2 lb.	
Flotation oils used.....			Coal tar..... 80 per cent.				Coal tar	
			Coal-tar creosote..... 30 per cent.				Coal-tar creosote	
			G.N.S. No. 17..... 40 per cent.				Pensacola, No. 400.	
Composition of heads.....			Carbonate..... 1.01				Carbonate..... 2.110	
			Silicate..... 0.89				Silicate..... 0.865	
			Sulphide..... 3.33				Sulphide..... 1.490]	
			Total Cu..... 5.22				Total..... 4.460 (3.91)	
Approximate screen analysis of crushing.....			+150 = 13.6 per cent.					
			-150 = 86.4 per cent.					

NOTE.—No tables were used in this flow sheet. The results would have been improved if there had been.

TABLE 13.—*Comparison of Laboratory Miniature Tests with Larger Scale Operations, Consolidated Utah-Nevada Corporation*

	Mill Results, Aug. 14, 1916	Miniature Results on Daily Head Sample of Aug. 14, 1916,		5-Ton Laboratory Test of Aug. 14, 1916	Miniature Results Prior to 5-Ton Laboratory Tests
		Salt Lake City Water	Mine Water		
Assay of heads, per cent. Zn.....	17.20	17.56	17.56	17.54	17.53
Assay of total zinc concentrates, per cent. Zn...	41.80	40.48	40.17	42.41	41.91
Assay of total tails, per cent. Zn.....	5.60	4.89	4.89	3.64	4.71
Ratio of concentration.....		2.91	2.81	2.87	2.94
Recovery of Zn in Zn concentrates.....	77.80	78.58	79.07	84.12	81.48
Assay of all table zinc concentrates, per cent. Zn.	42.60	40.63	40.63	41.32	41.77
Assay of flotation zinc concentrates, per cent. Zn.	37.00	39.80	38.52	46.77	43.19
Assay of zinc in total iron concentrates, per cent. Zn.....	10.10	14.30		12.88	10.06

While on carbonate ores the results may be considered very satisfactory, there is still considerable work to be done on the mixed ores. Several points are still obscure, which will take time, further experimenting and research to overcome. These are: (a) Uniform quality of gas, and what are the interfering elements in poor gas? (b) Why are silicates and the coarser sizes of carbonate mineral more difficult to film than fine carbonates?

The present results are encouraging, and positive enough to lead us to believe that in due time all such ores as these will be as successfully treated by this process. As to the treatment of oxidized ores other than copper, so far we have not been able to film zinc carbonates at all but lead carbonates are comparatively easy. We have had no success with gas on these, but sodium sulphide gives excellent results. The following figures are the average of 13 different experimental runs made, treating a lead carbonate tailings dump at the Prince Consolidated, Pioche, Nev.: heads, 6.97 per cent.; tails, 1.85 per cent.; concentrates, 38.81 per cent.; ratio, 7.21 to 1; recovery, 77.28 per cent.

CONSOLIDATED NEVADA-UTAH CORPORATION PLANT

Another illustration of the possibilities of a combination of gravity and flotation processes on a complex mixed lead-zinc-iron sulphide ore is the plant of the Consolidated Nevada-Utah Co., at Pioche, Nev. The following description of the ore, together with much of the following data, has been furnished through the kindness of Mr. Van Wagenen, general superintendent.

"The ore is a typical complex sulphide, consisting of pyrite, galena, sphalerite and chalcopyrite in a quartz matrix. The gangue is, of course, quartz, but the vein carries fragments, large and small, of the quartzite hanging wall, cemented intimately by quartz amidst the ore proper. All the sulphides are distinctly crystalline; they are argentiferous in varying degree, the silver occurring both native and as argentite.

Secondary sphalerite is present at certain horizons, and the carbonates are in evidence in increasing quantity upward from water level. By weight, the vein as mined and milled is in the neighborhood of 50 per cent. sulphides. Zinc and silver are the principal values."

This ore was first tested on miniature machines, and afterwards in a 5-ton lot by The General Engineering Co., who designed and built the commercial plant. The plant has been in operation since about April 16, 1916. For the 8 months elapsed since the initial start, the mill has run 87 per cent. of the total time. No material changes in the original flow sheet have been necessary, except the addition of one extra table for re-treating the flotation tails, and one extra flotation cell, so as to practice series treatment. Taking into account the complex mixture of the ore, the results, for a mechanical method, may be considered very satisfactory.

The mill water pumped from the mine has a temperature of from 100 to 120° F., and to find out if this was having any detrimental effect on the results, the mill was sampled at the vital points every $\frac{1}{2}$ hr. for a period of 2 days. A sample of the heads so obtained was sent to The General Engineering Co. A summary of the three results is given in Table 13, comparing the mine water and Salt Lake hydrant water tests on miniature machines (1,000-gram charges) with mill results for the period, also comparing the preliminary miniature tests with the 5-ton test sample. The comparison speaks well for the reliability and value of miniature tests. The 6 per cent. difference in recovery might easily be accounted for by a difference in the samples, especially in the proportion of mine to dump ore in the 5-ton test of April 15 and the ore milled in August, 1916.

All the essentials of the plant will be plain from a study of the flow sheet shown in Fig. 6 without further description. The tonnage averages 53 tons per day. The power required for crushing and conveying material into the mill bin is 10 hp. for 8 hr., and for all other purposes 95 hp. for 24 hr., equivalent to 46 hp.-hr. per ton = 34.5 kw.-hr. per ton.

A screen analysis of the final tailings made in July was as follows:

Mesh	Per Cent.
+ 60	2.6
+100	11.3
+150	20.3
+200	8.5
-200	57.3

Commercial results for the months of September and October, 1916, are given in epitomized form in Table 14.

Flotation Oil Mixture

Two standard Callow cells are used as roughers and one standard size for the re-treatment. All froth is cleaned in one half-size cell. The one

re-treatment cell lowers the tails practically two units. For dropping lead, iron and insoluble, a special oil at one time was used, being made by adding 1 part H_2SO_4 to 2 parts of No. 33 Gravity fuel oil. It has been discontinued. There is a tendency for the solutions to foul and kill the froth. At the first appearance of this condition, the clear thickener overflow is turned into the tail race for about 30 min.; this is found to be more satisfactory and cheaper than the use of chemicals. The oil mixture now in use consists of 70 per cent. G.N.S. No. 28 (coal tar), 10 per cent. G.N.S. No. 8, and 20 per cent. Pensacola No. 80. The coal tar (G.N.S. No. 28) is fed to the Marcy mill. Adding tar to the pebble mill was only of slight value while adding it to the Marcy mill lowered tailings about one and one-half units. A mixture of No. 8 and No. 80 is added to the elevator, elevating flotation feed sufficient for frothing, No. 8 alone having been found best added to the rougher tails to give the desired froth on the second series.

The lead concentrate made originally was too "zincy" to market, so that it is now cut in two, making a lead concentrate and an iron concentrate. The former is marketed and the latter is piled up awaiting further treatment.

The lead concentrate assays approximately:

Au	Ag	Pb	Cu	Zn	Fe	Insol.	S
0.42	60.00	14.00	0.50	7.00	33.00	1.40	40.00

TABLE 14.—*Commercial Zinc Results, Consolidated Utah-Nevada Corporation*

September	Tons	Gold	Silver	Lead	Zinc	Iron	Insoluble
Heads.....	1,567.9		11.5		15.90		
Concentrates.....	385.0	0.135	25.6	1.35	41.66	9.1	15.3
Tails.....			1.5		5.20		
October							
Heads.....	1,575.60		9.00		15.70		
Concentrates.....	495.00	0.10	19.95	1.34	42.45	9.75	11.5
Tails.....			1.10		5.30		

Results thus far in December are much better than any time heretofore. The month, so far, has averaged close to:

	Tons	Gold	Silver	Lead	Zinc	Iron	Insoluble
Heads.....	52 to 53 per day		9.0		18.0		
Concentrates.....	15 per day		20.0		43.0	7.5	9.0
Tails.....			1.0		4.8		
Float concentrates.....					46 to 47		

The iron concentrate assays approximately:

Au	Ag	Pb	Cu	Zn	Fe	Insol.	S
0.24	30.00	4.00	0.50	24.00	33.00	2.50	40.00

About 4 per cent. of the flotation feed consists of this latter material. The zinc and iron are in chemical combination. Floating this would mean an unsalable product, while dropping it means high tails.

Handling of Flotation Concentrates

For thickening the flotation concentrates prior to filter-pressing, three 6 by 14 tanks, fitted with Goldfield agitators, are used. Two would have been sufficient for the tonnage. These are run intermittently, the system being to fill, settle, and decant in rotation. This is the same scheme as that used at the zinc plant of the Magma Copper Co., also in a recent installation at the Bingham and New Haven Copper and Gold Mining Co.

The filtering is done on a 4 by 6-ft. Portland filter, the capacity being sufficient so that the filter operates only about one-third of the time. The blow is set so that it operates just above the scraper; an overflow pipe prevents the pulp level from rising too high. A Gould wet vacuum pump is used, giving an average vacuum of 17 in. The filter cake averages a thickness of about $\frac{3}{4}$ in., moisture seldom being above 8 per cent. Filter concentrates are mixed with table concentrates, which have been drained in the bins, for shipment. Car samples vary from 5 to 9 per cent. in moisture. Mr. Van Wagenen reports that this method of handling flotation concentrates is admirable, and one of the best features of the mill.

When the canvas needs cleaning, a water connection is made to the air pipe, and the canvas is rapidly cleaned simply by brushing.

The following brief summaries of the results from a few other plants, using Callow flotation, may possibly add to the interest of this paper.

WALKER MINING CO.'S PLANT, PORTOLA, CAL.

The important primary minerals are chalcopyrite and bornite, occurring associated in a heavy diorite gangue, containing much magnetite.

Because of this excessive amount of magnetite, water concentration is not practical. The ore is exceptionally well adapted to the flotation process and yields a concentrate practically free from magnetite. Covellite occurs in small quantities, but so far no chalcocite has been found. Some of the ore that occurs near the surface has been oxidized to cuprite and much of the chalcopyrite near this zone has been tarnished, giving it the appearance of bornite, though this is only a thin film covering.

The tonnage treated has averaged 75 tons per day, treated on four roughing cells run in parallel. All rougher concentrates are re-treated on one full-size cleaner. The following table gives the results for September and October, 1916:

Heads.....	3.14 per cent. total copper
Tails.....	0.52 per cent. total copper
Concentrates.....	18.10 per cent. total copper
Recovery.....	86.03 per cent. total copper.
The recovery of the sulphide copper varies from 90 to 93 per cent.	
Ratio.....	6.53 to 1.

The regular oil mixture is coal tar, 63 per cent.; coal-tar creosote, 33 per cent.; Pensacola No. 80 or No. 350, 4 per cent.

Screen Analysis

Mesh	Per Cent.
+ 80	0.0
+100	7.0
+200	20.0
-200	73.0

NEEDLES MINING AND SMELTING CO.'S PLANT, NEEDLES, CAL.

This is a small plant treating the zinc-lead slimes, partly from the wet concentrating plant, and partly from some old slime ponds. Trouble has been experienced from time to time owing to lubricating oils, which were allowed to get into the ponds at the time they were originally impounded. There is no trouble when treating regular mill slimes.

The output is about 40 tons per day, treated in three rougher cells operated in series. All rougher concentrates are re-treated on one half-size cleaner, the cleaner tails being returned to the third rougher cell. The following table gives some random results:

	Averages for October and November, 1916		Mill Slimes		Pond Slimes	
	Pb	Zn	Pb	Zn	Pb	Zn
Heads.....	1.80	8.80	1.3	9.5		
Tails.....	0.90	2.90	0.6	1.7	2.0	3.4
Concentrates.....	6.50	37.20				
Recovery.....		72.60				
Ratio.....	5.85 to 1					

The regular oil mixture is coal-tar creosote, 45 per cent.; fuel oil, 25 per cent.; crude pine oil, 30 per cent. Acid is necessary to the extent of about 25 lb. per ton of feed.

MCKINLEY-DARRAGH-SAVAGE PLANT, COBALT, ONT.

To show the possibilities of flotation on silver ores, a brief description is given of the above plant. Some of the mills in this district, notably the Nipissing, are handling their ores by cyanidation, and in other mills by a combination of concentration and cyanide. Generally speaking, it may be said that flotation will give from 1 to 1½ oz. better tails than cyanidation will, using the same degree of comminution. The principal problem involved in a wider application of flotation to this district is the disposal of the concentrates. The present smelting and freight charges on a 300-oz. concentrate varies from \$20 to \$25 per ton. A great many different methods have been tried for a local treatment for these concentrates. The most encouraging results have been obtained by a chloridizing roast and using the Holt-Dern roasting process. A movement is now on foot among some of the operators to install an experimental plant of this description. It is confidently expected that this will be the ultimate solution for the local treatment of Cobalt flotation and other concentrates, effecting a saving of \$10, and possibly of \$15, per ton from their treatment charges over smelting.

The feed to the flotation plant consists of slimes resulting from mining operations and primary crushing, which are separated in the jigging plant, and mill slimes resulting from crushing in stamps and one pebble mill. The mill treats 200 tons per day, approximately 50 per cent. of which reaches and is treated in the flotation plant; 93 per cent. of the feed to the flotation plant is finer than 200-mesh. The chief economic minerals recovered by flotation are argentite, proustite, pyrrargyrite and metallic silver.

The plant consists of two triple length Callow roughers, operating in parallel and two standard length Callow cleaners, operating in series, the tailing from cleaner No. 1 being re-cleaned in cleaner No. 2.

Table 15 gives a summary of typical results.

TABLE 15.—*Flotation Results at the McKinley-Darragh-Savage Mines, Ltd., Averages by Weeks*

Month	Week Os	Heads Os	Tails Os	Concentrates	Per Cent. Extraction
June.....	2nd	8.04	2.01	310	75.00
	3rd	8.91	1.75	386	80.30
	4th	7.74	1.64	363	78.80
July.....	1st	8.65	1.90	334	78.00
	2nd	8.03	2.10	231	73.80
	3rd	5.86	1.40	172	76.10
August.....	4th	6.10	1.41	233	77.00
	1st	6.20	1.48	309	76.10
	2nd	6.43	1.71	269	73.40
	3rd	5.88	1.66	233	71.70
	4th	6.71	1.53	260	73.20
September.....	1st	7.68	2.55	349	66.80
	2nd	6.55	2.16	272	67.00
	3rd	5.30	1.35	206	74.50
	4th	6.29	1.91	211	69.60
October.....	1st	8.05	2.35	238	70.80
	2nd	10.05	3.74	331	62.78
	3rd	8.03	2.60	279	67.62
	4th	8.07	3.04	177	62.30
November.....	1st	5.64	2.13	309	62.20
	2nd	4.71	1.38	248	70.70
	3rd	4.73	1.53	204	67.60
	4th	4.65	1.70	178	64.00
December.....	1st	5.36	1.60	220	70.70
	2nd	6.92	1.08	217	84.50
	Average..	6.78	1.909	262	71.37

HANDLING FLOTATION CONCENTRATES

Everyone who has used flotation appreciates the difficulties of taking care of the flotation concentrates, and the problems connected with their preparation for the smelters. So far, the area or tank volume necessary for a continuous settling system has never been decided on, from the fact that the area or volume required apparently varies with almost every different concentrate. The requirements for the continuous filter now in everyday use are severe, requiring a pulp of at least 50 per cent. solids and as much thicker than this as is possible. The overflow must be clear, and the accumulation of froth incident to a continuous settling plan must be taken care of. Our experience has not been satisfactory with the continuous plan, and it is for this reason that in all our recent plants, we have been installing the intermittent system. Until shown to the contrary, we think that this offers the best solution, in that with it one has complete control of the necessary density for the filters; there is no danger from losses in the overflow; the froth which accumulates during

the filling of the tank is completely disposed of at each cycle of the operation, and therefore cannot accumulate.

The agitator for stirring the contents of the tank during the discharging period is copied from those used at the Goldfield Consolidated mill.

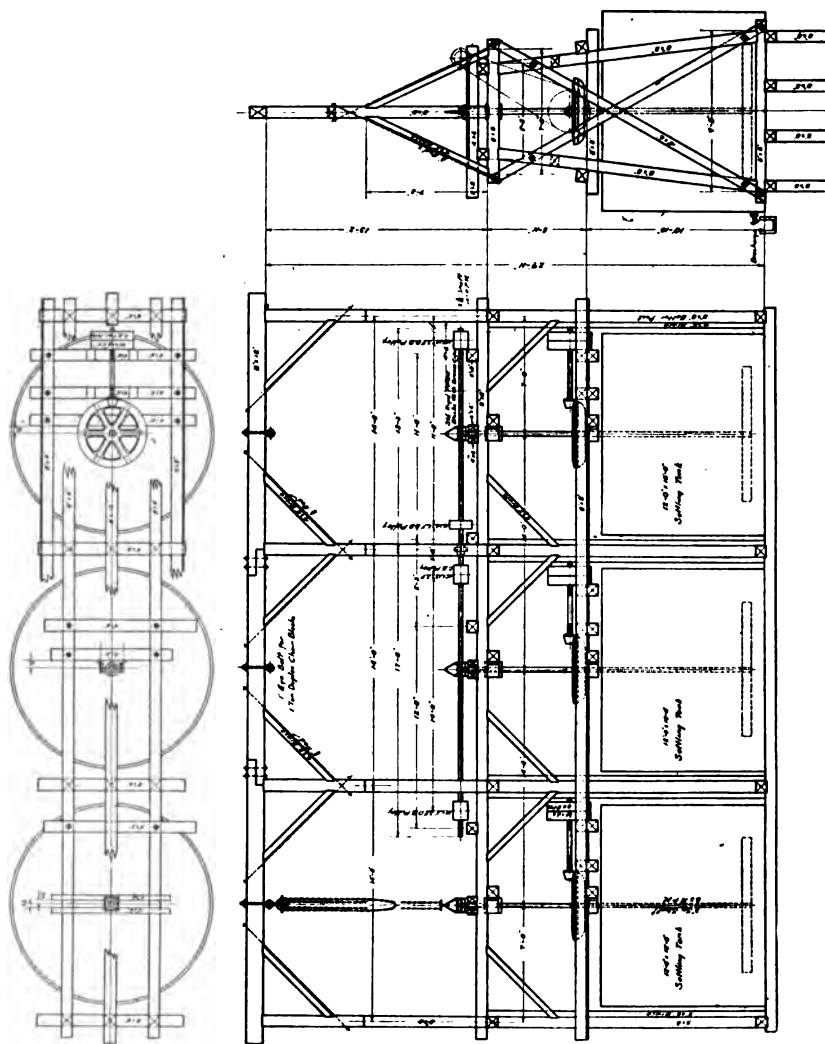


FIG. 7.—AGITATORS FOR 12 X 10-FT. TANKS, 50-TON ZINC PLANT, MAGMA COPPER CO.

As shown in Fig. 7 it consists of arms secured to a square revolving shaft, suspended by a chain block, and passing through a square hole in the driving gear. It is simple, inexpensive, and gives no trouble whatever. The thickened pulp may be drawn off from a central bottom discharge. More recent practice is to draw off through a valve or molasses gate on the side of the tank, or better still, with a diaphragm pump.

CONCLUSIONS FROM LATE PRACTICE

While the fundamental elements employed in flotation are few, yet the practice of the art is not so simple as it looks. It is a process wherein perhaps more than in any other, small causes bring about big results. The number of combinations possible with a list of even one-half dozen oils, and the various acids, alkalies, and salts, the different degrees of dilution and temperature, all of which are dealt with in varying quantities, can best be appreciated by remembering that with four different oils, three oil percentages, two pulp densities and two changes of temperature, the number of possible commutations would be 59,284. This statement, of course, is not to be taken too seriously, or to mean that 60,000 experiments are necessary on any sample of ore, but merely to illustrate by means of an exaggeration that it is evident that flotation experiments take time as well as patience to work out. It cannot be too strongly urged, moreover, that all flotation operations should be in charge of men whose powers of observation have been especially trained, and who not only have had the necessary training with the process in all its various phases, but who also have a practical working knowledge of the machines in operation. I especially emphasize this point because the only disappointments that we have ever suffered in our various installations have been where this advice has not been followed. This is not peculiar to flotation but is an axiom applicable to all technical operations, but one which is too often overlooked to the detriment and loss of all parties concerned.

In conclusion acknowledgment of obligations is made to Mr. Browning, Mr. Van Wagenen and the other managers, who have so kindly furnished much of the information used herein. Special acknowledgment is made of the help received from J. W. Thompson and of his resourcefulness and ingenuity in developing our present filming system, and also to my other able assistants in other flotation operations.



TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Effect of Time in Reheating Hardened Steel below the Critical Range

BY CARLE R. HAYWARD,* S. B., AND S. S. RAYMOND,† M. S., CAMBRIDGE, MASS.

(New York Meeting, February, 1917)

IN reheating quenched steel to remove part of the hardness, the softening effect has generally been considered to be a function of temperature and time. The temperature effect is well known, and long before pyrometers were heard of the blacksmith was able to do a good job of tempering, by rule-of-thumb methods and experience in judging temperatures.

Modern conditions, which demand steel that will withstand the severest tests, require that the heat treatment be carried out with every possible precaution to secure the best results obtainable. Pyrometers and heat-treatment furnaces of many types are on the market and improvements are continually being made so that it is now possible to regulate the temperature in heat treatment very accurately. There is still a question regarding the time that the steel should be subjected to treatment. It is customary with all operators to insure the desired temperature throughout the specimen and it is generally supposed that a longer treatment of a hardened steel produces a further softening but there are few published figures showing the exact effect of time of treatment at constant temperature.

A search of the literature disclosed several brief statements that the tempering effect was a function of both time and temperature, but in no case were any figures given. The opinion is very generally held that the well-known tempering colors are accurate indicators of the degree of tempering. This is disputed by Brearley (*The Heat Treatment of Tool Steel*, page 102) who states:

It has been said that the result would be the same in respect to both hardness and other properties whether the colors were obtained by a shorter heating at a higher temperature or a longer heating at a lower one. This, to say the least, is a very doubtful conclusion, and is certainly not borne out by mechanical tests on oil-hardened and tempered motor car steels, which, after tempering for periods varying from 15 min. to 2 hr., show no very great differences.

The present investigation had for its object the obtaining of some definite data regarding the effect of time, which might serve as a guide to those engaged in the heat treatment of steel.

The Rhode Island Tool Co. coöperated by furnishing the steel and machining the test specimens.

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STEEL USED

The steel furnished came from three rolled bars $1\frac{3}{16}$ in. in diameter. The analyses are given in Table 1.

TABLE 1

Mark	C	Si	S	Mn	P
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
C	0.45	0.03	0.049	0.056	0.016
D	0.45	0.03	0.048	0.053	0.017
F	0.44	0.03	0.049	0.053	0.013

HEAT TREATMENT

Outline of Operations

The procedure decided upon was to heat all specimens to 800° C., quench at this temperature, and reheat them in sets of three for 15 min., 30 min., 1 hr., 2 hr. and 4 hr. at 300° C., 400° C., 500° C. and 600° C.

Preparation of Specimens

The bars were sawed into $5\frac{1}{2}$ -in. lengths and marked as follows: 301, 302, 303, 304, 305, 401, 402, etc., corresponding to the five different periods of heating at each temperature. In each case there were three specimens with the same number. Three specimens marked 800 were not reheated after quenching. Before treatment, the ends of these were threaded to fit the testing machine, as they would be too hard to thread after treatment.

Furnace

The furnace used was made in the metallurgical laboratory of the Massachusetts Institute of Technology and has been described and illustrated in a previous paper.¹ The heating chamber is an alundum muffle 16 in. long, 9 in. wide and 2 in. high, wound with No. 15 excello resistance wire. The muffle is surrounded by 2 in. of powdered magnesia inclosed in an outside case of galvanized iron. The specimens were supported midway between the top and bottom of the muffle by an asbestos rack. Temperatures were measured by a platinum platinum-rhodium thermocouple connected to a Siemens & Halske recording galvanometer.

Heat Treatment

The procedure in heat treatment was as follows: The furnace was heated to 800° C. and nine specimens were introduced, which caused the

¹ Carle R. Hayward: The Effect of Sulphur on Low-carbon Steel, *Bulletin No. 118*, p. 1841 (October, 1916).

temperature to fall to about 500°. When the temperature had again reached 800°, which took about 40 min., the current was regulated to hold the heat uniform for 5 min., after which the specimens were withdrawn and quenched in water.

To insure uniformity in quenching, each specimen was dropped into a separate pail of tap water, the temperature of which was 4° C.

The above procedure was continued until all the specimens had been hardened.

In the reheating operations, the furnace was brought to the desired temperature and nine specimens were introduced. This caused a fall in temperature and about 40 min. was required to bring it back to the desired point. When this was reached it was maintained for 1 hr., three specimens being removed after 15 min., three after 30 min. and the remaining three at the end of the hour. Six more specimens were then introduced and after the furnace had regained the desired temperature the heat was kept uniform for 4 hr., three specimens being withdrawn after 2 hr. and the remaining three at the end of the 4-hr. period. When the specimens were removed, each was dropped into a pail of water at 4° C.

The above procedure was carried out for 300°, 400°, 500° and 600°.

Mechanical Testing

After a $\frac{1}{2}$ -in. length had been sawed from the end of each piece for microscopic examination, the heat-treated specimens were turned into

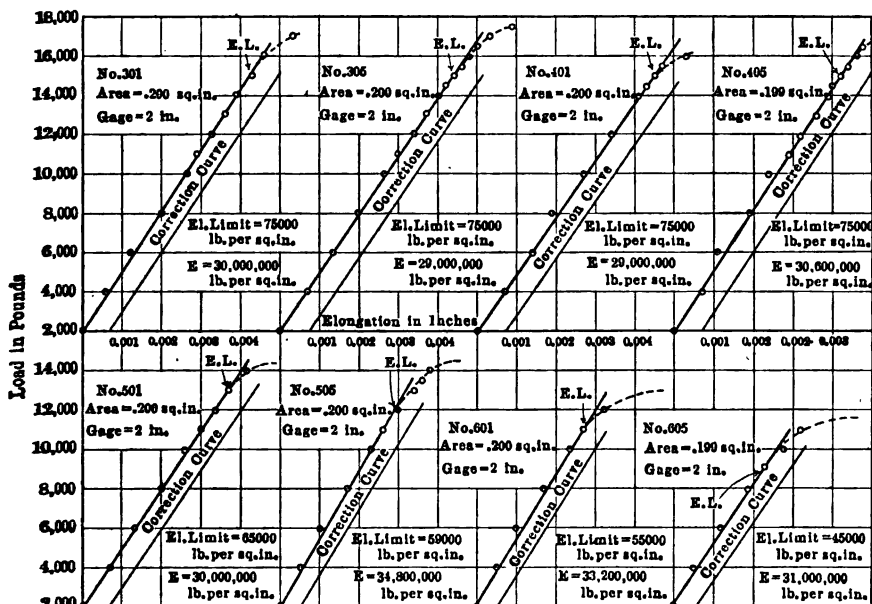


FIG. 1.—DIAGRAM SHOWING THE ELASTIC LIMIT.

standard tensile specimens 0.505 in. in diameter and 2-in. gage length with threaded ends.

The tensile tests were made on an Olsen machine in the Mechanical Testing Laboratory of the Massachusetts Institute of Technology. A Berry strain gage was used for determining the elongation under increasing load, but because of the time required and the large number of specimens to be tested this instrument was used only on one specimen heated for 15 min. and one for 4 hr. at each temperature. In the remainder of the specimens the yield point was determined by the drop of the balance arm.

Table 2 gives the elongations recorded by the Berry strain gage for the different loads applied. Specimens 301 and 305 are those heated to 300° C. for 15 min. and 4 hr. respectively; 401 and 405 those heated to 400° for 15 min. and 4 hr. respectively. Similarly, the 501 and 505 specimens are the 500° heats and 601 and 605 the 600° heats.

The results in Table 2 are plotted in the curves shown in Fig. 1.

TABLE 2.—*Results of the Elastic-Limit Tests*

Load in Pounds	Elongation in Inches							
	301	305	401	405	501	505	601	605
2,000	0	0	0	0	0	0	0	0
4,000	0.00060	0.00070	0.0007	0.0007	0.0007	0.0005	0.0005	0.0005
6,000	0.00120	0.00134	0.0014	0.0011	0.0013	0.0010	0.0010	0.0012
8,000	0.00200	0.00200	0.0019	0.0019	0.0020	0.0017	0.0017	0.0019
10,000	0.00264	0.00264	0.0027	0.0024	0.0026	0.0023	0.0023	0.0028
11,000	0.00290	0.00300		0.0029	0.0030	0.0026	0.0027	0.0032
12,000	0.00328	0.00340	0.0034	0.0032	0.0034	0.0030	0.0032	0.0140
13,000	0.00360	0.00370		0.0036	0.0037	0.0034	0.0178	
14,000	0.00390	0.00400	0.0041	0.0039	0.0042	0.0038		
14,500		0.00420	0.0043	0.0040	0.0173	0.0226		
15,000	0.00430	0.00440	0.0045	0.0042				
15,500		0.00460	0.0047	0.0044				
16,000	0.00460	0.00480	0.0053	0.0046				
16,500		0.00500		0.0048				
17,000	0.00530	0.00530		0.0222				
17,500		0.00590						

The figures obtained for the yield point, breaking load, elongation and reduction of area for all the specimens are given in Table 3. The first figure in each number denotes the temperature of reheating, 300°, 400°, 500° and 600°, while the last figure represents the period of heating: 1 = 15 min., 2 = 30 min., 3 = 1 hr., 4 = 2 hr. and 5 = 4 hr. The specimens numbered 800 were quenched at 800° and not reheated. Those marked 000 are from the original untreated bars.

The average results in Table 3 are plotted in Fig. 2.

TABLE 3.—Results of Tensile Tests

No.	Load, Pounds per Square Inch		Elonga- tion of Gage, Per Cent.	Reduc- tion of Area, Per Cent.	No.	Load, Pounds per Square Inch		Elonga- tion of Gage, Per Cent.	Reduc- tion of Area, Per Cent.
	Yield Point	Ultimate				Yield Point	Ultimate		
301	100,400	128,900	18.5	57.8	501	79,140	110,000	22.0	59.3
301	92,500	126,400	20.0	56.0	501	79,470	115,600	24.0	60.2
301	92,500	128,350	20.0	55.5	501	76,500	116,600	22.5	60.5
Av...	92,500	127,900	19.5	56.4	Av...	78,370	114,100	22.8	60.0
302	93,700	130,000	19.5	55.6	502	76,850	111,500	22.5	60.5
302	92,500	126,500	18.5	55.5	502	78,640	110,500	20.5	60.3
302	91,200	117,300	18.0	54.8	502	76,300	112,500	22.0	61.2
Av...	92,470	124,600	18.7	55.3	Av...	77,260	111,500	23.0	60.7
303	91,000	126,500	19.5	58.0	503	71,600	108,500	24.0	61.0
303	92,250	116,800	19.0	56.3	503	78,040	117,600	22.5	61.3
303	93,800	128,200	19.0	55.3	503	77,375	108,300	22.0	60.2
Av...	92,350	123,800	19.2	56.5	Av...	75,670	111,500	22.7	60.8
304	91,000	128,000	24.0	55.5	504	75,250	118,700	23.0	60.6
304	91,750	110,800	18.5	58.3	504	75,000	105,200	24.5	61.2
304	92,500	128,000	20.5	59.8	504	75,100	106,800	23.0	59.9
Av...	91,750	122,200	21.0	57.9	Av...	75,120	110,200	23.5	60.6
305	95,250	127,500	20.5	55.5	505	79,750	107,400	24.0	60.3
305	94,200	123,800	19.5	58.8	505	77,490	106,000	23.5	60.3
305		127,500	20.5	56.5	505	80,000	102,400	24.0	60.8
Av...	94,720	126,200	20.0	56.9	Av...	79,080	105,300	23.8	60.5
401	89,400	118,250	21.5	60.0	601	65,177	96,000	26.5	64.8
401	85,700	118,500	21.5	59.8	601	75,205	96,900	24.5	59.4
401	87,500	116,500	21.0	58.5	601	80,000	95,000	25.0	61.0
Av...	87,510	117,800	21.3	59.4	Av...	70,190	96,000	25.3	61.7
402	87,500	117,000	22.0	59.5	602	70,203	95,400	25.0	63.2
402	88,800	117,500	20.5	60.2	602	69,000	95,350	26.5	64.0
402	85,800	117,400	22.0	60.2	602	69,350	95,750	24.5	63.3
Av...	87,370	117,300	21.5	60.0	Av...	69,520	95,500	25.3	63.5
403	86,910	117,000	20.0	60.0	603	65,850	89,750	27.0	63.5
403	87,700	118,100	20.5	58.3	603	69,106	91,600	25.0	64.8
403	85,700	116,100	22.0	59.8	603	68,180	90,300	26.0	63.6
Av...	86,770	117,100	20.8	59.4	Av...	67,710	90,600	26.0	64.0
404	84,000	116,700	21.0	58.5	604	66,850	88,500	28.0	66.5
404	85,200	117,000	20.5	58.5	604	68,350	90,100	26.5	63.8
404	91,200	117,200	21.0	59.4	604	66,600	90,450	27.0	63.8
Av...	84,600	117,000	20.7	58.8	Av...	67,260	89,680	27.2	64.7
405	82,500	116,000	21.0	60.2	605	68,000	89,850	26.5	64.5
405	83,000	128,200	21.0	59.8	605	65,800	90,450	27.0	64.8
405	85,400	116,600	22.0	58.8	605	67,300	89,200	27.5	65.8
Av...	83,630	116,300	21.3	59.6	Av...	67,030	89,830	27.0	66.0
000	55,000	76,400	31.0	51.8	800	91,000	122,200	17.0	50.3
000	49,750	76,300	31.0	53.3	800		129,500	18.0	48.7
000	52,250	76,800	31.0	52.8	800	101,300	129,000	17.0	49.7
Av...	52,330	76,500	31.0	52.6	Av...	96,150	126,900	17.3	49.6

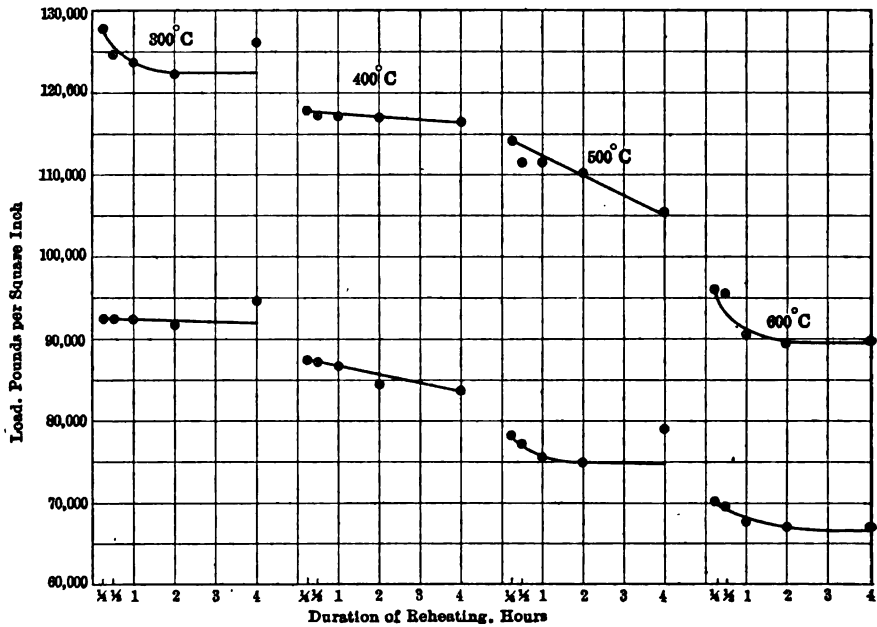


FIG. 2.—UPPER CURVES SHOW BREAKING LOAD. LOWER CURVES SHOW YIELD POINT.

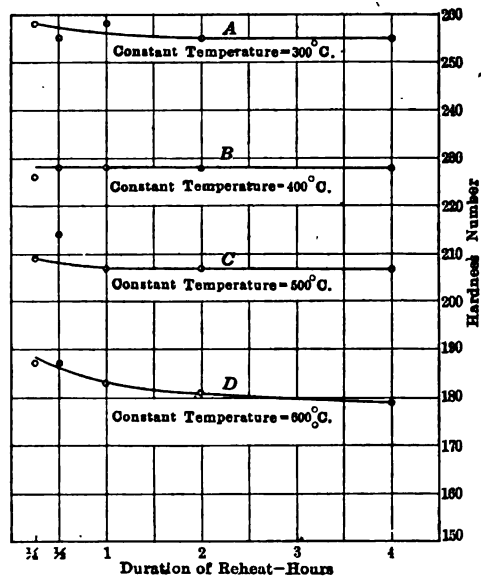


FIG. 3.

Microscopic Examination

A piece sawed from the end of each specimen was polished and examined under the microscope. No difference could be detected between the structure of the 15-min. and 4-hr. specimens for any given temperature. A gradual change was noticed as the temperature increased from 300° to 600°, as would naturally be expected. Because of the uniformity of the photographs, they are not included in the paper.

Brinell Hardness Tests

After completing the microscopic examination, two of the polished specimens from each set were tested in an Alpha Brinell machine. The results are given in Table 4 and the averages are graphically represented by the curves in Fig. 3.

TABLE 4.—*Results of Hardness Tests*

Specimens No.	Diameter of Ball Impression (Mm.)	Hardness No.	Specimens No.	Diameter of Ball Impression (Mm.)	Hardness No.
301	3.75	262	501	4.15	212
301	3.78	255	501	4.20	207
		258			209
302	3.80	255	502	4.10	217
302	3.80	255	502	4.15	212
		255			214
303	3.75	262	503	4.20	207
303	3.80	255	503	4.20	207
		258			207
304	3.80	255	504	4.20	207
304	3.80	255	504	4.20	207
		255			207
305	3.80	255	505	4.20	207
305	3.80	255	505	4.20	207
		255			207
401	4.05	223	601	4.40	187
401	4.00	228	601	4.40	187
		226			187
402	4.00	228	602	4.40	187
402	4.00	228	602	4.40	187
		228			187
403	4.00	228	603	4.45	183
403	4.00	228	603	4.45	183
		228			183
404	4.00	228	604	4.45	183
404	4.00	228	604	4.50	179
		228			181
405	4.00	228	605	4.50	179
405	4.00	228	605	4.50	179
		228			179
000	5.15	134	800		

Discussion of Results

The elastic limit as determined by the Berry gage is identical for the 15-min. and 4-hr. treatments at 300° and 400°. At 500° the elastic limit of the 4-hr. specimen is about 10 per cent. lower than the 15-min. specimen, and at 600° the falling off is about 18 per cent.

The curves showing the breaking load and yield point are somewhat erratic. For some reason the breaking load in the 300° specimens fell slightly up to 2 hr. and rose again after 4 hr. Since the yield point remained constant, it is probable that further tests would have shown

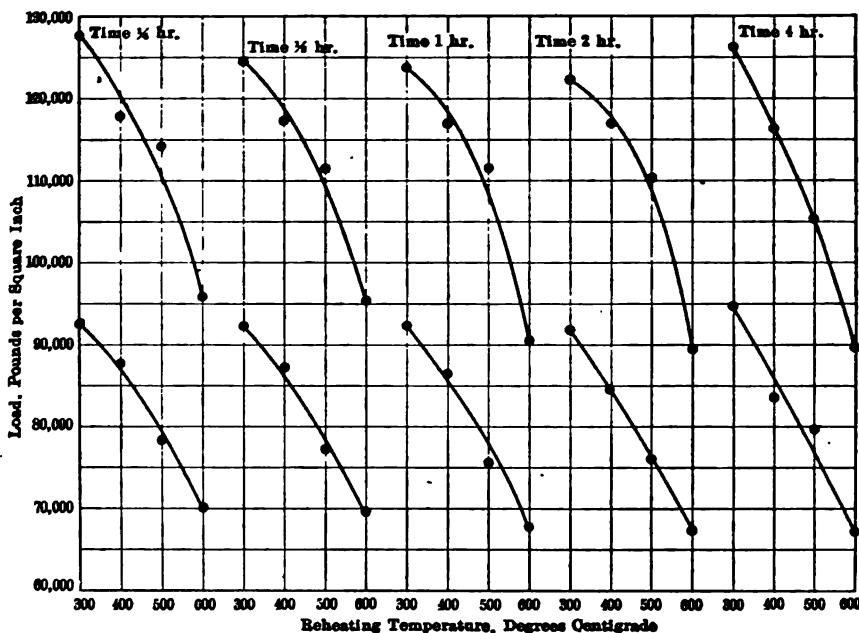


FIG. 4.—UPPER CURVES SHOW BREAKING LOAD. LOWER CURVES SHOW YIELD POINT.

that actually there is no falling off in tensile strength with increasing time of treatment. This would accord with the results obtained with the Berry gage for elastic limit.

In the 400° specimens the ultimate strength remained constant while the yield point fell off about 5 per cent. as the time increased from 15 min. to 4 hr.

In the 500° specimens there was a drop of about 8 per cent. in the ultimate load as the time increased from 15 min. to 4 hr. The yield point fell about 5 per cent. in 2 hr. and then for some unexplained reason rose again to about the original figure.

The 600° specimens showed a gradual falling off in both ultimate load

and yield point as the time increased. The total fall in the first case was about 6 per cent. and in the second about 3 per cent.

It is not fair to consider the question settled when only three specimens were used for each treatment, but the results obtained indicate that for reheating quenched medium carbon steel to temperatures below 500° it is only necessary to heat it through. Longer heating has little or no effect on the tensile strength, ductility or hardness. At temperatures above 500° increasing the time of treatment causes a slight falling off in hardness and tensile strength with a corresponding increase in ductility.

Although not intended as a part of this investigation, the curves in Fig. 4, plotted from the data of Table 3, are of passing interest. They show that for the 15-min. heating the ultimate strength falls about 25 per cent. and the yield point about 20 per cent. as the temperature increases from 300° to 600°, and the same figures hold approximately true for the $\frac{1}{2}$ -hr., 1-hr. and 2-hr. treatments. Heating for 4 hr. causes the ultimate strength to fall about 30 per cent. and the yield point about 25 per cent. as the temperature increases from 300° to 600°.

It is unfortunate that the heat capacity of the furnace used was so small that the introduction of the specimens caused such a large drop in temperature and a relatively long time in bringing it back to the desired point. However, in view of the fact that time was found to be of so little importance, it is not probable that the results were influenced appreciably by the time the specimens were in the furnace below the desired temperatures.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Need and Advantages of a National Bureau of Well-Log Statistics

BY W. G. MATTESON,* HOUSTON, TEX.

(New York Meeting, February, 1917)

In 1915, the State of California passed a law of great scope and importance. This law has been in successful operation for a year and may be briefly described as an act "establishing and creating a department of the State Mining Bureau for the protection of the natural resources of petroleum and gas from waste and destruction through improper operations in production; providing for the inspection of petroleum and gas wells; requiring all persons operating petroleum and gas wells to make certain reports" and providing the necessary officers and assessments to carry out the provisions of the act.

The statute provides for a separate department of the State Mining Bureau, under the general supervision of the State mineralogist, with a staff composed of a "state oil and gas supervisor" and four deputies.

It rules that all petroleum and natural gas wells shall be drilled, operated or abandoned under the supervision of the State supervisor, who shall order such tests or remedial work as seems to him necessary to protect the interests of property owners and the general public.

It also provides for a board of arbitration to consider orders given by the supervisor which are not acceptable to the well owner.

Section 18 of the act, which is the provision that is directly related to this paper, requires "the owner of any well referred to in this act to keep a careful and accurate log of the drilling of such well," and further requires that this log shall show the character and depth of the formations passed through or encountered in the drilling, particularly the location and depth of the water-bearing strata; the character of the water encountered (so far as ascertained); whether the water was shut off, and, if so, at what point; that it shall show completely the amounts, kinds and size of casing used; the depth at which oil-bearing strata are encountered; the depth and character of the strata, and whether all water was successfully and permanently shut off so as to prevent percolation or penetration into the oil-bearing strata.

This log is to be kept in the local office of the owner or operator, subject to the inspection of the supervisor or any of his deputies at any

* Geologist and Mining Engineer.

time during business hours, and a copy is to be filed with the deputy supervisor immediately upon the completion of the well, or upon the completion of any additional work on the well.

The owner of any well drilled previously to the enactment of this act must furnish to the supervisor a complete and correct log of the well, so far as may be possible, together with a statement of the present condition of the well.

It is evident from the foregoing that the main object of the act is to extend the life and increase the productivity of oil fields by preventing the injudicious drilling of wells without taking the proper precautions to prevent encroachment of salt water in the oil- or gas-bearing sand, and to shut off overlying and underlying water sands which might flood the oil-bearing strata. In a field where many operators are drilling quickly to obtain first production, the necessity of taking adequate measures against the encroachment and infiltration of water is fully understood. Yet the act, as outlined, is capable of even much broader application. The ideas, included or suggested, are so pertinent and important that similar statutes in all States having oil-producing properties or prospects should be strongly urged. For this and other reasons which follow, the enactment of a Federal law, similar in design to the California statute and supplemented by the establishment of a National Bureau of Well-log Statistics, seems advisable.

No one realizes the urgent need for such a law more than the geologist, who, in his frequent examination of territory where outcrops are scarce, scattered or nil, often discovers that logs of wells, drilled several years previous, were not taken, or are not available, or that such logs as are obtainable are almost useless because of the careless and unsystematic manner in which they were recorded. Hundreds of such monuments to inefficiency may be found in our oil-producing States today.

The problem, moreover, bears a close relationship to the theme of conservation. Various experts have calculated that with current consumption and production, our present oil fields will be exhausted in from 25 to 50 years. If America is to maintain her industrial supremacy, this exhaustion of our present oil supply must be offset by the discovery and development of new fields. The larger companies have foreseen this contingency, and, acting on their convictions, are organizing efficient geological corps and spending hundreds of thousands of dollars annually in an attempt to open up new territory. From an economic and industrial viewpoint, their efforts demand the strongest coördination on the part of State and Federal government. A good percentage of the territory now under private investigation has been marked by the work of the wildcatter, and were accurate logs of such test wells at hand, probably much of such territory would be condemned at once, permitting the time, labor and money to be devoted to the investigation of

some other region. Conservation of private capital devoted to the exploitation of natural resources must be given serious consideration as well as the conservation of the natural resources in question.

Since our recent progress in drilling methods and our increased understanding of the many peculiarities of oil and gas horizons, the geologist can no longer be satisfied with the statement that any wildcat test was a dry hole. He must ascertain which of the many factors might account for the dry hole in question and often only careful examinations of well logs will convey this information. In some districts where outcrops are few and scattered or covered by soil and detritus, the geologist is often forced to rely entirely on well logs in working out structural relationships. The Gulf Coast Tertiary fields, characterized largely by palustrine and marine deposits of cross-bedded, unconsolidated, often unstratified sands, clays, sandy clays and clayey sands, furnish an excellent illustration. Hundreds of wildcat wells are being drilled today by independent operators in Texas, Louisiana and Mississippi. In many instances well logs constitute practically the only data by which the structure of such virgin territory can be worked out, yet such data are either neglected or taken in such an inconsistent manner as to be of little value. It was during the examination of such territory that the potential possibilities of some such law and organization as is suggested in this article was forced upon the attention of the writer.

Although a State has taken the initiative in this important matter there are many reasons which favor national legislation and the establishment of such a bureau as part of the government organization, preferably as an adjunct to the U. S. Geological Survey. Doubtless there will be a conflict of opinion in this regard, but it is evident that State legislation is contingent on too many probabilities such as political favor, lack of funds, etc. State legislation is apt to be vacillating, delayed, and lack uniformity. A national law would insure standard methods of procedure, would become operative in all States simultaneously, and the question of funds should not prove a serious handicap. The U. S. Geological Survey possesses a statistical department unsurpassed in its ability to handle such work, while government geologists could render a much-needed service by devising a standard form and method of taking and recording logs. The importance of uniformity can not be over-emphasized.

The information accruing from such a bureau is as vital and valuable to the government as to the oil producer, since well logs are generally sought by the government geologists wherever possible in working out the structure of the various quadrangles comprised in the geologic folios being annually issued, and often such logs constitute the only positive information obtainable. All these considerations are strong arguments for government supervision and legislation and the importance of obtaining such data in intelligent form becomes more evident.

Of course, it might be objected by some operators that logs which were obtained by private investment are valuable to the original investors and should not be open to public scrutiny. Such an objection is often legitimate, but there are many ways of overcoming it, such as specifying that the logs will not be available for public inspection so long as the operator in question holds valuable options on said property.

Obviously the details of such a project as the establishment of a National Bureau of Well-log Statistics must command careful thought and development. The California law suggests an outline into which may be incorporated numerous, far-reaching provisions. Doubtless the California statute will be altered from time to time as experience gained from operation points to the advisability of new amendments. The author has merely attempted to give some definiteness to an idea which he believes to be sound, practical and worthy of serious consideration. He trusts that it may promote such discussion and suggestion as will eventually result in some constructive action embodying the principle which has been defined.

Recent Geologic Developments on the Mesabi Iron Range, Minnesota

Discussion of the paper of J. F. WOLFF, presented at the New York meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1763-1787.

J. G. WOLFF, Duluth, Minn. (communication to the Secretary*).—In raising the question ("Newly Discovered Fossil Remains in the Cretaceous Shale," p. 1784) as to the possibility of the so-called Cretaceous remnants found on top of Mesabi Range orebodies really being Jurassic, the writer was acting under a wrong impression. In speaking of the matter with different geologists of note who have visited Northern Minnesota during the past year, he understood one of them to refer to Jurassic shales existing quite extensively in Northwestern Minnesota. This apparently was entirely a misunderstanding. In answer to an inquiry in regard to this matter, Prof. W. H. Twenhofel, paleontologist at the University of Wisconsin, said that the known Jurassic nearest to Northern Minnesota is in Western Dakota. From the reproductions of fossils shown on p. 1785, Prof. Twenhofel tentatively correlated them as "middle-upper Cretaceous."

* Received Jan. 15, 1917.



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Temperature Measurements in Bessemer and Open-Hearth Practice*

BY GEORGE K. BURGESS,† SC. D., WASHINGTON, D. C.

(New York Meeting, February, 1917)

I. INTRODUCTION

THE suggestion has often been made that it would be highly desirable, at least for certain grades of steel, to be able to control more certainly, by pyrometric measurement or otherwise, the temperatures of the operation of the open-hearth and other furnaces used in the manufacture of steel and iron.

That the properties of the ingot and finally of the finished steel product are intimately related to the final temperatures attained by the metal in the furnace and to the temperature of casting, has been recognized by metallurgists for a long time, and this question has been very thoroughly treated, especially by Prof. Henry M. Howe, and by A. W. and H. Brearley.¹

It is the object of this paper to demonstrate that in so far as casting temperatures of furnaces, steel ingots, and similar operations involving the temperatures of streams of iron and steel are concerned, well-known pyrometric methods may be easily applied—if certain recently determined corrections are made and precautions taken—with a relatively high degree of accuracy. Greater but not insurmountable difficulties will be encountered in the case of open-hearth furnace temperatures, while for those of the converter type a ready solution does not seem practicable.

It appears that certain essential data have always been lacking in reporting observations such as those of Le Chatelier in 1892. Thus the correction to be applied for the characteristic radiation—or emissivity—has hitherto not been adequately taken into account nor the limitations of the several possible pyrometric methods clearly recognized.

In view of the fact that, in open-hearth practice the temperature to be measured in the furnace itself, in the stream of metal and slag and in

* Abstract prepared by the Editorial Department from the manuscript of a Technologic Paper to be published by the U. S. Bureau of Standards.

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¹ A. W. and H. Brearley: *Some Properties of Ingots, Iron and Steel Institute of Great Britain*, Sept., 1916.

the teeming into molds or ingots is usually above $1,500^{\circ}\text{C}$. ($2,700^{\circ}\text{F}$.) and may reach to above $1,750^{\circ}\text{C}$. ($3,200^{\circ}\text{F}$.), it is manifest that we are limited to the optical and radiation types of pyrometer. Preference will be given to the former for the reason that the errors in the use of the radiation pyrometer caused by intervening gases, distance, size and specific radiation (emissivity) of objects sighted upon are greater and more uncertain than with the several forms of optical pyrometer² using monochromatic light.

II. FURNACE TEMPERATURES

Although one may arrange to observe temperatures of any portion of the open-hearth furnace, and also of the slag surface, what one is mainly interested in is the temperature of the metal, which bears but a remote relation to that of the roof and walls on which the heat of the fuel impinges.

Pyrometric control of temperatures of the roof is easy and may be exact and, although somewhat misleading, may also be made without serious interference from gases and smoke by sighting through suitably placed ports. With a knowledge of the melting range of the roof and arches, the wasting away of the bricks by overheating may be retarded by pyrometric control. Observation of temperatures within the furnace are simpler than of streams of metal or slag for no correction of any importance has to be applied to the optical pyrometer readings for selective radiation (see Sec. VI).

The temperature conditions in the bath of the open-hearth furnace may perhaps be most easily determined by sighting an optical pyrometer through a peep hole in a door or through a port upon the surface of slag, preferably near the center of the bath. This temperature is not usually that of the metal but does not, in general, appear to be far removed from the metal temperature especially as the time of tapping the furnace approaches and the fuel supply has been fairly regular and uniform.

The temperature of the metal itself may be determined—but not as readily and surely as one could wish—by removing with a preheated spoon some 50 lb. and sighting upon the surface of the metal contained in the spoon with an optical pyrometer, taking a series of temperature-time readings and noting exactly the instant of withdrawal of metal and of temperature readings, from which data the temperature of the metal in the furnace may be estimated. Here complications arise, such as the slag residue and the formation of oxides on the surface, their solidification sometimes before the steel, and other surface effects dependent upon the condition of the steel, escape of gases, etc., with attendant uncertainties

² Burgess and Foote: Characteristics of Radiation Pyrometers, *Scientific Paper No. 250, U. S. Bureau of Standards*. Waidner and Burgess: Optical Pyrometry, *Scientific Paper No. 11, U. S. Bureau of Standards*.

in temperature readings, the speed required in taking readings and other difficulties of rapid manipulation and operations, for which at least three persons are required, whereas one observer is sufficient in taking temperatures of furnace and slag.

Illustrations of all these methods of estimating furnace, metal, slag, and stream temperatures are given later on.

At present, no pyrometric method seems adapted to give satisfactorily the temperature of the metal in the Bessemer converter. The apparent temperature of the flames may of course be observed but this has yet to be shown to bear any definite relation to the metal temperature. There are fortunately other methods of recognizing the status of the reactions and the moment when a Bessemer heat is ready to cast.

III. FURNACE TAPPING TEMPERATURES

If one can get a side view and within 50 ft. of the smoke free side of the tapping stream of an open-hearth furnace or converter, accurate observations may easily be taken of its temperature. Even when sighting upon the stream from a distance good observations may be obtained. The observed temperatures here must be corrected for the emissivity or specific radiation of the metal or slag, a correction depending on the radiation which is characteristic of the stream surface, and upon the colored light used in the pyrometer (see Sec. VI).

The greater brightness of the slag, readily noticed with the unaided eye, is caused by its higher emissivity and, also, in part, sometimes to the fact that it may be slightly hotter than the metal. Slag, however, is not a material of uniform composition nor is its brightness always the same from one cast to another at the same temperature. It follows that pyrometric estimations of slag-stream temperatures may be attended with considerable uncertainty.

IV. TEEMING TEMPERATURES

Here we have almost ideal conditions for obtaining accurately the temperature of the stream, which is clean metal with a slight evanescent surface of oxide giving to the stream, viewed through red glass, a characteristic transparent appearance, the color of the metal being greenish and of the oxide yellow. The corrections to be applied to the temperatures for the emissivity of iron viewed in red light have been exactly determined.²

V. LIMITATIONS AS TO PYROMETER STATION

The arrangement of the open-hearth or Bessemer mills is a considerable factor as to convenience with which temperature observations of

² G. K. Burgess and R. G. Waltenberg: The Emissivity of Metals and Oxides, II. Measurements with the Micropyrometer, *Scientific Paper No. 242, U. S. Bureau of Standards*.

furnace, tapping and teeming may be taken. Usually it is possible, however, to bring the pyrometer to within 10 ft. of the stream when teeming ingots; but distant observations—even to 100 ft. or more—may be taken with but a slight increase in uncertainty. These difficulties of pyrometer stations can usually be readily overcome. It is evidently necessary to have a portable apparatus that may be moved about while taking observations; and one that may be sighted from a considerable distance on a stream issuing from a 2-in. or 3-in. nozzle.

VI. PYROMETRIC METHOD AND EMISSIVITY CORRECTIONS

The observations here described were taken with the Holborn-Kurlbaum form of Morse pyrometer which has been frequently described;⁴ although, of course, other types of optical pyrometer might have been used. The outfit is portable and readily adjusted and permits taking temperature readings as fast as they can be recorded. The instrument is calibrated so as to give correct temperatures when sighting into a clear, closed furnace at uniform temperature, which approaches the theoretical condition of a "black-body," which is said to have an emissivity equal to unity.

When sighted on an incandescent, free surface having a specific radiation or emissivity e , always less than 1, its true temperature Centigrade $t(=T-273)$ may be computed from its apparent (too low) temperature $s(=S-273)$ by means of the formula

$$\log_{10} e = \frac{c \log E}{\lambda} \left(\frac{1}{T} - \frac{1}{S} \right)$$

where $c = 14,500$, E the Napierian base, and λ the wave-length of light used in the optical pyrometer; here $\lambda = 0.65\mu$. This type of formula holds for any optical pyrometer using monochromatic light. The accuracy attainable in practice by this method is about 5°C. at $1,500^\circ\text{C.}$

The emissivity (e) of a smooth surface of liquid iron free from oxide is 0.37 (for $\lambda = 0.65\mu$) and no appreciable difference appears to exist between the emissivities of pure iron and of the steels containing considerable percentages of carbon, nickel or manganese, nor is there any appreciable variation of emissivity with temperature.⁵ A solid surface of incandescent iron—which has the same emissivity as the liquid—cannot be maintained free of oxide in practice; it is always the oxide which is observed. Liquid iron oxide has a somewhat higher emissivity, 0.53, than liquid iron and a very much less value than the solid oxide the emissivity of which is about 0.92. Liquid slag has a variable emissivity of uncertain limits depending on composition but probably usually ranging

⁴ *Scientific Paper No. 11, U. S. Bureau of Standards and Technologic Paper No. 38.* Burgess and LeChatelier: *Measurement of High Temperatures*, 1912 Ed.

⁵ See *Scientific Paper No. 242, Bureau of Standards*, cited above.

between 0.55 and 0.75, and 0.65 is apparently the usual value of e for "dark" slag. For reasons noted in Sec. IV, the emissivity of the metal stream of steel is taken as 0.40 instead of 0.37 in computing true temperatures in Sec. VII.

VII. THE OBSERVATIONS

In the following tables are given certain typical series of observations—taken from among many obtained in several large steel plants during the past few years—of furnace and stream temperatures, including all the cases mentioned above. The attempt has not been made here to correlate the ingot or heat characteristics with the temperature, although some data have been accumulated on this subject. It is believed, however, that these observations demonstrate the possibility of obtaining accurate temperature observations for the establishment of any of the numerous correlations which may be suspected to exist; and it is also believed that these observations indicate the practicability of pyrometric control of open-hearth practice, ingot casting, and similar operations.

Of course, the tracing out of the effect, say, of casting temperatures upon this or that ingot characteristic, presupposes the possibility of varying mill practice at will; and in any particular case, considering the large number of other variables that may enter, it will probably take a carefully laid out plan involving a considerable quantity of metal—some of which will undoubtedly be scrapped—to accomplish the end sought. This is also a case where the experimental work has to be done on the same scale as in practice.

1. *Observations in Bessemer Mill*

Table 1 gives the record of furnace casting and ingot teeming temperatures for 19 consecutive Bessemer heats of some 20 tons each of rail steel with a statement of the condition of the ladle as observed by the melter. The temperature estimation of the stream pouring from the converter may be rendered somewhat uncertain by the interference from smoke during the addition of spiegel, and to the intermittent presence of slag, and to the considerable distance from which the stream was viewed since the pyrometer was located on an opposite gallery adjacent to the ingot molds. The temperatures given in the table, for pouring the converter, are each based on a series of observations taken after the addition of spiegel. It is assumed somewhat arbitrarily that the emissivity of this stream is 0.45, corresponding to a temperature correction of 119° C. which gives an average converter pouring temperature of $1,595^{\circ}$ C. ($2,903^{\circ}$ F.) which may be somewhat low. If these apparent temperatures were for a more strictly metallic stream ($e = 0.40$) the true pouring temperature would be $1,623^{\circ}$ C. ($2,953^{\circ}$ F.) on the average.

The teeming temperatures were more exact. Each temperature

recorded is the average of several observations, usually 3 or 4, taken during the filling of an ingot mold of $3\frac{1}{2}$ tons capacity, this operation occupying a time somewhat less than a minute. The emissivity of this stream is taken to be 0.40, which is probably exact to within 0.02 corresponding to a precision of 10° C. at $1,500^{\circ}$ C.; the corresponding correction to observed temperatures is about 125° C. The observed temperatures are uncorrected for emissivity; the average temperatures only are corrected.

The most striking facts these observations bring out are, the relatively narrow temperature range of less than 50° C. within which the Bessemer converter may be and apparently must be operated, and the corresponding remarkable uniformity of ingot teeming temperatures.

2. Observations in Open-hearth Mills

(a) *Temperature of Metal and Slag Streams.*—Table 2 shows the casting and teeming temperatures for a series of heats and top-poured ingots of rail steel from a bank of 60-ton basic open-hearth tilting furnaces. As some 4 to 6 min. are taken in tapping such an open-hearth furnace bath the slag and metal streams may be observed, although it is necessary, on account of the pyrometer station being on opposite gallery, to wait for the smoke to clear away after additions are made to the metal in the ladle. The emissivity of the metal is taken as 0.40, and if that of the (dark) slag is assumed to be 0.65, the slag and metal streams have on the average the same temperature of $1,607^{\circ}$ C.

As in the case of the Bessemer converter, Table 1, but a small variation in temperature from one heat to another is noticeable. The metal had its temperature lowered in the ladle by about 55° C. before teeming and ingot 19 was cast on the average at some 35° C. lower temperature than the first. The casting temperatures of the open-hearth and Bessemer practice here shown are seen to differ but slightly.

The tapping and teeming temperatures for a series of heats of steel of various compositions from acid and basic open-hearth furnaces in another steel plant showed the uncertainty of slag stream temperatures associated principally with the varying color of the slag. It was also demonstrated that all the heats were satisfactory and that the heat of the metal in each case was rated as "hot" which corresponded to a tapping temperature above $1,600^{\circ}$ C. and a teeming temperature between $1,550^{\circ}$ and $1,600^{\circ}$ C., whereas the teeming temperature of the Bessemer and open-hearth rail ingots of Tables 1 and 2 ranged very closely, about $1,525^{\circ}$ C. and $1,550^{\circ}$ C. respectively.

Very satisfactory detailed observations of the temperatures of tapping of several open-hearth furnaces were taken by viewing the stream from the side with the pyrometer within 20 ft. of the tapping stream. It was noted that the heat may or may not be distributed uniformly through the metal in the furnace but the uniformity attained would seem to

depend, in part at least, on the melter. Thus the temperature of the stream of one heat fell off some 80°C . in the 2 min. of pouring; that of two heats remained reasonably constant; and the temperature of the stream of a fourth heat increased some 50°C . showing that the top of the bath must have been considerably hotter than the bottom. The extreme range here observed in the true temperature of the metal in the tapping stream is $1,710^{\circ}$ to $1,520^{\circ}\text{C}$.

In order to show how readily and accurately the temperature of a stream from a furnace, which is being tapped, may be followed, the actual readings taken for tapping heat 12×36 are given in Fig. 1. The condi-

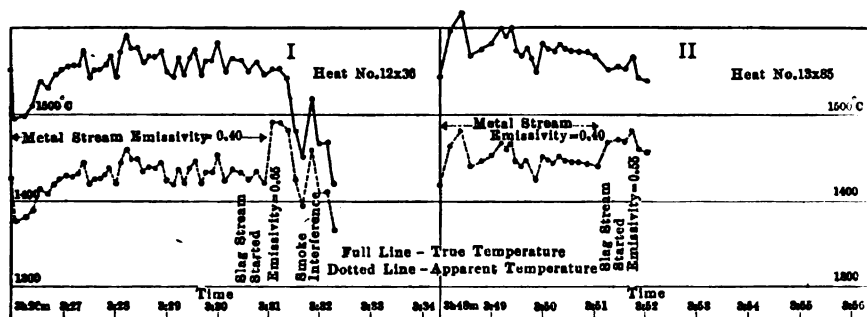


FIG. 1.—OPEN-HEARTH TAPPING TEMPERATURES.

tions of observing this tapping stream were less favorable, due to smoke, than usual. The fluctuations in pyrometer readings particularly at the start and finish are partly caused by intervening smoke. A piece of slag will give an occasional high reading. It will be seen that observations with the optical pyrometer may be taken easily at the rate of six or eight a minute. The observations on tapping another heat 13×85 are also shown in Fig. 1.

The temperatures of miscellaneous operations involving metal streams show that the average, corrected for emissivity, $e = 40$, for metal from a cupola is $1,394^{\circ}$; for metal pouring from a 1,000-ton mixer, $1,372^{\circ}$; for metal pouring into a 1,000-ton mixer, $1,396^{\circ}$; for spiegel pouring into the ladle, $1,289^{\circ}$; for recarbonizing iron pouring from small ladle, $1,287^{\circ}$; and for blast-furnace iron going into open-hearth mixer, $1,356^{\circ}$. The average values in each case have been corrected for emissivity on the basis that $e = 0.40$. All observations were taken with the Morse optical pyrometer.

(b) *Open-Hearth Furnace Temperatures.*—Observations of temperatures of open-hearth furnaces were made for some of the same heats as the casting temperatures of Table 2. The readings of the Morse and Féry pyrometers were corrected for instrument errors, but no correction was applied for emissivity or for absorption of radiation by the gases within the furnace. In the case of the Féry pyrometer, based on total

radiation and calibrated in terms of the Stefan-Boltzmann laws⁶ the correction is much larger than for the Morse or other optical pyrometer. As the amount of gases in the various parts of the furnace varies from time to time, it was not practicable to make any correction for the absorption of these gases. The use of a total radiation pyrometer to determine furnace temperatures does not appear to be warranted.

Each furnace had three doors with openings through which the slag bath could be observed. The method of firing here used impinged the flames down on the bath surface so that the flames interfered quite seriously with obtaining satisfactory observations. The effect of intervening flames is usually to cause the pyrometer to read too high temperatures. Those taken with the optical pyrometer through the middle door were, however, fairly satisfactory. The furnace casting temperature and that of teeming of first ingot were taken for comparison.

An indication of the uniformity of heating the open-hearth bath is seen by comparing the slag and metal casting temperatures with that of the slag surface as viewed in the furnace. Thus for heat 3 X 98 the slag in the furnace attained the unusually high temperature of 1,700° C. and maintained this for 20 min. up to 10 min. before tapping; the slag stream registered 1,621° and the metal only 1,559°. The slag stream appeared to be usually slightly hotter than the metal and the relation of the two is undoubtedly dependent upon the details of methods of firing. In noting the considerable difference between furnace and slag stream temperatures, it is also to be remembered that the furnace cools very rapidly when heat is shut off, which often occurs several minutes before tapping.

In another heat the slag in the furnace registered 1,653°, the slag stream 1,606° and the metal 1,625°; in another heat the three temperatures were 1,635°, 1,612° and 1,597°; in another 1,610°, 1,659° and 1,628°.

In Table 3 are given measurements of temperatures of an acid open-hearth furnace, furnace heat which was followed in detail for 4½ hr. before tapping.

These three methods of observation were used: Sighting through small ports situated well to right or left corners of furnace, the pyrometer being directed toward middle back wall of furnace at arch and wall intersection; on the surface of 30 to 50 lb. of metal removed from furnace in a spoon; and on the slag in middle of the bath through peep hole in center door.

That the roof temperatures may have no relation to those of the metal is seen from the observations of heat 10 X 90, for which the fuel was cut off for 20 min. immediately preceding casting and the temperature of the arch fell from 1,700° C. to 1,378° during the first 14 min., while the metal on the average remained above 1,585°, although the observations on the metal spooned out show that the metal at the top of the bath was

⁶ Burgess and Foote: *Scientific Paper No. 250, U. S. Bureau of Standards.*

cooling very rapidly from 1,640 to 1,520° during those 20 min. of no heat being applied.

The temperatures of the arch are seen to be higher generally than the bath or metal temperatures, usually by some 50° C., although there appears to be no definite relation, except that when the furnace has been clear for some time the bath and arch temperatures tend to become equal. All of these heats were rated as "hot" by the melter and the behavior of the metal in the molds was good.

The metal temperatures taken by observing the metal in a spoon were not as satisfactory as one could wish. They are not, however, inconsistent with the casting temperatures, and those of the bath when sighting on the slag surface.

There is some uncertainty as to the best value of emissivity to use for the clear surface (always greenish in color) of the liquid metal in the spoon. The value here taken is $\epsilon = 0.40$, but there soon develops a haze over the surface, probably of oxide, although a pronounced oxide surface, for which $\epsilon = 0.53$, will not usually form on a well skimmed, hot surface until the temperature has fallen considerably. It appears as if the oxide first formed by the air in contact with the very hot liquid metal is dissolved. It is possible that a value of $\epsilon = 0.45$ for the emissivity of this surface would be more exact; the recorded "spoon" temperatures would then have to be lowered by some 15 or 20° C.

The procedure followed for securing temperature determinations of the metal bath by means of observations on the surface of the metal dipped out in a spoon was as follows:

A place was marked on the floor near the open-hearth furnace charging door and the optical pyrometer sighted upon this mark at an angle of about 45° and set to the approximate reading expected; a spoon to hold 30 to 50 lb. of metal was first preheated, then filled with metal and placed on the mark; the metal was skimmed of slag with a single sweep of wood and the taking of temperature observations begun immediately. The instant of removal of spoon from the furnace was noted with a stop watch and of each temperature observation; also the condition of the surface, whether clear, hazy, oxide, slag, liquid or solid, quiet or boiling, or scintillating. Skimming the surface more than once had sometimes to be resorted to if the steel was not quiet, but with quiet metal the best results were obtained by leaving the surface intact after the first skimming and noting the formation of oxide and applying the emissivity correction for metal or oxide. Temperature observations can usually be begun within 12 sec. of removal of spoon from furnace.

The last two determinations in the spoon of metal temperature of heat 12 X 36 of Table 3 are shown graphically in Fig. 2.

A comparison of the temperatures of several open-hearth furnaces for a time of 3 hr. in various stages of melting was made, the observations

being taken by sighting on the surface of the slag. There is apparently no great difference for the five furnaces after melting has taken place, whatever the condition of the metal; the temperatures ranged from $1,567^{\circ}$ to $1,679^{\circ}$ C. and were usually above $1,620^{\circ}$ C. The bath temperatures, when they can be taken under conditions in which the flames from the fuel do not interfere, appear to give a fairly good indication of the temperature of the upper surface layer of the metal; if the stirring or heating

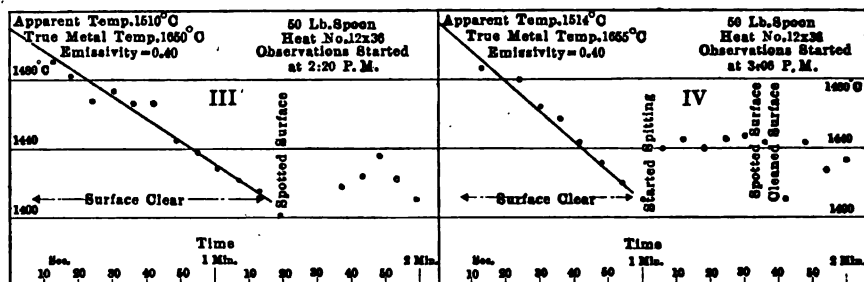


FIG. 2.—FURNACE TEMPERATURES BY SPOON METHOD.

has not been well done, however, the observations will be misleading if they are taken to apply to the metal as a whole.

VIII. SUMMARY AND CONCLUSIONS

The problem of temperature measurement and pyrometric control of furnace casting and ingot teeming temperatures is shown, by a series of observations taken in several steel plants, to present no serious difficulties or uncertainties.

For this purpose the most satisfactory type of instrument is one of the optical pyrometers using monochromatic light and permitting observation from a distance of streams of metal.

It is shown that the necessary corrections to the observed optical pyrometer readings for emissivity of metal and oxides to give true temperatures, are sufficiently well known, but there may be uncertainty in the case of liquid slags.

For streams of liquid iron or steel the most probable value of emissivity to take, with a pyrometer using red light of wave length $\lambda = 0.65\mu$, is $\epsilon = 0.40$, corresponding to a correction of 139° for an observed temperature of $1,500^{\circ}$ C. The value of ϵ for liquid slags is usually about 0.65 but varies with composition of the slag.

Determination of the temperature of the charge of Bessemer converters is not deemed practicable by pyrometric methods.

The operation of the open-hearth furnace can be gaged by the pyrometer, it being possible to control readily the temperature of the roof and of the bath of metal and slag by observations taken through ports; and

the temperature of the metal may be had at any instant, with a fair degree of exactness, by observation with the optical pyrometer of metal removed in a spoon. This operation is more certain with acid than with basic practice, the basic slag being more difficult to remove.

The temperatures of the roof of an open-hearth furnace are shown to bear no necessary relation to that of the metal bath, which, it is again shown, may have zones of considerable differences in temperature depending upon the operation of the furnace.

The temperature of the roof of an open-hearth furnace, dependent upon the firing practice, may vary rapidly and within wide limits, 1,550° to 1,750° C. The temperature of the open-hearth bath is usually kept between 1,600 and 1,670° C.

There is a remarkable degree of uniformity in casting temperature actually acquired by the melters in practice. Thus, for 19 consecutive Bessemer heats the teeming temperatures of the ingots were all between 1,500° C. and 1,555° C., and a similar degree of concordance, although at slightly higher temperatures, was found in the open-hearth practice of several mills.

It is believed that a continuous, systematic following of the temperature, by the methods above outlined, of the various furnace and casting practices, on the part of steel and iron mills, would show the possibility of improvements, and greater certainty of production in quality of product; also changes and the effects of variation in ingot or furnace practice could undoubtedly be carried out with greater certainty than at present appears to be the case.

Several observers assisted in taking the above observations and acknowledgment for their help is due to Messrs. Crowe, Merica, and Waltenberg of the Metallurgical Division of the Bureau of Standards.

Acknowledgment of the opportunity of taking these observations is also gladly made to the management and members of the technical staff of the steel plants at Sparrow's Point, Md., Philadelphia and Bethlehem, Pa., and Washington, D. C.⁷

⁷ In the complete paper are ten tables giving observations in detail with comments. Emissivity correction tables are also included.

TABLE 1.—*Bessemer Mill*

Heat No.	Apparent Temperature of Metal Pouring from Converter	Apparent Temperature of Pouring into Ingot Molds						Mean of 1-6	Average True Temperature of 1-6	Ladle
		1	2	3	4	5	6			
29	1,476	1,397	1,377	1,377	1,389	1,369	1,367	1,379	1,500	Spots.
30	1,472	1,401	1,387	1,393	1,401	1,397	1,393	1,395	1,517	Spots.
31	1,488	1,415	1,419	1,403	1,393	1,383	1,375	1,398	1,521	Spots.
32	1,458	1,397	1,389	1,387	1,379	1,387	1,387	1,388	1,510	Spots—clean.
33	1,468	1,407	1,401	1,407	1,405	1,403		1,405	1,529	Spots.
34	1,454	1,417	1,405	1,431	1,391	1,389	1,413	1,408	1,532	Clean.
35	1,462	1,419	1,417	1,403	1,393	1,405		1,407	1,531	Clean.
36	1,490	1,434	1,429	1,432	1,429	1,419	1,417	1,428	1,555	Clean.
37					1,383	1,372	1,397	1,384	1,505	Skull 4,000 lb.
38	1,490	1,385	1,393	1,377	1,403	1,385	1,385	1,388	1,510	Cut 1,500 lb. skull.
39	1,478	1,429	1,403	1,393	1,389	1,379	1,377	1,398	1,521	Cut 1,500 lb. skull.
40	1,486	1,424	1,408	1,408	1,401	1,399	1,399	1,406	1,530	Spots—cut 1,000 lb. skull.
41	1,492	1,436	1,401	1,375	1,389	1,424	1,399	1,404	1,528	Clean.
42	1,468	1,408	1,403	1,408	1,397	1,391	1,379	1,398	1,521	Clean—spots.
43		1,434	1,419	1,409	1,407	1,389	1,372	1,405	1,529	Hot.
44	1,488	1,433	1,419	1,409	1,405	1,391	1,383	1,407	1,531	Hot.
45	1,474	1,405	1,405	1,405	1,403	1,401	1,387	1,401	1,524	Spots No. 1.
46	1,462	1,383	1,383	1,383	1,391	1,383	1,403	1,387	1,509	Spots.
47	1,484	1,434	1,432	1,417	1,413	1,413	1,409	1,419	1,545	Warmer—clean ladle.
Mean.....	1,476	1,414	1,405	1,401	1,398	1,394	1,391	1,400	1,523	$s = 0.45$ for converter metal.
Correction for emissivity....	119	126	124	123	123	122	122			$s = 0.40$ for ingot teeming.
Mean true temperature, Centigrade...	1,595	1,540	1,529	1,524	1,521	1,516	1,513		1,523	

TABLE 2.—Casting Open-Hearth Furnaces and Ingots

Heat No.	Furnace No.	Apparent Temperature of Pouring into Ladle		Highest and Lowest Apparent Temperature of Pouring into Different Ingot Molds			Apparent Temperature, Av.	True Temperature, Av.	Remarks
		Metal	Slag	Highest	Lowest	Molds			
5	64	1,490	1,540	1,419	1,393	16	1,405	1,529	Spots.
3	97	1,464	1,546	1,409	1,367	16	1,390	1,512	8,000 lb. skull. Melter during teeming said this heat poured hot.
5	65		1,545	1,419	1,375	16	1,402	1,525	
3	98	1,431	1,554	1,421	1,403	19	1,410	1,534	500 lb. skull.
4	62	1,508	1,571	1,407	1,365	17	1,387	1,508	
1	32			1,466	1,393	21	1,420	1,546	Clean.
1	25		1,530	1,448	1,389	22	1,407	1,531	
2	10	1,499		1,427	1,367	18	1,401	1,524	Spots.
3	73	1,425	1,520	1,427	1,397	18	1,412	1,537	5,000 lb. skull.
3	74	1,429	1,572			..			
4	51	1,490	1,546	1,429	1,397	19	1,413	1,538	Spots.
1	36		1,525	1,439	1,389	21	1,414	1,539	
1	41		1,527	1,457	1,410	19	1,414	1,539	Clean.
3	80	1,478	1,546	1,450	1,397	20	1,411	1,536	New ladle spots.
1	42	1,516	1,545	1,448	1,389	19	1,414	1,539	Hot ladle spots.
3	82		1,532	1,424	1,383	19	1,402	1,525	Spots.
1	43	1,468	1,532	1,415	1,375	19	1,392	1,514	
Mean		1,473	1,542	1,424	1,392	26	1,406	1,530	
Emissivity correction ..		134	66	127	122	26			
True temperature, °C		1,607	1,608	1,551	1,514	26		1,531	

 $e = 0.40$ for metal streams. $e = 0.65$ for slag.

TABLE 3.—*Open-Hearth Furnace Temperatures*
 Acid Furnace No. 12, Heat No. 36

Oil from	Roof through Ports		Bath Slag	Spoon Metal	Remarks
	Left	Right			
R	1,647				Pyrometer sighted over flames. Not melted yet.
R	1,666				
R		1,667			
R			1,623		Bubbles appear dark.
R			1,602		Slag bergs.
R			1,627		
R	1,682				Furnace stirred.
R		1,667			
L	1,694				Oil reversed to left. About melted.
L		1,727			
L		1,729			On flames.
L			1,655		Black bubbles.
L				1,566	Reversed oil to right.
R	1,693				Flames.
.....		1,710			Not on brightest flames.
R		1,694			O. K. "Medium furnace."
.....			1,653		
R	1,687				Uniform.
L				1,584	Reversed oil to left.
L				1,618	
L				1,604	
L			1,672		Metal quiet, few bubbles.
L		1,745			Fairly uniform.
L	1,725				Reversed oil. Charging hematite.
R	1,710				Fairly clear.
R		1,710			
R			1,679		Reversed oil.
L				1,649	Reversed oil.
R				1,660	
R	1,648				Clear.
R		1,647			Clear.
R			1,638		
R		1,626			Clear.
R			1,601		Slag bergs.
R	1,633				
R				1,650	
R	1,628				Clear.
R			1,606		
R		1,627			Clear.
R	1,660				Charged ferrosilicon.
R	1,655				Stirred furnace.
R	1,650				
R		1,674			Flames.
.....			1,638		
R				1,655	Charged ferromanganese.
R	1,687				
R		1,702			Flames. Hard tap lasted 5 min. Tapped.

Casting temperature 1,520 to 1,570 in 6 min. Stream viewed from smoky side.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the New York meeting, February, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Apr. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Viscosity of Blast-Furnace Slag*

BY ALEXANDER L. FEILD,† A. B., M. S., PITTSBURGH, PA.

(New York Meeting, February, 1917)

INTRODUCTION

THE Bureau of Mines is investigating the problem of slag viscosity, its variation with the temperature and with the composition of the slag, and its effect upon the distribution of the sulphur between molten iron and slag, as introductory to a series of contributions to the theory of the metallurgical processes. The method of measurement, viscosity apparatus, furnace and other accessories mentioned in this paper have been described fully in a Bureau of Mines report.¹

THE LITERATURE ON SLAG VISCOSITY

The viscosity of blast-furnace slag had not been measured previous to the investigation described in this report. A number of investigators have determined the "fusibility," or softening temperature, of slags by means of the cone method or other deformation method, but these methods supply only limited information regarding the temperature-viscosity relations of the slag in question. Any deformation method that may be employed indicates only the temperature at which the slag attains a more or less definite viscosity, the magnitude of the value obtained depending on the method used. For details of the various methods used by different investigators, see the Bureau of Mines paper already referred to. An isolated measurement by Doelter on a single mixture at 1,300° C. represents the highest temperature reached in the various viscosity investigations previous to those described in this report, in which results have been referred to absolute units. Arbitrary deformation tests have of course been made at much higher temperatures.

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¹A. L. Feild: A Method for Measuring the Viscosity of Blast-Furnace Slag at High Temperatures, *Bureau of Mines Technical Paper* 157, 1916, 29 pp.

THEORY OF THE METHOD EMPLOYED

In these investigations the author has used a modification of the method originated by Margules² in 1881. In this method the liquid is confined between two concentric cylinders. The outer cylinder is rotated at a constant speed and the torque exerted upon the inner cylinder is measured. The method is applicable to liquids of a wide range of viscosity, and has been used, with modifications, by Gurney³ in the accurate measurement of the viscosity of water. The principle of the method is used in the Stormer and Clark viscosimeters, and in those of Carmichael and Grosvenor, and of Kottmann.

The principle of the method employed by the author is therefore not a new one. The torsion method, however, had not hitherto been employed at all for measurements of viscosity at temperatures higher than are generally used in testing oils, etc., *i.e.*, about 250° C. Quite paradoxically it appears to be the only method of measurement which is possible experimentally over a large range of temperature and of viscosity, and for this reason may be called a quite universal method. In applying the method to temperatures as high as 1,600° C., which is about 400° C. higher than the property of viscosity had been hitherto measured by means of any method, certain modifications in the construction of the apparatus were necessary and which alone made possible the attainment of a relatively great accuracy under the rather severe experimental conditions. These modifications consisted in the use of Acheson graphite in the construction of all parts of the apparatus subjected to high temperatures, and in the use of a system of damping the suspended system so as to give to it the stability and aperiodicity characteristic of the familiar damped D'Arsonval galvanometer.

The present modification of the method of Margules has been applied without difficulty to measurements on slags over a range of viscosity from 200 to 3,000 (H_2O at 20° C. = 1). It is certain, however, that the method is applicable over a much wider range of viscosity than this.⁴

For a full discussion of the theory of the method the reader is referred to *Bureau of Mines Technical Paper* 157, pp. 7-10. It is sufficient here to say that the viscosity of the substance under examination is proportional

² Max Margules: Über die Bestimmung des Reibungs- und Gleitungscoefficienten aus ebenen Benegungen einer Flüssigkeit, *Sitzungsberichte, Kaiserliche Akademie Wissenschaften (Vienna)* vol. 83, pt. 2, p. 588 (1881).

³ L. E. Gurney: The Viscosity of Water at Very Low Rates of Shear, *Physical Review*, vol. 26, pp. 98-120 (1908).

⁴ See *Metallurgical and Chemical Engineering*, vol. 15, p. 541 (Nov. 1, 1916), where the application of the method to very viscous substances is discussed, with experimental data on a sample of asphalt over a range of viscosity from 80 to 4,000.

to the torsion couple exerted on the inner concentric cylinder and inversely proportional to the speed of rotation in revolutions per second; while, conversely, the torsion couple is proportional to the viscosity and to the speed of rotation. With cylinders of fixed dimensions, therefore, and with the inner cylinder suspended by means of a calibrated steel ribbon, the viscosity of the slag melt is directly proportional to the product of the time of rotation of the outer cylinder and the deflection produced by means of the confined slag melt upon the inner cylinder.

THE APPLICATION OF VISCOSITY DATA TO METALLURGICAL OPERATIONS

Before taking up the description of the viscosity apparatus, furnace, and accessories, and tabulation of the experimental results obtained, it is advisable to consider briefly the application of accurate viscosity determinations on metallurgical slags to manufacturing processes, and also to related research in this field. Particular attention must be given in the present paper to a consideration of the principles of the metallurgy of iron, although in many other metallurgical operations, such as the smelting of copper for instance, a knowledge of the temperature-viscosity relations of different types of slag is of great importance.

Apart from the question of mining cost and freightage, the value of an iron ore sufficiently rich in iron to be considered marketable is largely dependent upon whether it can be made to yield economically a slag of desirable viscosity and desulphurizing power. A casual glance at the slag analyses in Table 1 shows at once the comparatively wide range of slag composition that has been found practicable by different manufacturers. In each case the particular slag composition was undoubtedly determined in a large measure by the composition of the ore mixture and fuel which it was deemed expedient to use, and also by the grade of iron produced. However, it is entirely probable that in certain cases the slag composition was not the optimum one from the standpoint of economy and excellence of product. It is one of the purposes of these investigations to determine what are the optimum conditions.

In spite of the lack of scientific research on the physical and chemical properties of slags at high temperatures, it is quite well understood what functions the slag must perform in the blast furnace. In the first place, it must be sufficiently fluid to flow from the cinder notch at the temperature which exists in the hearth. In the case of charcoal practice where desulphurization is a minor item, the viscosity of the slag at flush is a primary consideration. If, for instance, it is found that a siliceous charcoal slag possesses a viscosity of 800 at the temperature of the hearth, the question arises whether a limey slag would perform its functions properly if it possessed an equal viscosity. The limey slag may be

prevented from performing these functions for one or both of the following reasons. In the first place, the limey slag with a viscosity of 800 might have an extremely high rate of change of viscosity with temperature, *i.e.*, this particular point on the temperature-viscosity curve might occur at a temperature where the slag underwent rapid softening or hardening with small changes of temperature. In the second place, it might not be at a sufficiently high temperature properly to desulphurize the pig iron.

THE DESULPHURIZATION PROCESS AND ITS RELATION TO SLAG VISCOSITY

The greater portion of the sulphur enters the furnace in the coke, in which it is present as ferrous sulphide to the extent of from 0.5 to 1.4 per cent. sulphur. The total quantity of sulphur is for practical purposes entirely distributed between the molten iron and slag. It may be assumed quite safely that the sulphur which is dissolved by the pig iron exists as ferrous sulphide, FeS . On the other hand, it is necessary to assume in the case of the sulphur dissolved in the slag that it exists for the most part in normal slags as calcium sulphide, CaS . In high-manganese slags, however, it is probable that a portion of the sulphur exists as manganese sulphide.

According to the well-known distribution law of Nernst,⁵ when a substance is distributed between two immiscible solvents, such as molten iron and slag, the ratio of the concentrations of the given substance in the two solvents, when the condition of equilibrium is reached, is constant for any given temperature, *provided* the dissolved substance has the same molecular weight in the two solvents. Moreover, in the case of several dissolved substances each substance distributes itself as though the others were not present.

Let us consider an ideal case where manganese is entirely absent from the slag and iron. Since calcium sulphide is insoluble in molten iron, the distribution effect must be that of ferrous sulphide between molten iron and slag. The ferrous sulphide present in the slag undergoes the following reaction:



Since the concentration of lime, CaO , is very large in comparison with the concentrations of the other reacting substances, it may be considered to be a constant; wherefore, it would follow, from the general law in chemical equilibrium,⁶ that the concentration of the ferrous sulphide of the slag is proportional to the product of the concentrations of the calcium sulphide, CaS , and the ferrous oxide, FeO .

⁵ Walter Nernst: *Theoretical Chemistry*, pp. 485-487. New York, MacMillan & Co., 1904.

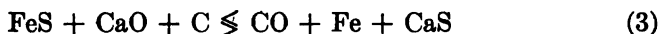
⁶ Nernst: *op. cit.*, pp. 434-435.

Assuming ferrous sulphide to be distributed according to Nernst's distribution law, the concentration of the ferrous sulphide, and, therefore, of the sulphur, in the pig iron would be proportional to the product of the calcium sulphide and ferrous oxide concentrations in the slag layer, provided no other side reactions occurred.

However, there is a side reaction which must inevitably occur since ferrous oxide is not stable at high temperatures in the presence of carbon. This reaction is as follows:



On combining equations (1) and (2), the net effect is seen to be:



Equation (3) is the usual one given by most authorities to illustrate the course of the desulphurization process.

In the course of the chemical reaction represented by equation (3) it is necessary that FeS diffuse from the iron into the slag and also that the CaS, which is a product of the reaction, diffuse with sufficient rapidity to prevent the reversible reaction from coming to a standstill prematurely. For this reason the desulphurization reaction is conditioned largely so far as its speed is concerned by the viscosity of the slag, and resolves itself into a reaction confined to a great extent to the boundary plane between molten iron and slag. This boundary plane exists around the molten iron globules previous to their fall to the bottom of the hearth and between the contiguous layers of iron and slag in the hearth. To what extent desulphurization is completed previous to and during the fall of the iron globules through the slag layer is not known.

If it is true that during the smelting process equilibrium is reached under normal conditions in the case of the partition of sulphur between iron and slag, then the desulphurizing power of any given slag is independent of the viscosity of the slag, as such. In such a case the distribution of the sulphur would be a function only of the composition of the iron and slag and of the temperature of the hearth. However, if equilibrium is not reached under normal conditions, there cannot be in the strict sense of the word a distribution of sulphur, since the laws of distribution refer only to equilibrium conditions. In this case, the amount of sulphur absorbed by the slag depends on the speed of the reaction of desulphurization, which, as has been noted above, is conditioned by the viscosity of the slag, as well as by the temperature and composition of molten iron and slag.

With the help of the temperature-viscosity data of a number of different slags, obtained by means of the method which is described in this report, the Bureau of Mines is undertaking research investigations to determine the following facts: (1) By determining the speed of sulphide

absorption by different slags at such temperatures as to possess identical viscosities, to deduce the effect due simply to differences of temperature and composition; (2) by determining the speed of sulphide absorption by the same slag at different temperatures, to deduce the effect due to changes in viscosity and temperature; (3) and by determining the speed of sulphide absorption by different slags at the same temperature, to deduce the effect due to changes in slag composition and in viscosity. Also, in each case, the final conditions of equilibrium are to be determined.

This work when completed should give a clear insight into the mechanism of the desulphurization process, and should be a test of the validity of the theory of Turner⁷ and of Schafer,⁸ that the silicate of the slag acts as a comparatively inert solvent for the spinels (aluminates) which alone are capable of reacting with the sulphide sulphur in the pig iron. According to this theory a slag low in alumina should possess a relatively weak desulphurizing power; and there should exist a lower limit of alumina content below which a slag could not be made to act satisfactorily in the blast furnace.

THE RELATION BETWEEN THE COMPOSITION OF BLAST-FURNACE SLAGS AND THEIR SOFTENING TEMPERATURES DETERMINED BY THE CONE TEST

Formerly in the absence of a method of viscosity measurement at the extreme temperatures of the blast-furnace hearth, it was customary for investigators to obtain an idea of the softening temperature or deformation temperature of slags by means of the familiar cone tests, in which test pieces similar to Seger cones are made from the slag and the temperature noted at which these test pieces bend over, melt to a ball, or otherwise show signs of incipient fusion.

In the ceramic industries and also in determinations of the fusibility of coal ash, cone tests furnish valuable information, since in these cases the important point in view is to determine the minimum temperature at which deformation or softening occurs. If, for instance, a firebrick softens appreciably at 1,600° C., it is not a matter of technical interest to know how this brick would behave at 1,800° C., nor would a cone test give any information in regard to this question.

Mellor⁹ has defined the softening temperature concisely, as follows: "The softening temperature measures the temperature at which the inward surface pressure of the substance becomes greater than those intra-

⁷ *Journal of the Society of Chemical Industry*, 1905, p. 1142; *Stahl und Eisen*, vol. 26, No. 1, p. 172 (1906); *Metallurgie*, p. 164 (1906).

⁸ *Ferrum*, vol. 5, p. 129 (Feb. 8, 1913).

⁹ J. W. Mellor, A. Latimer, and A. D. Holdcroft: *The Softening Temperatures of Lead-Silica Glasses*, *Transactions of the English Ceramic Society*, vol. 9, pp. 126-149.

molecular forces which hinder the molecules taking up a position of stable equilibrium-minimum surface area."

It is undoubtedly true that silicate mixtures at temperatures at or near their softening point possess a surface tension of considerable magnitude, as may be shown by the rounding off at the end of a glass rod when held in the flame of a Bunsen burner. What is observed actually is the resultant effect of decrease of viscosity and action of surface forces upon the softened body.

There is given in Table 1 the softening temperatures and analyses of 18 commercial slags. These determinations were made in a platinum-wire resistance furnace in an atmosphere of air, temperatures being measured by a platinum-rhodium thermocouple. The sulphur content of the slag is accordingly entirely converted to the sulphate by the oxygen of the atmosphere, and possibly a portion was volatilized. The measurements cannot, therefore, be considered as representing entirely the conditions within the blast furnace with its atmosphere of carbon monoxide and nitrogen. The slags are arranged in the table in the order of increasing refractoriness.

TABLE 1.—*Softening Temperatures of Slags by the Cone Test*

Slag Lab. No.	SiO ₂	Al ₂ O ₃	Percentage of		CaS	MnO	Softening Temperature, °C.
			CaO	MgO			
22,967	48	8	32	5	2.0	0.1	1,244-1,254
22,965	38	10	40	4	3.1	1.2	1,262-1,264
22,964	38	9	43	2	2.4	0.2	1,263-1,266
22,968	44	9	40	2	2.7	0.2	1,279-1,279
22,960	37	11	25	20	3.5	2.2	1,297-1,300
22,958	34	27	27	6	4.9	0.3	1,342-1,342
22,953	36	12	41	6	3.1	0.7	1,331-1,346
22,956	35	11	42	7	3.6	0.5	1,352-1,357
22,963	34	14	41	6	3.4	0.6	1,343-1,360
22,969	34	12	43	6	3.2	0.5	1,358-1,364
22,961	34	15	38	10	2.9	0.3	1,365-1,368
22,955	32	16	44	1	4.4	0.1	1,356-1,390
22,952	32	12	45	6	3.4	0.5	1,383-1,391
22,957	31	15	36	10	5.5	0.2	1,388-1,398
22,954	18	35	31	10	4.1	0.3	1,410-1,410
22,966	33	11	44	4	5.9	0.5	1,425-1,441
22,962	32	15	48	2	3.5	0.2	1,403-1,443

While it is impossible to draw any definite conclusions as to the variation of the softening temperature with the composition from these measurements, it is at once evident that, in so far as the effect of silica is concerned, low softening temperatures are in general associated with high

silica content in the slag. In regard to the effect of lime or of alumina the data in the table yield nothing conclusive.

As is well known, the softening of a silicate mixture such as blast-furnace slag depends upon the amount of eutectic which is formed during incipient fusion and upon the viscosity of this eutectic. It is obvious that, given a sufficient amount of this eutectic to overcome the rigidity imparted to the test piece by the unfused portion, deformation will occur even though the eutectic possesses an extremely high viscosity. This softening may require time on account of the high viscosity of the eutectic mixture, but in cone tests sufficient time is usually furnished for this slow deformation.

Reference to the work of Rankin and Wright¹⁰ at the Geophysical Laboratory on the system lime-alumina-silica, shows that the minimum ternary eutectic of this system is composed of CaO, 23.25 per cent., Al_2O_3 , 14.75 per cent., and SiO_2 , 62 per cent., which melts at a temperature of $1,170^\circ\text{C}$. Further examination shows that in all cases low-melting binary or ternary eutectics in the system lime-alumina-silica correspond to a higher silica content than is usually found in blast-furnace slags. Therefore, in all blast-furnace slags which possess more than the usual amount of silica there is formed on incipient fusion a large amount of a low-melting eutectic, which when it attains a sufficiently low viscosity, causes the test piece to soften and deform visibly.

It follows that high-silica slags, which are known to be more viscous at furnace temperatures than more basic slags, possess quite paradoxically a very low softening temperature. One might attempt to explain the high viscosity of siliceous slags in practice by claiming that such slags produce a low hearth temperature, and that the high viscosity is due to the low temperature of the slag. Whether this be true or not, the measurements which have been made in these investigations show conclusively that at all practicable hearth temperatures a siliceous slag is much more viscous than a normal slag at the same temperature.

If, therefore, one should assume that the slags given in Table 1 required the same amount of superheating above the softening temperatures there given, 200°C ., for instance, no importance could be attached to subsequent deductions, because actual measurements of viscosity and temperature show that during a range of 200° of superheating certain slags attain a fluidity three times as great as others.

Emphasis should be placed upon the distinction between the physical melting point and the softening temperature of silicate mixtures, which latter is often loosely referred to as the "melting point." The former refers to a temperature which is perfectly definite in the case of the majority of silicate mixtures, viz., the temperature at which the *last*

¹⁰ G. A. Rankin and C. E. Wright: Ternary System: $\text{CaO} - \text{Al}_2\text{O}_3 - \text{SiO}_2$, *American Journal of Science*, ser. 4, vol. 39, pp. 1-79 (1915).

crystalline phase disappears upon slowly raising the temperature of the melt. Only in the case of silica¹¹ itself and certain alkaline feldspars¹² does the physical melting point appear to be a rather uncertain point. The softening temperature may differ enormously from the physical melting temperature, nor can deductions be made simply on the basis of melting-point determinations as to the relative positions of the softening and melting points, with respect to each other. For complete information in regard to the melting points and stability relations of the silicates the reader is referred to the monumental work¹³ of Day, Shepherd, Rankin, Wright, Merwin, and Sosman at the Geophysical Laboratory.

SLAG TEMPERATURES MEASURED AT THE BLAST FURNACE

In the course of the investigations described in the present paper a number of attempts were made by the author, with the coöperation of Dr. J. K. Clement, physicist of the Bureau of Mines, to determine slag temperatures by means of a Holborn-Kurlbaum optical pyrometer. The conclusion reached in the course of these observations was that, due to the impossibility of obtaining a clean surface on the outflowing slag and of obtaining a slag which was entirely free from fumes, such measurements, at the best, were liable to a very large error. Accordingly a special type of platinum-platinum-rhodium thermocouple was constructed, the inner mechanism of the couple being protected by a sheath of Acheson graphite where subjected to the action of the molten slag.

For the sake of comparison, a number of simultaneous measurements of temperature by means of the optical pyrometer and thermocouple were made at furnaces Nos. 2 and 3 of a large Pittsburgh steel company. For purposes of illustration a few of these results are given in Table 2.

¹¹ C. N. Fenner: The Stability Relations of the Silica Minerals, *American Journal of Science*, ser. 4, vol. 36, pp. 331-384 (1912).

¹² A. L. Day and E. T. Allen: The Isomorphism and Thermal Properties of the Feldspars, with optical study by J. P. Iddings, *Publication No. 31, Carnegie Institute of Washington*, 1905.

¹³ A. L. Day and E. S. Shepherd: The Lime-silica Series of Minerals, with optical study by F. E. Wright, *American Journal of Science*, ser. 4, vol. 22, pp. 265-302 (1906).

E. S. Shepherd and G. A. Rankin: The Binary Systems of Alumina with Silica, Lime, and Magnesia, with optical study by F. E. Wright, *American Journal of Science*, ser. 4, vol. 28, pp. 293-333 (1909). Preliminary Report on the Ternary System $\text{CaO} - \text{Al}_2\text{O}_3 - \text{SiO}_2$; a Study of the Constitution of Portland Cement Clinker, with optical study by F. E. Wright, *Journal of Industrial and Engineering Chemistry*, vol. 3, pp. 221-227 (1911).

G. A. Rankin and F. E. Wright: Ternary System $\text{CaO} - \text{Al}_2\text{O}_3 - \text{SiO}_2$, *American Journal of Science*, ser. 4, vol. 39, pp. 1-79 (1915).

G. A. Rankin and H. E. Merwin: The Ternary System $\text{CaO} - \text{Al}_2\text{O}_3 - \text{MgO}$. *Journal of the American Chemical Society*, vol. 38, pp. 568-588 (1916).

R. B. Sosman: The Common Refractory Oxides, *Journal of Industrial and Engineering Chemistry*, vol. 8, pp. 985-990 (1916.)

TABLE 2.—*Temperatures of Slag at Flush by Means of Holborn-Kurlbaum Optical Pyrometer and Platinum Platinum-Rhodium Thermocouple*

Date	Flush	Temperature	
		H. K. Pyrometer	Thermocouple
Furnace No. 2.		°C	°C
Jan. 24, 1916.....	2d, a.m.	1,487	1,470
.....	3d, a.m.	1,487	1,572
Furnace No. 3.			
Jan. 24, 1916.....	1st, a.m.	1,465	1,525

Further measurements were made with the thermocouple alone. Some of these measurements are given in Table 3.

TABLE 3.—*Temperatures of Slag by Means of a Platinum Platinum-Rhodium Thermocouple*

Date	Furnace No.	Temperature, °C.	Date	Furnace No.	Temperature, °C.
Jan. 25, 1916.....	2	1,515	Jan. 25, '16	2	1,415
.....	3	1,485		2	1,490
.....	3	1,502		2	1,538
.....	3	1,440		3	1,475

The average temperature at flush by means of the thermocouple was, on the basis of the data given in Tables 2 and 3, equal to 1,500° C. for furnace No. 2, and 1,485° C. for furnace No. 3, with a variation in temperature at flush during the day of 108° C. (the 1,415° reading is omitted since its correctness is questionable) in the former case, and of 85° C. in the second case. Both furnaces appeared to be working perfectly normally, so far as could be judged from the behavior of the slag at flush.

Table 4 gives the analysis of the slag from both furnaces on the day upon which the temperature readings were taken.

TABLE 4

	Furnace No. 2, Per Cent.	Furnace No. 3, Per Cent.
SiO ₂	35.18	37.60
Al ₂ O ₃	12.08	12.46
CaO.....	43.73	40.80
MgO.....	4.23	3.66
CaS.....	4.21	3.65
MnO.....	0.39	0.55
	99.82	98.72

An examination of what Johnson calls the curve of free-running temperature¹⁴ shows a decided minimum at a silica content of approximately 44 per cent., this minimum free-running temperature corresponding to 2,450° F. or 1,339° C.

The results of the present investigations demonstrate conclusively that, for alumina content approximating 12 per cent., the slags of minimum free-running temperature correspond with a certain degree of approximation to a silica content of 36 per cent., rather than 44 per cent.

It is of interest to note that such a silica content (36 per cent.) corresponds to average blast-furnace practice using coke as fuel.

Of course, in charcoal practice using high-silica slags, it is possible to work with a more viscous slag than is the case in coke practice. However, in constructing such a curve as given by Johnson, the term "free-running temperature" necessarily refers to a temperature corresponding to a definite and constant viscosity.

THE TEMPERATURE-VISCOSITY RELATIONS OF GRADED SERIES OF SYNTHETIC SLAGS

The only satisfactory method of studying the temperature-viscosity relations of blast-furnace slags and the effect of the different constituents on the viscosity is the determination of such relations for a graded series of synthetic slags in which one constituent is gradually replaced by another. Such investigations are now in progress in these laboratories.

To illustrate the experimental results which are being obtained, there is shown in Fig. 1 the temperature-viscosity relations of a synthetic slag containing 48 per cent. lime (curve A) and of a synthetic slag of the same composition except that 20 per cent. of the lime has been replaced by an equal percentage by weight of magnesia.

DESCRIPTION OF FURNACE AND ACCESSORIES

The electric furnace used in the experiments of the author is of the granular carbon, resistance type, and is similar as regards the arrangement of resistor and supporting refractories to the one described by Jeffries.¹⁵ The arrangement of the furnace and accessories are shown in Plate 1. This is also described fully in *Technical Paper No. 157* of the Bureau of Mines.

¹⁴ J. E. Johnson, Jr.: On The Operation of the Blast Furnace, *Metallurgical and Chemical Engineering*, vol. 14, pp. 363-372 (Apr. 1, 1916).

¹⁵ Zay Jeffries: Notes on the Gran-Annular Electrical Furnace, *Metallurgical and Chemical Engineering*, vol. 12, pp. 154-157 (March, 1914).

Rotating Device

A $\frac{3}{4}$ -in. steel shaft, by means of which the cylindrical crucible containing the slag is rotated, projects up into the furnace through a stuffing box placed centrally in the base of the sheet-iron shell. The lower end of the shaft is supported on a bearing fixed to the tripod. The steel shaft

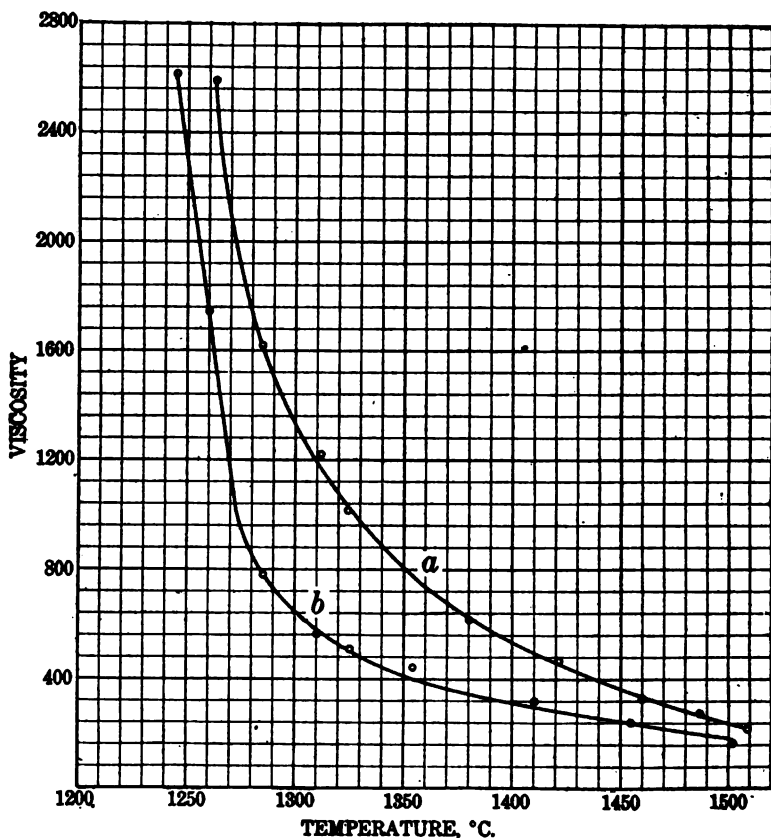


FIG. 1.—(a) TEMPERATURE VISCOSITY CURVE OF SYNTHETIC SLAG CONTAINING 48 PER CENT. CaO . (b) SAME SLAG IN WHICH 20 PER CENT. CaO HAS BEEN REPLACED BY 20 PER CENT. MgO .

extends only a short distance beyond the bottom of the furnace shell, and is bored for the reception of a slightly smaller supporting rod of Acheson graphite. This supporting rod is threaded at its upper end for the reception of the graphite slag crucible (see Plate 2), the graphite rod being tapered at its upper end to a diameter of $\frac{3}{8}$ -in. in order to eliminate, as far as possible, conduction of heat from the crucible. The crucible possesses an inside diameter of $\frac{3}{4}$ in. and an inside depth of 3 in., with a thickness of wall of $\frac{1}{4}$ in. The bottom of the crucible is $\frac{1}{2}$ in. thick, so

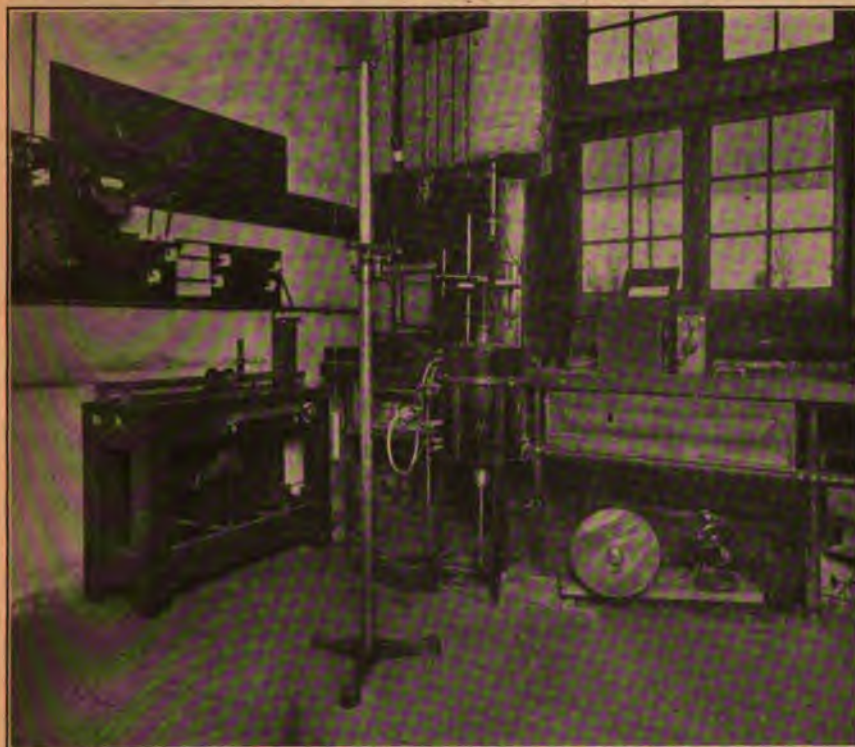


PLATE 1.—APPARATUS FOR MEASURING VISCOSITY OF BLAST-FURNACE SLAGS.

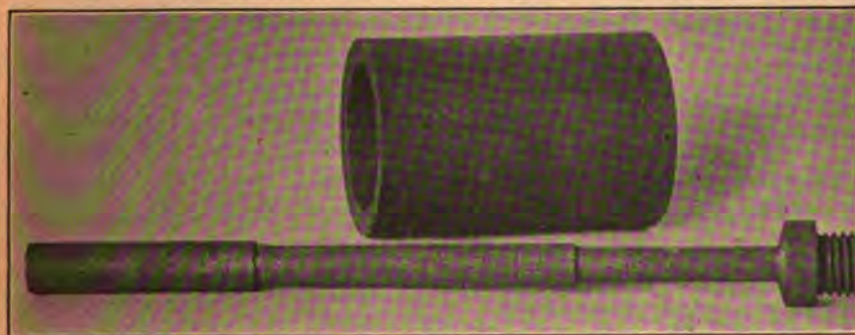


PLATE 2.—GRAPHITE CRUCIBLE AND SUPPORTING ROD.

that the rod may be firmly threaded into it. The supporting rod fits snugly into the hole in the axis of the steel shaft to a depth of $2\frac{1}{2}$ in., and its length between the top of the steel shaft and the bottom of the crucible is about 6 in. In the earlier experiments with the method it was thought necessary to introduce a Marquardt porcelain rod between the crucible and the steel shaft to prevent excessive heat conduction. By using a graphite rod tapered down to a diameter of only $\frac{3}{8}$ in., no excessive heat conduction has been noted; and this also eliminates the undesirable experimental features of using a brittle porcelain part in the supporting system.

Source of Heat

The heating current is supplied by a transformer equipped with an adjustable panel which gives voltages ranging from 40 to 100 volts in steps of 2 volts. The maximum power input amounts to slightly over 5 kva. and the maximum current to 110 amp.

Furnace Atmosphere

The normal atmosphere of the furnace is a mixture of carbon monoxide and nitrogen, corresponding to that in the hearth of the blast furnace.

Viscosity Apparatus

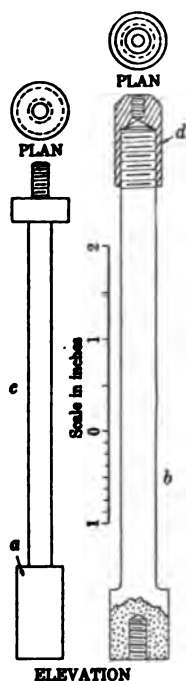


FIG. 2.—VISCOSITY SPINDLE AND CONNECTING ROD. LEFT VIEW, VISCOSITY SPINDLE—*a*, GRAPHITE SPINDLE; *c*, SPINDLE SHAFT. RIGHT VIEW, CONNECTING ROD—*b*, GRAPHITE ROD; *d*, STEEL CONNECTING COLLAR.

Acheson graphite seems to be the only refractory available for such use as that of high-temperature viscosity measurements on blast-furnace slag. This possesses the advantages of having a very low coefficient of expansion, a relatively great mechanical strength at high temperatures, desirable working properties, and shows inappreciable alteration on being kept for long periods at high temperatures. The action of the molten slag on the viscosity spindle immersed in it is so small that the radius of the spindle decreases only approximately 0.005 cm. during a viscosity determination. The action of the slag is perfectly uniform over the surface of the spindle, so that the same spindle may be used repeatedly, due allowance being made for the slight change in its radius.

The construction of the viscosity spindle and connecting rod is shown in Fig. 2. The large cylindrical part, *a*, at the base of the spindle (on the left of the figure) dips into the molten slag, while the upper part is threaded

into the base of the connecting rod (shown in the right of the figure). This connecting rod is also made of graphite, except for the steel cap at its upper end, *d*, by means of which the connecting rod is in turn attached to the upper part of the suspended system shown in Fig. 3.

The distance from the surface of the slag to the bottom of the steel suspension (*r*, Fig. 3) is approximately 21 in. Such a system would not in itself be at all stable under the conditions of measurement. It was therefore necessary to place a weight of about 200 grams between the sus-

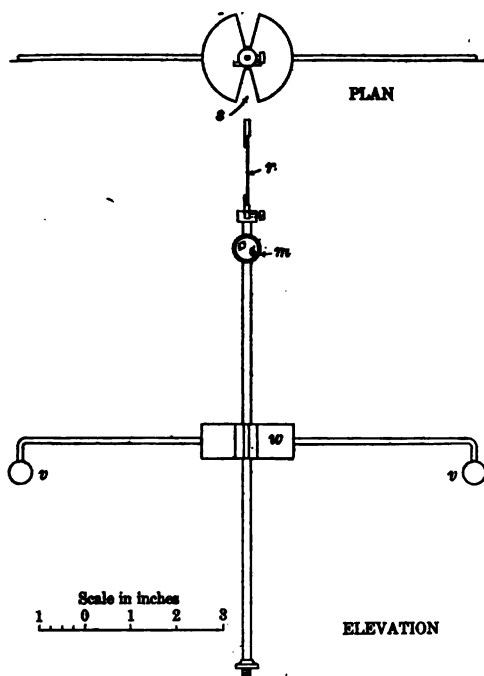


FIG. 3.—UPPER PART OF SUSPENSION SYSTEM. *m*, MIRROR; *r*, STEEL SUSPENSION RIBBON; *s*, OPEN SECTOR, *w*, WEIGHT; *v*, VANES.

pension ribbon and the spindle at a point just outside the furnace shell and about $4\frac{1}{2}$ in. below the suspension. This weight gives stability to the system, decreases the effect of vibrations upon the mirror, and holds the spindle in its position in the slag mass. Moreover, in order to further increase the inertia of the system about its longitudinal axis and to render the system aperiodic and practically non-susceptible to disturbing influences of short duration, the 200-gram weight (see *w*, Fig. 3) is furnished with two diametrically opposite vanes (*v* and *v'*) which dip into cups filled with a viscous fluid, such as castor oil, at a distance of about 4 in. from the axis of the system.

The mirror deflection is measured in the usual way by means of a telescope and scale.

The entire suspension system is fixed to a rod that projects horizontally from an apparatus support, firmly fixed to a heavy table and provided with a millimeter scale for precise vertical adjustment of the apparatus. The furnace itself is first adjusted so that the steel shaft is in a vertical position, which insures the alignment of the crucible also. The point to which the suspension system is fixed is then aligned by means of a plumb bob with the axis of the shaft and crucible. In this manner the viscosity spindle is suspended centrally within the crucible.

The fluctuation of the scale reading rarely amounts to more than 1 per cent. of the scale reading. It is probable that the mirror reading which is a measure of the viscosity of the slag is more precise than the reading of temperature in certain cases, although not so accurate.

The Measurement of Temperature

The temperature of the slag was measured by means of an optical pyrometer of the Holborn-Kurlbaum type. The pyrometer is sighted directly upon the surface of the rotating slag through an open sector in the 200-gram weight (see Fig. 3).

As a check on the black-body conditions of the furnace used in the viscosity measurements, a sample of artificial diopside, $\text{CaO} \cdot \text{MgO} \cdot 2\text{SiO}_2$, was prepared, and its temperature-viscosity relations determined in the same manner as adopted in the case of slags. This compound has a physical melting point at $1,392^\circ \text{C.}$ and is converted, coincidentally with its melting, from a crystalline substance to a fairly fluid liquid.

The results of these measurements are given in Fig. 4. The viscosity curve bends quite sharply upward in the neighborhood of the theoretical melting point, the observed difference of about 20°C. being probably due to the fact that the artificial diopside did not have exactly the theoretically correct composition. Analysis of the artificial diopside gave the following composition: CaO , 25.98 per cent., MgO , 19.48 per cent.; SiO_2 , 55.08 per cent. The correct composition is: CaO , 25.84 per cent.; MgO , 18.58 per cent., SiO_2 , 55.58 per cent.

Operation and Manipulation of Apparatus

The method of using the apparatus is as follows: 175 grams of slag is placed in the graphite crucible. This quantity of slag, when ground to 100 mesh, approximately fills the crucible. Although the height of the crucible internally is 3 in. the slag after having been melted and cooled forms a cylinder only about $1\frac{1}{2}$ in. in length. The crucible, containing the slag, is properly placed, and the upper removable part of the furnace is fixed in position. The crucible is then rotated and the furnace is heated at such a rate that at the end of about $1\frac{1}{2}$ hr. the slag reaches a temperature of $1,200^\circ \text{C.}$ The rate of heating is then decreased in order that the large mass of slag may acquire the temperature of the heating

tube and diminish the lag effect. When the slag has become sufficiently fluid, as determined by several tentative lowerings of the viscosity spindle, the suspension system is carefully lowered until the viscosity spindle rests on the bottom of the crucible. During this operation the rotation of the crucible is stopped in order to avoid the possibility of breaking the graphite spindle while it is being introduced into the viscous slag. The spindle is then raised exactly 0.5 cm. above the bottom of the crucible, the adjustment being made by means of the scale in the apparatus support.

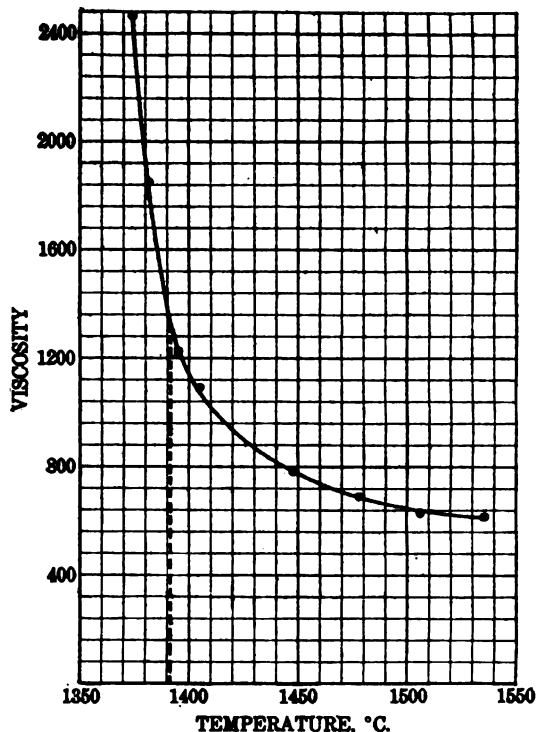


FIG. 4.—TEMPERATURE-VISCOSITY CURVE OF AN ARTIFICIAL DIOPSIDE.

After the suspension system has been lowered and the depth of the viscosity spindle adjusted, the temperature of the slag is slowly raised at a rate of 2° to 5° C. per minute, the temperature and scale deflection being noted at approximately regular intervals. When the highest temperature desired, usually $1,500^{\circ}$ to $1,600^{\circ}$ C., is reached, the heating current is so adjusted that the furnace cools at approximately the same rate that it was heated. Readings of deflection and temperature are taken, until the maximum deflection (corresponding to a viscosity of 2,000 to 3,000) is reached. The spindle is then raised out of the slag before the latter becomes hard enough to prevent withdrawal.

Calibration of the Viscosity Apparatus

For the method of calibration used, the reader is referred to *Technical Paper 157*, p. 18. Measurements are referred to the viscosity of water at 20° C. as unity. In order to convert to c.g.s. units, these values should be divided by 100.

RESULTS OF VISCOSITY MEASUREMENTS

The results of viscosity measurements on eight different commercial slag samples (Lab. Nos. 22,954, 22,962, 22,958, 22,963, 24,016, 23,663, 22,960, and 22,968) are given in Table 5 for the temperature range 1,275° to 1,575° C. The composition of the slags is shown by the results of chemical analyses given in Table 6.

TABLE 5.—*The Viscosity of Typical Blast-furnace Slags at Various Temperatures*
[H₂O at 20° C. = 1]

Temperature, °C.	Viscosity of Slag, Lab. No.							
	22,954	22,962	22,958	22,963	24,016	23,663	22,960	22,968
1,275.....				1,400	2,600		1,900	
1,300.....			1,700	1,000	1,500			
1,325.....			1,060	820	1,080	2,000	1,100	3,000
1,350.....	2,600		800	680	800	740	840	1,800
1,375.....	1,150		640	540	620	560	640	1,220
1,400.....	750	3,000	510	460	500	480	510	1,000
1,425.....	540	580	420	400	420	410	440	820
1,450.....	410	480	350	350	350	360	380	680
1,475.....	320	415	300	310	290	300	320	580
1,500.....	230	380	260	280	250	250	280	480
1,525.....	160	340	225	250	230	200	240	400
1,550.....		290		220	205			340
1,575.....								310

TABLE 6.—*Results of Analyses of the Eight Slags*

Lab. No.	Composition of Slag													Total
	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	FeO	Fe	TiO ₂	CaO	MgO	MnO	CaS	Alka- lies	Mois- ture		
	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.		
22,954	18.30	35.30	0.00	0.47	0.05	0.58	31.24	9.69	0.35	4.07	0.50	0.02	100.57	
22,962	31.54	14.79	0.00	0.07	0.22	0.29	47.65	1.80	0.21	3.56		0.10	100.23	
22,958	33.67	26.62	0.00	0.28	0.51	0.28	26.67	6.43	0.33	4.86		0.12	99.77	
22,963	34.27	13.78	0.00	0.07	0.28	0.56	41.30	6.39	0.55	3.35		0.04	100.59	
24,016	35.76	13.36					42.11	3.94	0.49	3.70			99.36	
23,663	36.04	12.10					42.04	4.03	0.35	3.92			98.48	
22,960	37.18	11.46	0.00	0.31	0.11	0.52	25.33	19.58	2.21	3.51		0.04	100.25	
22,968	43.56	9.48	0.00	0.21	0.38	0.19	40.18	2.08	0.21	2.75	0.50	0.05	99.59	

*Includes Fe and FeO as Fe₂O₃.

Temperature-viscosity curves were plotted, as shown in Figs. 5 to 8 for slags Nos. 22,954, 22,958, 23,663, and 22,968. The curves closely approximate in form a rectangular hyperbola. These figures contain all the experimentally determined points obtained in the heating and cooling

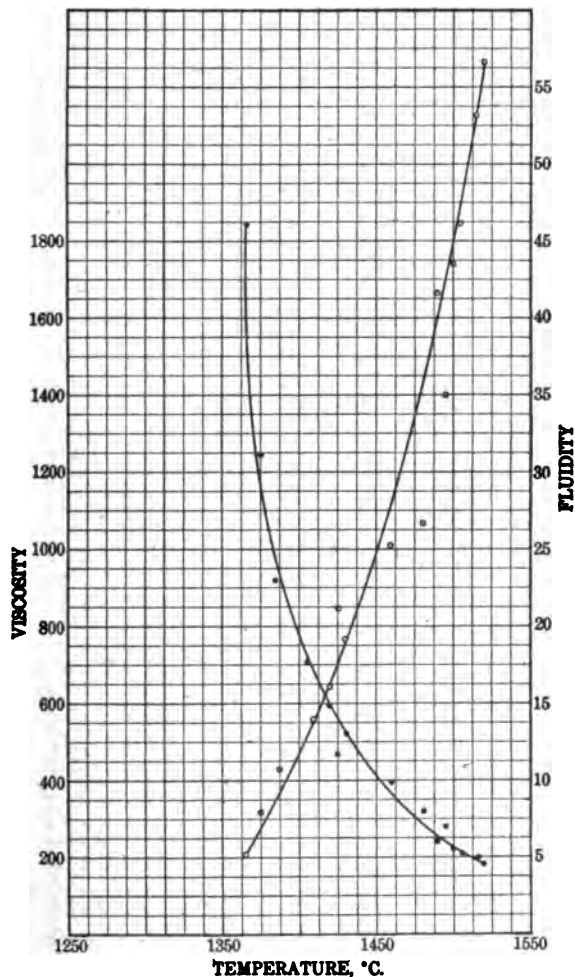


FIG 5.—RELATIONS OF TEMPERATURE TO FLUIDITY AND VISCOSITY SHOWN IN EXPERIMENTS WITH SLAG NO. 22,954.

of the sample, so that a smooth curve drawn through all the points must represent very closely the ideal conditions desired, the deviations tending to neutralize one another.

The temperature-fluidity curves are also given in Figs. 5 to 8 for the four slags mentioned above. The values are calculated by means of the

familiar relation, Fluidity = $1/\text{Viscosity}$. As was to have been expected from the general shape of the viscosity curves, the curves representing fluidity at various temperatures approximate in form a straight line.

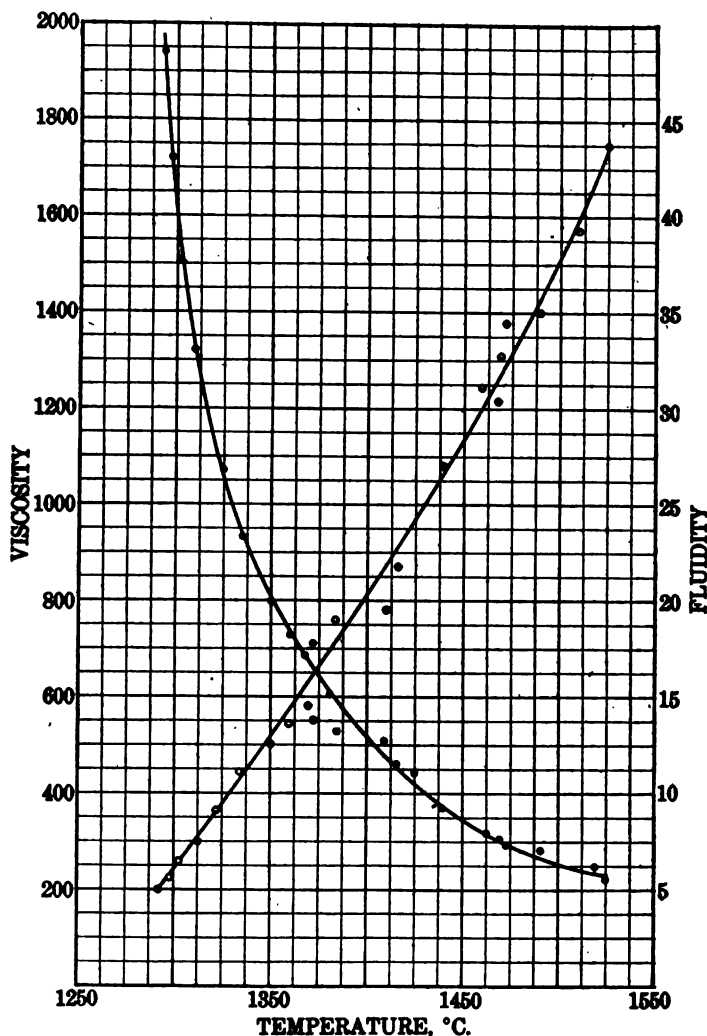


FIG. 6.—RELATIONS OF TEMPERATURE TO FLUIDITY AND VISCOSITY SHOWN IN EXPERIMENTS WITH SLAG No. 22,958.

The actual experimental data obtained in the case of slag No. 22,958 are given in Table 7, which contains results obtained with rising and falling temperatures.

TABLE 7.—*Comparison of Viscosity Values Obtained with Rising and Falling Temperatures*

[Results obtained in experiments with slag designated Lab. No. 22,958]

Rising Temperature		Falling Temperature ^a			
Temperature, °C.	Viscosity	Temperature, °C.	Viscosity	Temperature, °C.	Viscosity
1,382	605	1,292	1,940	1,382	611
1,410	513	1,297	1,707	1,387	524
1,415	460	1,302	1,505	1,425	446
1,425	442	1,312	1,319	1,440	375
		1,322	1,074	1,466	323
1,463	320	1,337	922	1,470	304
		1,350	805	1,488	285
1,473	291	1,359	728	1,525	228
1,510	255	1,368	689		

^a Results are given in reverse order, for easy comparison with results under "Rising Temperature."

Duplicate determinations on separate samples of the same slag gave results which agreed well within the known errors of measurement. In Table 8 are given the results of duplicate determinations on slag No. 23,663. The deflections observed with falling temperature are given for both slags, these deflections being proportional to the viscosity of the melt.

TABLE 8.—*Duplicate Measurements on Slag No. 23,663*

Original		Duplicate	
Temperature, °C.	Deflection	Temperature, °C.	Deflection
1,510	26		
1,487	31	1,487	32
1,477	34	1,472	34
1,460	38		
1,450	39	1,450	40
1,435	40	1,440	40
1,425	42	1,430	43
1,410	44	1,410	45
1,387	48	1,387	48
1,337	90	1,347	82

GENERAL APPEARANCE OF SLAG MELTS

When cold the slag was easily removed from the graphite crucible by pressing on the bottom of the melt through the hole into which the graphite crucible support is threaded. The slag on cooling shrinks away

from the graphite crucible wall sufficiently to prevent objectionable binding and sticking. The melt as taken from the crucible is cylindrical with an upper surface which is almost flat, as the viscosity at the moment when the spindle was withdrawn was not great enough to prevent the slag from flowing into the space previously occupied by the graphite spindle.

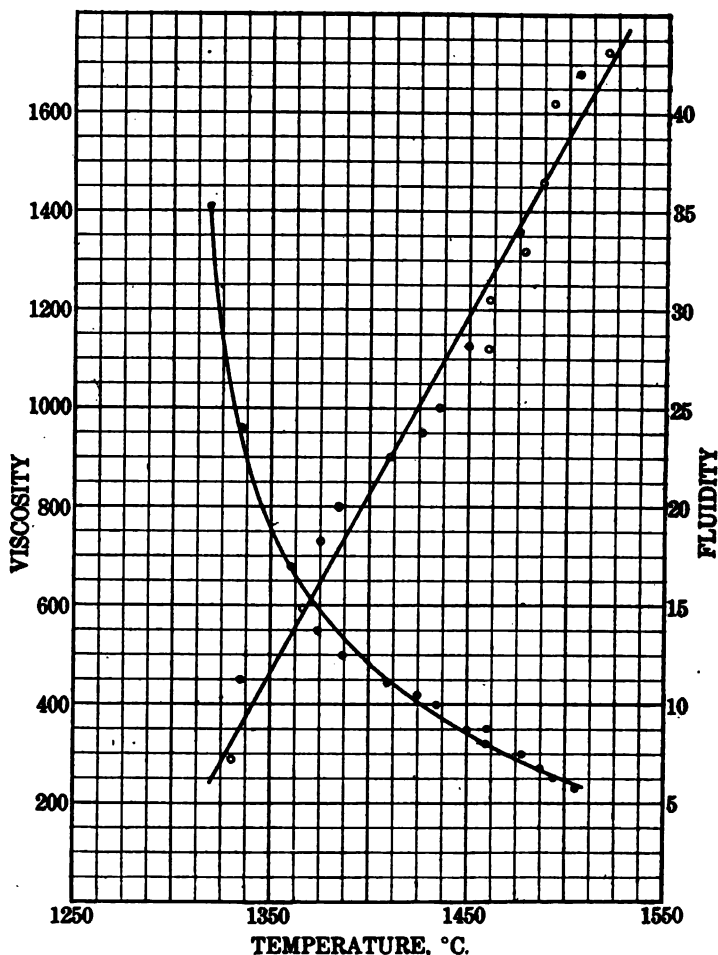


FIG. 7.—RELATIONS OF TEMPERATURE TO FLUIDITY AND VISCOSITY SHOWN IN EXPERIMENTS WITH SLAG No. 23,663.

Plate 3 shows photographs of six fragments of slag melts after removal from the crucible. Samples Nos. 1 and 3 (reading from left to right) are the upper and lower portions of the same melt (slag No. 22,968). Sample No. 2 is a fragment of melt from slag No. 22,954, while sample No. 4, containing the lower end of a viscosity spindle embedded in it,

is a fragment from a duplicate determination on the same slag. In this case the spindle was not removed soon enough, and in attempting to do so the shaft was broken off at the surface of the melt. Sample No. 5 is a fragment of the artificial diopside melt used in calibrating the black-body conditions of the furnace, as described hitherto. The crystalline struc-

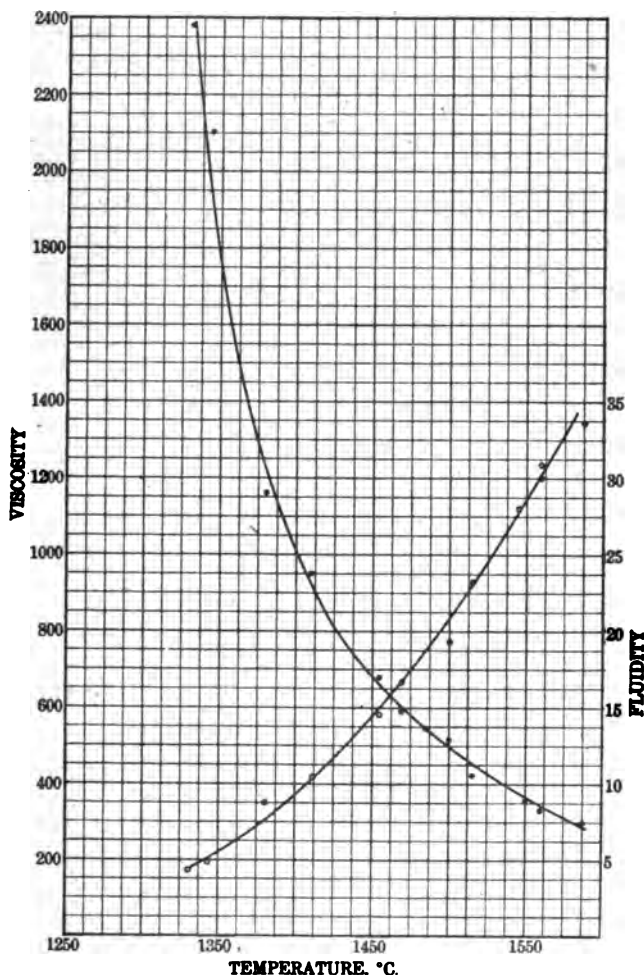


FIG. 8.—RELATIONS OF TEMPERATURE TO FLUIDITY AND VISCOSITY SHOWN IN EXPERIMENTS WITH SLAG No. 22,968.

ture of this melt is easily seen from the photograph. Sample No. 6 (on extreme right of photograph) is a fragment of a synthetic slag melt which does not differ greatly in composition from that of gehlenite, $2\text{CaO} \cdot \text{Al}_2\text{O}_3 \cdot \text{SiO}_2$. The network of large crystals seen in the photograph of this melt are presumably crystals of gehlenite.

Samples 1 to 4 also show the presence of numerous smaller crystals easily visible with the naked eye.

However, the crystalline structure of the cooled slag was not studied systematically in connection with the experiments reported.

The separation of metallic iron, owing to the strongly reducing atmosphere of the furnace and the duration and intimacy of contact between molten slag and the furnace atmosphere, was frequently observed. These metallic globules were found mostly on or near the surface of the



PLATE 3.—FRAGMENTS OF SLAG MELTS AFTER COOLING, SHOWING GENERAL SHAPE AND STRUCTURE.

cooled melt; some were 2 to 3 mm. in diameter. Their formation was only noted in the case of slags abnormally high in iron.

DISCUSSION OF EXPERIMENTAL MEASUREMENTS

The measurements described above give for the first time an accurate idea of the actual magnitude of the viscosity of blast-furnace slags in units which are capable of a physical interpretation, as well as of the rate of change of viscosity of different slags with temperature over a wide range of temperature and viscosity.

If an average slag temperature of $1,500^{\circ}\text{C.}$ is assumed, an examination of the data given in Table 5 discloses the fact that the viscosity of the normal blast-furnace slag as it flows from the cinder notch approximates $250(\text{H}_2\text{O at } 20^{\circ}\text{C.} = 1)$. This represents a viscosity less than that of castor oil at room temperature but greater than that of olive oil.

The viscosity of the eight slags, considered in Table 5, at a temperature of $1,500^{\circ}\text{C.}$ varied from 230 to 480. A synthetic slag upon which measurements have been recently made showed a viscosity of 770 at $1,500^{\circ}\text{C.}$, this slag containing 48 per cent. silica; while reference to Fig. 4 shows that at this temperature the artificial diopside containing 55 per cent. silica had a viscosity of 640. The effect of silica in such amounts upon the viscosity of silicates at temperatures above their melting points is thus clearly demonstrated, the increase in viscosity over that of normal

slags being marked even in the case of diopside which contains no alumina and a high percentage of magnesia.

It would seem that even such high percentages of alumina as are present in the slag designated Lab. No. 22,954, although causing a high initial softening temperature, do not have a very noticeable effect in increasing the viscosity of the melt at high temperatures. This peculiar property of alumina, which may be a quite general characteristic, is probably associated in some manner with the uncertainty of its action within the furnace.

COMPARISON OF VISCOSITY MEASUREMENTS WITH CONE TESTS

It was thought a matter of interest to compare the results of viscosity measurements with cone tests made with corresponding slags. The cone tests were made, as has been previously described, under conditions which converted the sulphide content of the slag to the state of the sulphate. For this reason the comparison is not a strict one. In Table 9 are given in relative order the softening temperatures of six slags and their viscosities at 1,400°, 1,450°, and 1,500° C.

TABLE 9

Slag Lab. No.	Softening Temperature, °C.	Viscosity at		
		1,400°	1,450°	1,500°
22,968	1,279-1,279	1,000	680	480
22,960	1,297-1,300	510	380	280
22,958	1,342-1,342	510	350	260
22,963	1,343-1,360	460	350	280
22,954	1,410-1,410	750	410	230
22,962	1,403-1,443	3,000	480	380

ACKNOWLEDGMENTS

In concluding this account of the investigations carried out by the author, acknowledgment is made of the active interest and 'help' of F. H. Willcox, metallurgical engineer, at whose suggestion and under whose general supervision the work was undertaken. F. G. Cottrell, chief metallurgist, and D. A. Lyon, metallurgist, have given helpful co-operation. The slag analyses were made by F. D. Osgood, junior chemist.

SUMMARY

1. There has been described a modification of the torsion method of Margules which is applicable to the measurement of the viscosity of

blast-furnace slag at high temperatures—the upper limit of temperature of 1,600° C. being imposed by the furnace refractories and not by inherent limitations in the viscosity apparatus.

2. Viscosity values are given for eight typical commercial slags, two synthetic slags, and an artificial diopside.

3. The temperature-viscosity curve generally approximates in form that of the rectangular hyperbola, whereas the temperature-fluidity curve approaches a straight line.

4. The average viscosity at 1,500° C. of eight commercial slags was found to be 301 (H_2O at 20° C. = 1).

5. A refractory slag—that is, a slag possessing a high initial softening temperature—is not necessarily more viscous at high temperatures than a more fusible slag.

6. The theory of the method has been discussed, and the literature on slag viscosity and high temperature viscosity measurements has been reviewed.

7. The application of viscosity data to metallurgical operations has been briefly discussed, as well as the blast-furnace desulphurization process and its relation to slag viscosity.

8. The differences between softening temperature determined by a deformation test, the physical melting point of slags, and the viscosity of slags have been emphasized.

9. The nature of the free-running temperature of slags and the measurement of slag temperature at the furnace have been discussed.

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MARCH, 1917



Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

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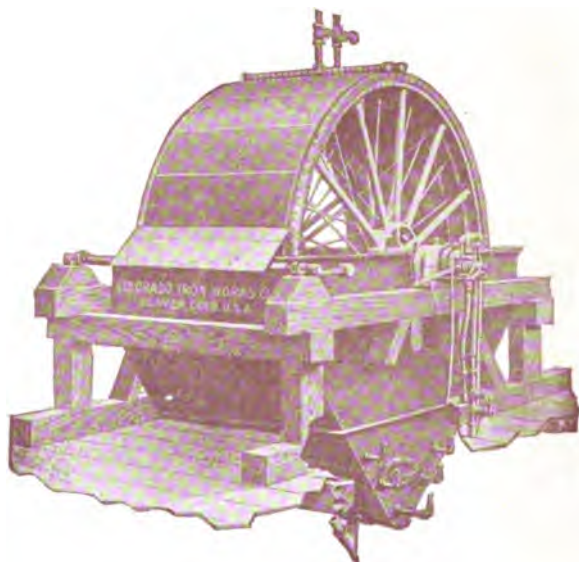
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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 123

MARCH

1917

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTSKY, Publication Manager. Editorial Office, 20 West 39th St., New York, N. Y., BRADLEY STOUTERON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$12 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

TELEGRAM SENT TO PRESIDENT WILSON

Feb. 5, 1917.

To the President,
Executive Mansion,
Washington, D. C.

We, the presidents of the national societies of Civil, Mining, Mechanical and Electrical Engineers and of the United Engineering Society, with a membership of thirty thousand, cordially unite in supporting Congress and the Administration in its stand for freedom and safety on the seas, and we are confident that we represent the membership of the four societies in offering to assist toward the organization of engineers for service to our Country in case of war.

GEORGE H. PEGRAM, President of the
American Society of Civil Engineers.
L. D. RICKETTS, President of the
American Institute of Mining Engineers.
IRA N. HOLLIS, President of the
American Society of Mechanical Engineers.
H. W. BUCK, President of the
American Institute of Electrical Engineers.
CHARLES F. RAND, President of the
United Engineering Society.

PROCEEDINGS OF THE MEETING OF THE BOARD OF DIRECTORS, JAN. 26, 1917

At the meeting of the Board of Directors of the Institute on Jan. 26, 1917, the following actions were taken:

Messrs. A. C. Clark, Lawrence Addicks and G. D. Van Arsdale were appointed Tellers to count ballots for Officers and Directors.

Messrs. T. T. Read, W. S. Harper, and E. J. Hall were appointed Tellers to count ballots on revised spelling.

The Finance Committee's audit for the year 1916 was approved and ordered filed.

The Report of the Finance Committee upon the method of raising this Institute's share of the deficit on the additions to the building was approved and accepted.

Upon the recommendation of the Secretary, increases in the salaries of some of the employees were approved.

The St. Louis Section received an appropriation of \$200 for the year 1917.

The Columbia Section received an appropriation of \$150 for the year 1917.

The Montana Section received an appropriation of \$100 for the year 1917.

The Colorado Section received an appropriation of \$175 for the year 1917.

The Utah Section received an appropriation of \$125 for the year 1917.

The Puget Sound Section received a set of *Transactions* in standard binding.

The International Engineering Congress received a loan of \$300 until the Congress can realize upon some of its assets.

Invitations from cities in the United States for early meetings of the Institute were received and laid upon the table.

The reports of the Joint Conference Committee of National Engineering Societies and of the Pan-American Engineering Committee were presented and ordered filed.

The report of the Committee on Increase of Membership was approved and the suggestion of the Chairman of the Committee was ordered to be sent in writing to all members of the Board of Directors with a request for an expression of opinion thereon.

Upon the recommendation of the Executive Committee of the Committee on Papers and Publications it was

Resolved, that Volume LV of the *Transactions* be known as the "Arizona Volume;" and also it was

Resolved that any member may be permitted to purchase the volumes of *Transactions* for the price of the annual dues of the year in which the *Transactions* are issued, provided such purchases begin with the year next previous to the said member joining the Institute and proceed thence in reverse order.

The report of the Joint Conference Committee on Standards was received and ordered filed, and upon recommendation of the said Committee it was resolved that this Committee be organized as an Engineering Standards Committee and that the representatives of this Institute thereon be the present representatives on the Joint Conference Com-

mittee on Engineering Standards consisting of Messrs. James R. Finlay, J. W. Richards, and George C. Stone.

The recommendations of the Committee on Membership were approved and the members, associates and junior members were elected.

The dues were suspended of two members of the Institute who are in active duty at the front.

The resignations of three members were accepted, and twelve members who paid back dues were reinstated.

The resolution regarding the Belgian Kiddies, Limited, given in full below, was then passed.

The meeting then adjourned.

BELGIAN KIDDIES, LTD.

Minute of the Board of Directors, Jan. 26, 1917

In the name of all the members of the American Institute of Mining Engineers, this Board extends to Mr. Herbert C. Hoover, an honored Vice-President of the Institute, on the occasion of his present temporary return to his native land, the cordial welcome of his colleagues, and their congratulations upon the noble work splendidly done by the Commission for Relief in Belgium of which he is the head.

It is understood that a leading purpose of Mr. Hoover's visit to America at this time is the collection of additional funds, to be administered by his Commission in the care of the destitute children of Belgium. To this enterprise endorsed by both the belligerent parties, as well as inspired by the purest and tenderest sentiments of humanity, contributions are solicited from engineers, especially through an organization known as the "Belgian Kiddies Limited," which this Board, heartily approving its object and method, recommends to the consideration of the members of the Institute.

On motion duly seconded it was unanimously

VOTED.—That this minute be spread upon the record of the Board, and published in the *Bulletin*, and that a copy thereof be transmitted to Mr. Hoover, and to the San Francisco Section of the Institute.

MR. HOOVER AND HIS WORK IN BELGIUM

Herbert C. Hoover, who became a member of this Institute in 1896 and who is now one of its Vice-Presidents, is the man who in 1914 extended a helping hand to Americans stranded in Europe at the outbreak of the war, and who made it possible for many such travelers to return safely to their homes. When his countrymen no longer needed his services, he turned to the Belgians, whose need of food had been presented to him. Engineers have been among his assistants from the first, and it is because we believe that members of his own profession will still be particularly interested in his work that we thus mention it in the *Bulletin*.

A Minute adopted by the Board of Directors of the Institute, in which the Belgian Kiddies, Ltd., is recommended to the consideration of the members, immediately follows the Proceedings of the Meeting of the Board of Directors.

With the thought that members of the Institute may be glad to read Mr. Hoover's own words in regard to his work, we print in full the speech

that he delivered at the dinner given in his honor by the Rocky Mountain Club on Jan. 29, and precede it by the address of John Hays Hammond, Past-President of this Institute and President of the Rocky Mountain Club, who presided.

PRESIDENT JOHN HAYS HAMMOND: Our Honored Guest, Gentlemen of the Rocky Mountain Club and Friends: At a meeting of the Board of Governors of the Rocky Mountain Club some few weeks ago it was decided to take steps to build a clubhouse sometime in the near future. This course was regarded by the Governors of the Club not only as justified by existing conditions, but as highly desirable, and indeed, more or less a matter of urgency.

Within the last few days, however, one of our distinguished members, Mr. Herbert C. Hoover, submitted for their consideration a measure which was deemed by the Governors of the Club of far greater urgency than the immediate erection of a club building. I refer to Mr. Hoover's appeal to the Club to coöperate with him in the noble work he is doing to relieve destitution in Belgium.

While the Board of Governors have not in any way changed their opinion as to the desirability and the feasibility of erecting a clubhouse sometime in the near future, they have nevertheless unanimously resolved to defer action in this regard, to give precedence over all other considerations to this great cause of relieving suffering humanity. The Governors of the Rocky Mountain Club have accordingly unanimously resolved to coöperate with Mr. Hoover in this laudable work.

In pursuance of that resolution there will be appointed Committees from New York and the Western States, to solicit funds under the auspices of what will be called the "Rocky Mountain Club-Hoover Fund for Relief in Belgium."

We have every reason to hope for a generous response to our appeal.

I am sure that when they have heard Mr. Hoover—and there is none better qualified to represent the cause of Belgium—describe the deplorable condition of that afflicted land, the members of the Rocky Mountain Club will unqualifiedly endorse the action of its Board of Governors.

Our fellow-member, Mr. Alexander Hemphill, who has been associated with Mr. Hoover in the Belgian Relief Work, will have the honor of introducing our distinguished guest of the evening, but I cannot refrain, as a fellow-member of the Rocky Mountain Club, as a fellow-engineer, and as a fellow-citizen, from expressing a note of appreciation of the credit that Mr. Hoover has conferred upon the Rocky Mountain Club, the mining profession, and our Nation, by the high order of ability, the unselfish zeal and tireless energy he has displayed in the discharge of the great trust reposed in him.

Mr. Hemphill then introduced Mr. Hoover, who was greeted with prolonged applause, members and guests standing.

MR. HERBERT C. HOOVER: Mr. President and Fellow-Members: I feel greatly embarrassed and greatly moved by the words I have heard and by the action of the Club. It was a response which I received within twenty-four hours after arriving here, in a mood of a great deal of discouragement. I have to beg for forbearance because I am so poor at making public addresses that I cannot express that appreciation which is in my heart.

I feel that I have to justify to you the action which your Governors have taken by giving you some description and some explanation of what we have done, and the work we are trying to do.

I always find three difficulties in talking about Belgian Relief—four in fact. One is that I have never delivered a public address on the subject. My entire public appearances have been in combative moods where I have had to defend the Belgian Relief, and I can assure you that I have a torrent of words on these occasions. With friends, it is more difficult. If some one would get up and assault this enterprise, I assure you that I could do pretty well.

My next difficulty is one of some delicacy as to the position which I hold, that it is vitally necessary for us to be absolutely neutral, not only in deed, but in word, lest we should, by some act of ours, increase the sufferings of ten millions of people.

A third difficulty I have is to visualize to men who have not been in Europe since the war began the actual situation which exists in an occupied territory. Perhaps, if you would endeavor to imagine New York and about five adjoining counties occupied by an enemy army, blockaded from without, and surrounded with a wall of steel, the

normal flow of food, seventy per cent. of which food normally comes from without, stopped, you would awaken within thirty-six hours to find your markets empty, and your bakeries stopped.

Add to this that your railways would be taken over for military purposes; that your telephones and telegraphs would be suppressed; that your newspapers would be prohibited; that every form of communication and intellectual life would cease instantly; that you could not move outside your own wards and your own villages; that you could not assemble except by special permission; that every street corner and every cross road would be occupied by a sentry; that the normal feeling of security which every man has, that he has at least an appeal to justice, is gone, submerged in the fact that he is subject only to an army—even then you have an inadequate picture because it is almost impossible to convey to you the psychology of a people in such a stress.

The feeling that the food supply of the community, the food supply to the individual, may cease at any moment; that your women and children are in imminent jeopardy, and the feeling of every thinking man that disturbance by the population only means blood in the streets; that there is no possible salvation or solution; a population which shivers with rumors—it goes beyond ability to describe.

That has been the situation of Belgium and Northern France with 10,000,000 of people, for nearly two years, with one exception, and that exception was the Commission for Relief. The Commission for Relief has not only been the salvation of their food supply, but has been their protection in their despair.

Now you will say at once, and I will answer it at once, that it is the duty of the occupying army to provision the civilian population. I commit no indiscretion when I repeat to you the official statement by the German Government on the one side and the British Government on the other. The Germans say that the shortage in food supply in Belgium is due to a blockade; the ports are open except for the Allied Navies; the blockade is illegal and is inhumane; that as a result of the blockade they have not a sufficient supply of food of their own; that they invite relief and that they will support it in Governmental measures, and that in the name of humanity it ought to be done. The British Government, on the other hand, says that the prime responsibility and the prime wrong was the occupation of Belgium; that, therefore, the responsibility rests on the occupying army, and that in tradition and International Law it is the obligation of an occupying army to provision the civilian population. The Englishman goes further than that and contends that the situation is this: The British people are fighting for the independence of Belgium and Belgium comprises not only her territory but her people; it is, therefore, useless to reestablish the independence of Belgium if she is to be found an empty husk, and they will support the relief not only morally but financially.

Now it is not for me to say what is right and what is wrong in these various contentions. It is not the right of any American to refuse to intervene to save a mass of 10,000,000 people from being ground between these millstones. Every thinking man in Europe knew that this situation must ensue and ensue quickly after the occupation.

The Belgian people themselves sent up the first plea to the Americans for help, and through the American Ambassadors in London, Brussels, and Berlin we opened the gate through this wall of steel, and we have tried to defend and have defended it since. These gentlemen, in their official capacities, could do no more than intervene for us, and they must leave it to the many organizations to carry it forward. Under International agreements which surround the matter and which are recognized by both of the belligerents, we are under the control of both.

Initially we appealed to the world for charity, as we had practically no resources except perhaps some banking arrangements which we had created. We had a generous response to our appeal. When we started out, three to four million dollars a month seemed to be compassable by the charity of the world. None of us anticipated that more than three or four months of such a situation could exist. Within two or three months it was well understood that this was not a war of days, but would be a war of months, or perhaps years; that there was no solution to it except Governmental assistance, and that it required wider and broader charity than had ever been known.

We built up Governmental assistance, first, through the Belgian Government at Havre from their slender resources, and then through the Allied Governments. We have added to our responsibilities, however, in addition to the Belgian civilian population, the civilian population of Northern France. So that relief has grown in organization from a moderate charitable enterprise to one, I think, without a parallel in history.

Up until the end of January we had spent over two hundred and fifty millions of

dollars; we had imported into Belgium two and one-half million tons of concentrated food supplies, and we are now issuing over two and one-half million loaves of bread every month, and twenty-five to thirty million pounds of fats and distributing five million tins of condensed milk—but I won't worry you further with these statistics.

It will occur to every business man that he ought to know something of this organization. Everyone has a right to know whether it is efficient; whether it is honest, and whether it accomplishes its purpose. Now there have been three tenets in this organization: the first is de-centralization; the second, voluntary service, and the third, high ideals.

We realized from the start that it was necessary to have the coöperation of every intelligent man in Belgium and Northern France, and the assistance of every commercial house into whose realm we might tread. We have never appealed for assistance in any quarter, either to banks or commercial houses, or to experts of any sort, that we have not received the fullest and most skillful advice.

We have built up by degrees some four or five thousand committees recruiting charity throughout the world, and we have recruited about thirty millions of dollars, of which about nine millions came from this country.

Our central organization consists of sub-committees here in New York, with my friends, Mr. Honnold, as Director, and Mr. Hemphill, as Treasurer; a committee in London, another in Rotterdam and another in Brussels. We coöperate again with the large committees in Belgium.

The basis of our distribution organization lies in the creation of a committee in every commune, some 4,000 in all, each of which assists in the distribution of the food supply to the population. In addition, we have regional committees and central committees in all the important cities, over each of which is a general manager.

Thus you will see we have built up about 8,000 sub-committees and have facilities for the collection of money on the one side and its distribution on the other.

The distribution is the most interesting end and the one which is confronted with the greatest difficulties. I cannot speak too highly of the devotion of the Belgian and French people to this cause. I cannot speak of the devotion those people have shown without emotion. For over two years there has been a constant consultation of committees—not any bi-weekly meetings. It has been a work of most exacting character, lasting from early morning until late at night. The problems have been enormous.

If one is going to provision an entire nation, with the slender resources we could command, the first measure is the repression of the food consumption of the country by at least fifty per cent. The human family can live on that proportion of its normal food, and our only hope of success was to do it. When you exert a repression of that kind on food supply you must get absolute justice in distribution, lest, if one man gets too much, it means some other man must starve.

If you couple the difficulties of an organization of that kind with the normal difficulties of shipping in these times, with the difficulties of financing in an organization that has never yet seen sixty days of certainty ahead, that has never seen the day that its contracts did not exceed its assets by from five to twenty-five million dollars; if you couple that with the incidental tragedies of the loss of six ships within a single week; with the difficulties of dealing with a people of so different a national character as the Americans in coöperation with Belgians and French, whose whole mental attitude is so different from ours, with the difficulties of dealing with an occupying army which, by necessity in any army, must be arbitrary in its methods, the difficulties of protecting the native food supply, which was as vital to us as our imports; the difficulties of negotiating with practically every Government in Europe in a matter which to them was entirely a side-issue, even then you could have but a dim picture of Belgian Relief because no one who has not engaged in the work and seen its vital importance can realize the terror that goes through the Relief at every slight break, for fear it may fail.

Now relief work consists not alone in the distribution of food, but in the handling of destitution. There is in Belgium about fifty per cent. of unemployment, and about seventy per cent. in the North of France. These people are absolutely destitute. They must have the means with which to obtain food; it is not enough to give them rations, they must have the means with which to buy their own production and their own local food supply. They must be clothed; they must be housed and they must be kept warm. All of this entails a vast organization of committees and sub-committees horizontally and vertically through the population, endeavoring constantly to remedy one weakness after another, because it is a situation of constant degeneration. The shortage of fertilizer, the shortage in seeds, the shortage in labor and cattle necessarily make each succeeding harvest poorer and poorer.

So that every month or every few months there develops some new weakness; some class begins to show the signs of under-nourishment. We had such a case about five months ago, when we first heard of glandular tuberculosis among the adolescent children. We brought an American physician over and he cooperated with the very skillful Belgian physicians, and together they developed the fact that this was a clean case of under-nourishment; a shortage of the necessary food supplies. We always try to apply a rough and ready remedy, and with our Belgian colleagues and the French we decided to install public feedings in the schools, and to give one meal a day as a supplement to the normal family ration; to give it direct to the children so as to be sure of its going to the right place.

That service required an expenditure of two and one-half million dollars a month, of which one-half was for internal supplies, and which many families have themselves advanced from their own slender resources. The million and a quarter for the imported portion of that food supply amounted to about \$1 per child per month. We appealed to the American public to assume that responsibility, and it failed. We have not received the response which we should have. This failure is one which appears to us in a probably larger measure than it does to you.

The Belgians have come to look upon Americans as their sole saviours; they look upon the American flag as the flag that is fighting to protect them. Within six weeks time I have visited in the slums of Brussels where 1,500 children were sitting down to their one meal, and when they saw me those youngsters arose and warbled the first stanza of the Star Spangled Banner in French. Now I, knowing that that food supply was not American, could feel nothing but shame.

Europe has begun to take stock of this Relief Commission. We started off with high promises as to American support and American ideals; we have rested on the belief that we had the undoubted backing of the American people in our endeavor to keep this gate open. Oftentimes we have had to defend the portals of the enclosure, and we have always used as our last and final weapon that interference with the Belgian Relief would offend the sensibilities of the American people more deeply than anything that has happened in this war. And yet Belgian Relief is being paid for with foreign money.

But of more importance than this purely moral question is the fact that our finances are in great jeopardy. As time goes on, the situation becomes more desperate in Europe; our necessities grow greater; it is growing more and more difficult to see our way through. Today our budget calls for about \$18,000,000 to \$19,000,000 a month, if we are not to reduce the already short food supply of these people.

We are receiving \$14,000,000 a month from the Allied Governments. There is no hope of any increase in that, and there is always the possibility that at some stage we may lose it altogether. In any event, we are from \$3,000,000 to \$5,000,000 a month short. We have to raise this money by public charity and the few financial concessions which we are able to secure, and we thought that perhaps the American people would take off our shoulders the burden of those 1,250,000 children.

This failure is crushing to our national pride; it undermines our ability to defend these people, and what is more, it is fraught with the utmost suffering, to a point which should touch the heart of every American deepest.

I feel that the failure is perhaps partially or largely due to ourselves. I can assure you that my colleagues and myself have a sufficient labor in maintaining the detailed aspects of this organization; in buying \$18,000,000 worth of food and distributing it to the Belgian people, and a few hundred other things which tax the capacity of most men.

We have not the time nor do we have the capacity to effectuate an organization in the United States which would give us the response that I believe the American people are willing to give and are capable of giving, and to such organizations as yours we appeal to carry on the work in our behalf.

I make this appeal here with a certain feeling of confidence because I know I am appealing to men largely from my own section of the United States. Many of you have been my friends for years; many of you are of my own profession—you are miners. Sometime ago, in a moment of desperation, I assessed the mines in Australia where I had a connection for many years. I told each one what I thought they ought to do. I received a total, within, I think, about two months, of \$750,000, from a country already combed to the bottom for relief and distress work.

We appealed to the miners in Johannesburg, and the laborers in the mines gave ten per cent. of their wages, and the owners duplicated the amount.

I feel that in this stress the American miner might also give some help, and it gave me a feeling of peculiar satisfaction that the first gentleman who rang me up on the

telephone when I arrived here was a miner, who said, "My assessment is about \$100,000."

Now, gentlemen, this matter is one of more importance even than the feeding of 1,250,000 children, as large as that may be. This Relief has come to be America's greatest exhibit in Europe. We are undertaking to do it with faith in the American people; we are undertaking to do it knowing that the American people will help in an American way. We want to give a demonstration of that great strain of humanity which we know runs through our people because we know the character of the people that make up this Republic.

Now, gentlemen, if we succeed it will be because we have received the support of the American people instead of being forgotten.

UNITED ENGINEERING SOCIETY

Report of President

The important fact of the year 1916 is that on July 25 contracts were executed by which the American Society of Civil Engineers became an additional Founder Society and arranged to make its permanent home in our building. The contracts provide for the construction of three additional stories to the building, the American Society to pay the cost up to \$250,000, this being considered equivalent to what each other Founder Society had contributed, the American Society to participate in the original Carnegie gift and have an equal share in the property with each other Founder society.

The construction of the addition to the building is under way, \$62,-525.04 having been expended thereon in 1916.

It has been found that, due to war conditions and the advanced price of materials, the cost of the addition will be \$300,000. The founder societies have been requested to assume the extra \$50,000, one quarter each.

This work is in charge of a Building Committee consisting of H. H. Barnes, Jr., Chairman, E. G. Spilsbury, Charles Warren Hunt and Charles F. Rand.

At the request of the Founder Societies, important alterations were made in the lecture halls on the fifth floor of the building to make the same suitable for social functions of the societies. The cost, \$6,642.08, was advanced by the United Engineering Society but is to be paid eventually by the Founders.

The Library of the Civil Engineers is being merged with the Library of the United Engineering Society and the other founder societies.

At the present time the membership of the four founder societies is 29,000 and of associate societies 23,000, so that a total of 52,000 engineers now have their headquarters in our building.

The value of the real estate now owned by this Society is \$1,647,171.16. This sum will be increased at the end of 1917 by the amount of the cost of the addition to the building.

The income of the Society during 1916 was	\$53,062.03
The expenditure was	44,316.20
Gain for the year	8,745.83
The surplus at the close of last year was	7,307.42
Total	16,053.25
From this \$10,000 has been taken for the Depreciation and Renewal Fund	10,000.00
Leaving a surplus at present of	\$6,053.25

Funds for the benefit of the Library were obtained during the year from the following sources:

Contributed by American Institute of Mining Engineers . . .	\$4,000.00
Contributed by American Society of Mechanical Engineers .	4,000.00
Contributed by American Institute of Electrical Engineers .	4,000.00
Contributed by American Society of Civil Engineers	1,000.00
Contributed by United Engineering Society	2,542.33
Income from Endowment Fund	600.00
Income from sale of books	121.30
An unexpended balance from previous year	1,029.73
Gross income from searches	5,382.98

Total income	\$22,676.34
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The Library expenses have been as follows:

Library books purchased	\$2,340.88
Library binding expense	1,442.55
Library supplies and miscellaneous expense	1,379.38
Library salaries	10,923.81
Library photostat	1,091.02
Library new lighting fixtures	132.47
Library searches expense	5,366.23

	\$22,676.34
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Dr. James Douglas, who started our Library Endowment Fund with a gift of \$5,000, has added \$95,000 thereto. The total of the fund is now \$102,559.70. An effort is being made to materially increase this fund as the Library requires the income of a million dollars for its proposed development.

The securities in the Engineering Foundation Fund were sold and the proceeds re-invested to produce a higher income, as shown in detail in the treasurer's report. The Fund now amounts to \$203,374.80.

The General Reserve Fund remains unchanged at	\$10,000.00
The Depreciation and Renewal Fund is now	71,456.12
having been increased \$2,610.45 from interest earned during the year and \$10,000 by direct appropriation from surplus.	
The Surplus Dec. 31, 1916 is	\$6,053.25

In the report of the treasurer will be found many interesting details and in the report of the Library Board will be shown the extension and increasing importance of the Library which is now believed to be the most complete engineering collection in existence.

To Professor Frederick R. Hutton, who has acted as secretary from the organization of the Society in 1903 to the present time, and to Joseph Struthers, treasurer for eight years, the trustees owe and acknowledge a deep indebtedness.

The trustees also desire to express most sincere thanks to H. H. Barnes, Jr., Chairman of The Finance Committee and Chairman of the Building Committee, for very valuable services rendered:

The year 1916 has closed with the affairs of the United Engineering Society in satisfactory condition.

Respectfully submitted

by Direction of the Board of Trustees of
United Engineering Society,

CHARLES F. RAND, *President.*

JOHN FRITZ MEDAL AWARD

The John Fritz Medal for the year 1917 has been awarded to Professor Henry Marion Howe for his investigations in metallurgy, especially in the metallography of iron and steel. *Record of the John Fritz Medal.*

The John Fritz Medal Board of Award has published a book giving a history of the Medal, the Rules of Award, biographical notices of the medallists illustrated with portraits, etc.

This beautiful book, designed, engraved and printed by Bartlett Orr Press, New York, may be obtained by addressing the Secretary of John Fritz Medal Board of Award, Engineering Societies' Building, New York. Price \$3 per copy.

AMERICAN SOCIETY OF CIVIL ENGINEERS HOLD SESSION IN ENGINEERING SOCIETIES' BUILDING

One session of the annual meeting of the American Society of Civil Engineers—the first since this society became a Founder society with representation on the Board of the United Engineering Society—was held on Jan. 17, 1917, in the Engineering Societies' Building. Mr. Charles F. Rand, Past-President of this Institute, and President of the United Engineering Society, welcomed the new Founder society in the following words:

"As one who has looked forward very earnestly to this day, and who has made a modest effort in favor of the new arrangements, it is a peculiar satisfaction to be the representative of the Trustees of the United Engineering Society and to welcome you on the occasion of your first meeting in this building in which you now own a quarter interest.

"As this is your annual meeting, I cannot take your time for extended remarks, yet it seems proper that I should call attention to a few facts respecting the United Engineering Society, with which you are now identified.

"The United Engineering Society exists for the purpose of holding the legal title to certain of the property of the societies of Civil, Mining, Mechanical and Electrical Engineers—and to act for them in certain matters.

"The value of its property, including real estate, the Library and the Reserve and Endowment funds, exceeds two million dollars, all free and clear.

"The total membership of the four Founder societies is 29,000 and the membership of associate societies is 23,000, making a grand total of 52,000 engineers who now have headquarters in this building.

"As is well known, the United Engineering Society stands not over but under the Founder Societies. It is organized to perform for the Founders certain specific acts which are governed by contracts. There is no merger of societies, each retains its individuality. The United Engineering Society enables the Founders to coöperate conveniently.

"The Engineering Foundation is a fund of \$200,000 belonging to the United Engineering Society, established for engineering research with a gift from Ambrose Swasey. It is hoped that this fund will soon be increased many times.

"The great library now includes that of the Civil Engineers and contains approximately 75,000 bound volumes and 50,000 unbound. It is believed to be the largest engineering collection in the world. Plans have been made for the development of the library which contemplate the expenditure of \$50,000 annually. At present the societies can afford to spend only \$20,000 per year. The library has an endowment of \$100,000. This we hope may soon amount to a million, as the library needs the income from that sum.

"The United Engineering Society has been established thirteen years. It was originally developed through the efforts of the societies of Mining, Mechanical and Electrical Engineers. To this organization the American Society of Civil Engineers has joined its strength and prestige, and takes a leading part.

"Your new home is being rapidly prepared for you and will be ready during the current year. The six months of intimate association and cordial coöperation we have already had with the officers of your society indicate that a move has been taken that is of great value and importance to engineers.

"Gentlemen—Welcome to your own home."

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

Members and guests who called at Institute headquarters during the period Jan. 10, 1917 to Feb. 10, 1917:

Victor C. Alderson, Denver, Colo.
 Arthur W. Allen, New York, N. Y.
 George S. Blair, Pittsburgh, Pa.
 F. J. Brulé, Toronto, Canada.
 W. M. Corse, Buffalo, N. Y.
 John J. Croston, Cullasaja, N. C.
 W. A. Deane, Waxhaw, N. C.
 P. H. Dudley, New York, N. Y.
 J. D. Gilchrist, Denver, Colo.
 L. Hoffman, Pottersville, N. J.
 Guy R. Johnson, Florence, Wis.
 W. McA. Johnson, Hartford, Conn.
 John S. Kennedy, Philadelphia, Pa.

G. Komar, New York, N. Y.
 H. H. Lauer, Allentown, Pa.
 Frederick MacCoy, Antofagasta, Chile.
 E. P. Mathewson, Toronto, Canada.
 Douglas Muir, San Antonio, Tex.
 F. W. Osborn, Ajo, Ariz.
 Mortimer A. Sears, Washington, D. C.
 W. H. Selden, Jr., Iron River, Mich.
 W. H. Staver, Yonkers, N. Y.
 Ralph H. Sweetser, Easton, Pa.
 Benjamin F. Tibby, Salt Lake City, Utah.
 Carl J. Trauerman, Butte, Mont.
 H. C. Wilmot, Arequipa, Peru.

D. C. Bard has become a member of the geological department of the Chicago, Milwaukee & St. Paul R. R., with headquarters at Seattle.

Jay A. Carpenter has resigned his position as metallurgist for the West End Consolidated at Tonopah and has been appointed superintendent of the Nevada Packard mines at Rochester.

Ellard W. Carson has been appointed superintendent of the Oceanic Quicksilver Mine, Cambria, Cal.

H. H. Colley has been made superintendent of the Old Dominion mine and smelter at Globe, Ariz.

Walter Douglas has been appointed member of the Committee on Mining Law Revision in place of Dr. James Douglas.

C. A. Filteau has resigned as manager of the St. Lawrence Talc Co., Natural Springs, N. Y., and is now manager of the National Mines, Ltd., Cobalt, Ontario.

Robert Hawxhurst, Jr., has been made manager of the Eden Min. Co., in Nicaragua.

W. R. Ingalls has been appointed by the Director of the U. S. Bureau of Mines as chairman of the Committee of Consulting Engineers on Mining Law Revision.

Mark R. Lamb has resigned as manager for the Allis-Chalmers Mfg. Co., in South America.

Robert Linton has been appointed first vice-president and assistant to the president of the North Butte Min. Co., with headquarters at New York.

J. B. Moore has accepted the position of mine manager for the Tigre Min. Co., Esqueda, Son., Mexico.

William Motherwell has opened an office as consulting flotation engineer at Nelson, B. C.

R. K. Stockwell, formerly superintendent of construction for the Braden Copper Co., Rancagua, Chile, has become the representative of the Robins Conveying Belt Co., Salt Lake City, Utah.

Curtis Pigott has been promoted to the assistant superintendency of the United States Smelter at Midvale, Utah.

J. Harold McCreery, formerly associated with Fred Ophuls, consulting engineer, 50 Church St., New York, has been elected secretary and

treasurer of the new firm of Ophuls, Hill & McCreery, Inc., consulting engineers, 112 W. 42nd St., New York. Mr. McCreery will devote his time to the mining and mechanical branches of the firm. The firm will also specialize in appraisals of industrial power plants as well as ice and refrigerating plants.

R. H. Sweetser, formerly president of the Thomas Iron Co., Hoken-daqua, Pa., has been appointed works manager of the Columbus Iron & Steel Co., Columbus, Ohio.

W. M. Weigel has resigned as associate professor of mining at Pennsylvania State College and has accepted the position of superintendent of concentration for the International Molybdenum Co., Ltd., Renfrew, Ont., Canada.

W. W. Wishon has been appointed consulting engineer of the Big Casino Min. Co., Searchlight, Nev.

Herbert C. Woolmer has resigned his position with the Spassky and Atbasar copper companies.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Member, Graduate Colorado School of Mines. Four years' experience in West and Northwest. Two years with American Smelting and Refining Co. Wishes to join field party on geological work, mining or areal, where opportunity will be given to be of direct assistance to chief of party in solution of field problems. Age, 26. Unmarried. At liberty June 1. No. 342.

Member, technical graduate, aged 30, with 7 years' experience in concentrating and mining, desires a position of authority with reliable company. At present employed. No. 343.

Member. Ambitious, energetic, tactful engineer, college graduate, married, aged 30, desires location with a large coal mining organization. Seven years with present employers in middle west; has had broad experience in the past eight years on examination, development and construction; successful record in handling men. Position must carry responsible work and a future. Will go anywhere for proper opportunity. Exceptional references as to reliability, character and capacity for work. No. 344.

LOCAL SECTION NEWS

PUGET SOUND SECTION

SIMON H. ASH, *Chairman*,I. F. LAUCKS, *Vice-Chairman*,CHARLES SIMENSTAD, *Sec.-Treas.*, 425 Lyon Building, Seattle, Wash.

GLENVILLE A. COLLINS,

JOHN N. POTT.

The Puget Sound Section met at the Arctic Club, Oct. 28, 1916, for the annual meeting.

Those present were: Chairman Glenville A. Collins, Vice-Chairman H. L. Manley, Sec.-Treas. Amos Slater, Chester F. Lee, R. P. Tarr, G. E. Rockefeller, Simon Ash, Geo. W. Evans, Joseph Daniels, I. F. Laucks, F. C. Greene, Darsie C. Bard, Marcel Daly, Milnor Roberts, F. A. Hill, Percy E. Wright, and as guest, Mr. Coates of British Columbia.

After dinner had been served, Chairman Collins called the meeting to order. The reading of the minutes of the last meeting were suspended upon request of the Secretary.

The Secretary read the report from the Committee on the Associated Engineering Societies of Seattle. Upon motion, duly seconded, the report was adopted and placed on file.

Mr. Lee, Chairman of the Committee on the Mining Men's Convention to be held this coming winter, gave his report orally, and upon motion the committee was continued.

The report of the Secretary-Treasurer was read, received, and placed on file.

The officers named above were elected for the year 1916-1917.

Mr. Collins then invited the new officers to take charge of the meeting. Mr. Ash, the new Chairman, made a few remarks and called for new business.

Mr. Lee then proposed that the meeting be thrown open for a discussion of the Mining Men's Convention. A thorough discussion was entered into by all the members on the ways and means of working on this convention. Mr. Collins made a motion that the Committee add to their number any number, as they see fit, to assist in the work. This was seconded by Mr. Hill and carried. Mr. Collins and Mr. Greene were asked to join the Committee.

Mr. Lee made motion that the Committee be allowed to draw upon the Treasurer to the amount of \$25.00 to cover expense; this was seconded by Mr. Hill and carried.

Mr. Slater brought to the attention of the members the appointment of two councilors to the Associated Engineering Societies; upon motion, duly seconded, the Chairman was instructed to make the appointment. The Chairman then appointed Mr. Chester F. Lee and Mr. George W. Evans as councilors.

Mr. Collins suggested that the Chairman call upon the Chamber of Commerce and consult with them regarding the Mining Convention and publicity; Evans, Wright and Ash appointed.

No further business coming before the meeting, the meeting adjourned.

AMOS SLATER, *Acting Secretary*.

The December Meeting of the Puget Sound Section was held at the Arctic Club on Dec. 9, 1916. Those present were: Percy E. Wright, Joseph Daniels, I. N. Daly, Milnor Roberts, T. M. Daulton, Chas. E. Weaver, M. Daly, Geo. Watkin Evans, S. H. Ash, M. C. Butler, Amos Slater, Chas. Simenstad. B. W. Bristnall and C. P. Dexter were guests.

After dinner had been served, Chairman Ash called the meeting to order. Minutes of previous meeting were read, corrected and approved.

The Committee on Mining Men's Convention reported that it believed the time was too short to fully prepare for the Convention to take place during the coming year. After discussion, it was moved and seconded that the Convention be postponed. Motion carried.

It was moved and seconded that the Secretary be instructed to write Mr. Van H. Manning, Director of the U. S. Bureau of Mines, expressing appreciation of the establishment of a metallurgical station at Seattle, and offering the support of our local section. Motion carried.

The business meeting then adjourned, and the members and guests foregathered in the main dining hall of the Club, where Mr. Geo. Watkin Evans delivered a very interesting illustrated lecture on the Ground Hog Anthracite Coal Fields of British Columbia.

CHAS. SIMENSTAD, *Secretary*.

MONTANA SECTION

W. C. SIDERFIN, *Chairman*,

OSCAR ROHN, *Vice-Chairman*,

E. B. YOUNG, *Sec.-Treas.*, 526 Hennesey Bldg., Butte, Mont.

C. D. DEMOND,

F. W. BACORN,

The Montana Section of the American Institute of Mining Engineers enjoyed a dinner at the Silver Bow Club Friday evening, February 2, 1917, on the occasion of their annual meeting for election of officers. The new executive committee of the Section is as shown above:

Chairman J. L. Bruce, of the retiring officers, called the meeting to order and talked for a few minutes on membership. He called attention to the activities of the Naval Consulting Board, which has recently made an inventory of 30,000 business organizations for purposes of national preparedness, and mentioned the attendance at military engineering lectures by the Institute.

Mr. Goodale spoke at some length regarding contributions to the relief of Belgian children. His special appeal was to the Institute members in behalf of their colleague, H. C. Hoover, who is in charge of the relief work in Belgium.

The business of the meeting was promptly disposed of and a committee appointed to consider the printing of a booklet concerning the Montana Section of the Institute.

A very interesting paper was given by Mr. L. D. Frink, superintendent of the Speculator Mine, upon some of the applications of mine ventilation in Butte. He had a number of exhibits of special prepared canvas tubing, joints, and other devices in use at the Speculator. A large number of flash-light photographs in the mine showing the operation of the ventilation system added to the interest of his talk.

Mr. F. A. Linforth, of the Anaconda Geological Staff, exhibited a number of maps and models to show a few points in the graphic interpretation of geological problems, and his talk was of very great interest. Discussion by Mr. Bunshee of the Anaconda company brought out still further the value of such work to the mining department.

Mr. Dunshee also gave an account of his recent trip to the Hawaiian Islands.

Sixty-two members and guests attended the meeting.

WALTER E. GABY, *Secretary*, 1916.

SAN FRANCISCO SECTION

W. H. SHOCKLEY, *Chairman*, FRANK H. PROBERT, *Vice-Chairman*,
C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post Street,
San Francisco, Cal.
J. F. NEWSOM, C. E. BRAYTON.

The annual meeting and dinner of the San Francisco Section was held at the Engineers' Club on Tuesday evening, Jan. 9, 1917. Dr. Ricketts, President of the Institute, was the guest of honor of the Section.

Dr. Ricketts in his address called attention to the rapidly increasing membership of the Institute. He pointed out the continuous need for good papers and said he was proud of the papers that are now being submitted. The value of the Local Sections was pointed out and intercourse between Sections was urged. The great work done by the Joint Committees of the four National Engineering Societies in New York was explained and Dr. Ricketts urged increased activity of the Mining Engineer in the Conservation of the Mineral Resources of the United States.

C. W. Merrill then voiced a plea for the recognition by the engineer of his civic duty and of his duty to the community as a whole.

A. C. Lawson, Gardner Williams and G. H. Clevenger discussed briefly the labor problem, emphasizing its importance to the engineer. Prof. Newsom said that he felt Conservation to be our most important problem and that the engineering societies should bring this before the public in the proper way.

The officers named above were elected for the current year.

Votes of thanks to Dr. Ricketts for the honor he had done the Section in attending our meeting, and to T. A. Rickard for his able Chairmanship during the past year, were unanimous.

The meeting then adjourned.

C. E. GRUNSKY, JR., *Secretary*.

AFFILIATED STUDENT SOCIETIES

The Mining and Metallurgical Club of the University of Toronto held a very successful meeting at the Engineers' Club on Jan. 29, 1917. Interesting papers on the Arizona Mining Districts and on the Creighton Mine at Sudbury were read.

Several members of the freshman class who have changed from other branches of engineering to mining have been taken into the Club, so that we are now booming. We are also making an attempt to have more of our members apply for junior membership in the Institute, and hope to report favorably along these lines in the near future.

C. A. RICHARDSON, *Secretary*.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS

AMERICAN SOCIETY OF MECHANICAL ENGINEERS

AMERICAN INSTITUTE OF MINING ENGINEERS

UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy and Chemistry

- CALCIUM CARBIDE AND ACETYLENE. Ed. 3. By G. G. Pond. State College, Pa., 1917. (Gift of Henry B. Koons.)
- ELEMENTARY FIRST AID FOR THE MINER. Miners' Circular 23, U. S. Bureau of Mines. Washington, 1916.
- GOLD MINING IN KOREA. By Edwin W. Mills. Royal Asiatic Society Trans. Vol. VII, pt. 1, v, p. 1916. (Gift of Edwin W. Mills.)
- MANUFACTURE OF CARBIDE OF CALCIUM. By Charles Bingham. London, 1916.
- PETROLEUM WITHDRAWALS AND RESTORATIONS AFFECTING THE PUBLIC DOMAIN ISSUED BETWEEN JAN. 16 AND SEPT. 30, 1916. Bull. 623—Appendix A, U. S. Geological Survey. Washington, 1916.
- PRINCIPLES AND PRACTICE OF SAMPLING METALLIC METALLURGICAL MATERIALS. Bull. 122, U. S. Bureau of Mines. Washington, 1916.
- SURFACE SUBSIDENCE IN ILLINOIS RESULTING FROM COAL MINING. Bull. 17, Illinois State Geological Survey. Urbana, 1916.
- UNDERGROUND LATRINES FOR MINES. Technical paper 132. U. S. Bureau of Mines. Washington, 1916.
- VAPOR PRESSURES OF VARIOUS COMPOUNDS AT LOW TEMPERATURES. Technical paper 142. U. S. Bureau of Mines. Washington, 1916.

Geology and Mineral Production

- ALBANY MOLDING SAND. By David H. Newland. Bull. 187, from N. Y. State Museum. Albany, 1916. (Gift of University of the State of New York.)
- BIBLIOGRAPHY OF AUSTRALIAN MINERALOGY. Mineral Resources No. 22. New South Wales, Department of Mines. Sydney, 1916.
- CALIFORNIA MINERAL PRODUCTION FOR 1915. Bull. No. 71, California State Mining Bureau. San Francisco, 1916.
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- GEOLOGICAL MAP OF MYSORE. 1915.
- MINERAL RESOURCES OF ALASKA, REPORT ON PROGRESS OF INVESTIGATIONS IN 1915. Bull. No. 642, U. S. Geological Survey. Washington, 1916.
- NOTES ON SOME MINING DISTRICTS IN EASTERN NEVADA. Bull. No. 648, U. S. Geological Survey. Washington, 1916.
- REPORT OF THE GEOLOGICAL RECONNAISSANCE OF THE STATE OF VIRGINIA. -By William B. Rogers. Philadelphia, 1836.
- REPORTS OF THE PROGRESS OF THE GEOLOGICAL SURVEY OF THE STATE OF VIRGINIA. 1st-2d. By William B. Rogers. Philadelphia, 1838.
- REVIEW OF THE GEOLOGY OF TEXAS. Bull. No. 44, Texas University. Austin, 1916.
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- STATE OF NEW YORK AT THE PANAMA-PACIFIC INTERNATIONAL EXPOSITION, SAN FRANCISCO, CAL. Feb. 20-Dec. 4, 1915. Albany, J. B. Lyon Co., 1916.
- TERTIARY FORMATIONS OF WESTERN WASHINGTON. Bull. No. 13, Washington Geological Survey. Olympia, 1916.
- YUKON-KOYUKUK REGION, ALASKA. Bull. No. 631, U. S. Geological Survey. Washington, 1916.

Gift of Albert F. Ganz

- EFFECT OF ELECTROLYSIS ON ENGINEERING STRUCTURES. (Paper read at meeting International Engineering Congress, 1915.)
- ELECTROLYTIC CORROSION OF IRON BY DIRECT CURRENT IN STREET SOIL. (Paper presented at 29th Annual Convention, Amer. Inst. Electrical Engr., June 25, 1912.)
- ELECTROLYSIS FROM STRAY ELECTRIC CURRENTS. (Before Amer. Water Works Association, June 6, 1912 and New England Association of Gas Engineers, Feb. 19, 1913.)
- REPORT. Investigation for stray electric currents in the City of Winnipeg and in adjoining municipalities to H. A. Robson. 1915.
- REPORT ON ELECTROLYSIS CONDITIONS IN SPRINGFIELD, OHIO. 1914.
- REPORT ON INSULATED RETURN FEEDER SYSTEM FOR MITIGATING ELECTROLYSIS INSTALLED BY THE UNITED RAILWAYS COMPANY OF ST. LOUIS, MO., IN THE ANN AVE. SUBSTATION DISTRICT. May, 1914.

General

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Catalog 4-B. Westinghouse Feeder Voltage Regulators and Transformer Apparatus. Nov., 1916.

Catalog 7-A. Westinghouse Arc Lamps and Lighting Systems. Dec., 1916.

Catalog 7-B. Westinghouse Cooper Hewitt Mercury Rectifier Outfits. Dec., 1916.

Catalog 8-C. Westinghouse Electric Ware. April, 1916.

Catalog 8-D. Westinghouse Electric Ranges. April, 1916.

Leaflet 1660. Automatic starters for squirrel-cage induction motors.

Leaflet 1661. Automatic starters for wound rotor induction motors.

Leaflet 3668-B. Electric Arc Welding Equipments.

Leaflet 3814. Types CO and CR A.C. Relays.

Leaflet 3843. Auto starters for large squirrel-cage induction motors.

Leaflet 3903. Alternating-current grinder motors—Type CS.

Leaflet 3927. Feeder Voltage Regulators.

Leaflet 3949. Types EW and EI D.C. automobile motors.

Leaflet 3950. Type PW portable meters.

D. L. 3555-A. Westinghouse self-contained vertical shaft A. C. generators.

Special Publication No. 1574. Maintenance of Equipment Harvard Shops. Cleveland Railway Co., Oct., 1916.

WHITE FUEL OIL ENGINEERING CORPORATION, New York, N. Y. Low pressure mechanical oil burning system. 1916.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Jan. 1, 1917 to Feb. 10, 1917.

- ALLEN, ROBERT SEXTON, Chief Chemist at Concentrator,
Inspiration Cons. Copper Co., Globe, Ariz.
- ALTIO, KARL FREDRIK, Min. Engr., The Sissert Mining District Co., Ltd.,
Polefskoy Zavod, Perm Government, Russia.
- ARGALL, PHILIP HENRY, Mgr., Ozark Smelt. & Min. Co., Magdalena, New Mexico.
- BAILEY, MARC.....Supt., Wolverine & Arizona Min. Co., Bisbee, Ariz.
- BAKER, DAVID FRANKLIN, Asst. Supt. of Blast Furnaces,
Alan Wood, Iron and Steel Co., Swedeland, Pa.
- BATES, BENNETT R, Min. Engr.....1025 Peoples Gas Bldg., Chicago, Ill.
- BECK, HOWARD A.....Shift Foreman, Aurora Cons. Mines Co., Aurora, Nev.
- BELL, LEO RICHARD.....Casilla 270, La Paz, Bolivia, South America.
- BENJAMIN, FLOYD SHAW, District Mgr., Aetna Explosives Co., Inc.,
800 Torrey Bldg., Duluth, Minn.
- BENTLEY, CHARLES, Analytical and Experimental Work, S. D. Mine Experiment
Station, S. D. State School of Mines, Rapid City, S. D.
- BENSON, PAUL T.....Assayer, Tacoma Smelting Co., Tacoma, Wash.
- BIRCH, G. HOWARD, Vice-Pres. and Genl. Mgr., Dan Creek Min. Co.,
45 Broadway, New York, N. Y.
- BOOTH, LIONEL E., Met., Concentration Dept., Anaconda Copper Mining Co.,
600 Main St., Anaconda, Mont.
- BOTSFORD, MILTON PERRY, Technical Demonstrator, Aetna Explosives Co., Inc.,
800 Torrey Bldg., Duluth, Minn.
- BRADLEY, JOSEPH MARKLEY, Supt., The Dives and Shenandoah Mines,
Silverton, Colo.
- BURGREN, ARTHUR W., Min. Engr., Amer. Smelt. & Ref. Co., 1112 Mills Bldg.,
El Paso, Tex.
- CADY, GILBERT HAVEN.....Geol., Illinois State Geological Survey, Urbana, Ill.
- CAMPBELL, E. E., Supt., Hidden Creek Mines, Granby Cons. Min. Smelt. & Power Co.,
Anyox, B. C., Canada.
- CARTER, RALPH R., Division Engr., Lehigh and Wilkes-Barre Coal Co.,
Audenried, Carbon Co., Pa.
- CHEDA, GILBERT ERNEST, Min. Engr.....San Luis Obispo, Cal.
- COLLBRAN, HERBERT EDWARD, Mine Operator, 1277 Williams St., Denver, Colo.
- COOPER, WILLIAM H., Concentrator Supt., Minas Tecolotes y Anexas,
1112 Mills Bldg., El Paso, Tex.
- CROSSLEY, GEORGE C., Owner and Operator, The Crossley Mines,
Crossley Station, Toms River, N. J.
- DENNIS, GRAHAM B., Pres., Northwest Mining Assn., The Knickerbocker,
Spokane, Wash.
- DETWEILER, ALFRED NICKS.....Supt., National Zinc Co., Springfield, Ill.
- DONOVAN, PERCY WILLIAMS, Min. Engr., Supt. of Contract Drilling,
E. J. Longyear Co., 710 Security Bldg., Minneapolis, Minn.
- DUDLEY, BOYD, JR., Asst. Prof. of Metallurgy, Pennsylvania State College,
State College, Pa.
- DUGGLEBY, ALFRED L., Mine Supt., Juragua Iron Co., Box 383, Santiago de Cuba,
Cuba.
- DYMOCK, JOHN SOMERVILLE, Min. Engr., Calumet and Hecla Min. Co., Box 973,
Warren, Ariz.
- EDDY, HAROLD C., Engr., Petroleum Rectifying Co. of Cal., 350 California St.,
San Francisco, Cal.
- ELLIS, ARTHUR G.....Genl. Mill Foreman, Ray Cons. Copper Co., Hayden, Ariz.
- ERIC, AXEL, Designing Engr., Utah Copper Co., McCormick Bldg.,
Salt Lake City, Utah.
- FARRAND, IRA WILLIAM, Foreman, Dutton Mine, East Butte Copper Min. Co.,
842 W. Galena, Butte, Mont.
- FIDLAR, JOHN BRAINARD.....Geol., Gypsy Oil Co., 410 Clinton Bldg., Tulsa, Okla.
- FRENCH, C. L.....Engr., Research Corporation, 63 Wall St., New York, N. Y.
- FRENCH, STUART W.....Genl. Mgr., Phelps, Dodge & Co., Douglas, Ariz.

- GARNER, AUBREY HAMILTON, Geol. and Field Supervisor,
The Caribbean Petroleum Co., Maracaibo, Venezuela, South America.
- GARNER, CARROLL A., Div. Engr., Lehigh Valley Coal Co., 343 W. Broad St.,
Hazleton, Pa.
- GARNETT, THOMAS HERNDON, Supt., San Xavier Mine, The Empire Zinc Co.,
Bin Q, Tucson, Ariz.
- GENTRY, MARTIN BUTLER, Min. Engr., Andes Exploration Co., Casilla 85-D,
Teatinos 351, Santiago, Chile, South America.
- GLEASON, PHILIP RAYMOND, Met. and Asst. Smelter Supt., Central Chile Copper Co.,
Ltd., Panulcillo, Chile, South America, via Coquimbo.
- GODDARD, JOHN NEWTON, Met., St. Joseph Lead Co., Herculanum, Mo.
- GOODWIN, L. HALL, Min. Engr., 3 Barstow St., Allston Station, Boston, Mass.
- GRENINGER, IRA L., Min. Engr., P. O. Box 841, Miami, Ariz.
- GRONNINGSATER, ANTON M., Genl. Mgr., Kristiansands Nikkelfraffineringsverk,
Kristiansand S, Norway.
- HAIGH, J. GLENN, Mill Supt., Empire Zinc Co., Silver City, New Mexico.
- HAIN, GEORGE W., District Supt., Lehigh Coal and Navigation Co., Lansford, Pa.
- HALL, WILLIAM S., Min. Engr., Inspiration Cons. Copper Co.,
P. O. Box 696, Miami, Ariz.
- HART, RAY WESTON, Geol., Cananea Cons. Copper Co., Cananea, Son., Mexico.
- HAWKINS, T. G., JR., Asst. Mgr., Alvarado Min. and Mill Co., 706 Mills Bldg.,
El Paso, Tex.
- HOULE, ALBERT JOSEPH, Prof. of Metallurgy, Michigan College of Mines,
117 Hubbel Ave., Houghton, Mich.
- HOWRY, HAMILTON H., Research Work, Met., Chile Exploration Co.,
120 Broadway, New York, N. Y.
- HUNT, HENRY DUFF, Research Engr., Miami Copper Co., Miami, Ariz.
- HUNTER, HARRY G., Mgr., Pike Hill Mines, Corinth, Vt.
- IRWIN, JOSEPH STEWART, Geol., Carter Oil Co., 327 Central National Bank Bldg.,
Tulsa, Okla.
- JOHNSON, A. D., JR., Anaconda Copper Min. Co., 406 E. Park St., Anaconda, Mont.
- KNOWLTON, ARTHUR D., Mining and Hydraulic Engr., 912 Boston Bldg.,
Salt Lake City, Utah.
- KRAFT, PHILIP, JR., Geol., The Dome Mines Co., Ltd., South Porcupine,
Ont., Canada.
- LEE, CHARLES SHEPARD, Met. Engr., Asst. to Testing Engr.,
Copper Queen Cons. Min. Co., Douglas, Ariz.
- LINDLEY, J. GARY, Met. Accountant, Calumet and Arizona Min. Co.,
Box 333, Douglas, Ariz.
- LINTHICUM, J. FRANCIS, Chief Chemist and Mine Engr., American Bauxite Co.,
Bauxite, Ark.
- LOGAN, WILLIAM WELLS, Min. Engr., 3532 Telegraph Ave., Oakland, Cal.
- LONGMORE, E. L., Assayer and Chemist, Hollinger Cons. Gold Mines, Ltd.,
Timmins, Ont., Canada.
- LONGYEAR, CLYDE STANLEY, Exploring Engr., E. J. Longyear Co.,
710 Security Bldg., Minneapolis, Minn.
- LORD, JAMES OSBORN, Illinois Steel Co., Gary, Ind.
- LUMAGHI, OCTAVIUS LOUIS, Engr., St. Louis Smelt. & Ref. Co., St. Francois, Mo.
- LUNDQUIST, EMIL H., Supt., Magma Copper Co., Superior, Ariz.
- MCATEE, BERNARD M., Min. Engr. in charge of flotation, Inspiration Concentrator,
Inspiration Cons. Copper Co., P. O. Box 904, Miami, Ariz.
- MCCORMICK, JAMES HOMER, Mining, Supt., Barnes King Development Co.,
(Piegan Gloster Mine), Marysville, Lewis and Clarke Co., Mont.
- MCDONALD, FELIX, Mine Supt., Inspiration Cons. Copper Co., Miami, Ariz.
- MCINTOSH, ROBERT, Asst. Supt., Stamp Mills, Calumet and Hecla Min. Co.,
Lake Linden, Mich.
- MCINTYRE, RICHARD S., Asst. Min. Engr., Burma Mines, Ltd., Namtu,
Northern Shan States, Burma, India.
- McKEE, WALTER S., Vice-Pres., American Manganese Steel Co.,
332 Michigan Ave., Chicago, Ill.
- MANWARING, EDGAR G. R., Min. Engr., Barnes-King Development Co.,
Kendall, Mont.
- MIEYR, GEORGE, Master Mechanic, Copper Queen Cons. Min. Co., Bisbee, Ariz.
- MONTAGUE, CHARLES L., Mgr., The Cusi Min. Co., 711 Mills Bldg., El Paso, Tex.
- MOORE, STANLEY B., Supt., Bacchus Works, Hercules Powder Co., Bacchus, Utah.

- MORE, JOHN, Genl. Mgr., The International Nickel Co. of Canada, Ltd.,
Port Colborne, Ont., Canada.
- MORRIS, GEORGE, Supt., Pittsburg and Erie Coal Co., So. Burgettstown,
Washington Co., Pa.
- MOTHERWELL, HARRY AULD BELL, Chemist, Arizona Copper Co., Clifton, Ariz.
- MOULTON, HERBERT G., Cons. Min. Engr., Eugene Meyer, Jr. & Co., 14 Wall St.,
New York, N. Y.
- MURPHY, JOHN FRANCIS, Associate Prof. of Mining, Univ. of Minnesota,
School of Mines, Minneapolis, Minn.
- NELSON, HENRY CLINTON, Foreman, King Mine, Arizona Copper Co., Metcalf, Ariz.
- NOGUÉS, DOMINGO, Engr. for Argentine Government,
Real Estate Trust Bldg., Washington, D. C.
- OLSON, KARL ELOFF.....Chemist, Goldschmidt Detinning Co., East Chicago, Ind.
- PALMER, WALTER S.....Prof. of Metallurgy, University of Nevada, Reno, Nev.
- PARK, MOREY A., Chemist and Metallographist, Parkesburg Iron Co.,
Parkesburg, Chester Co., Pa.
- PAUL, H. W.....Min. and Met. Engr., Yamashita-cho No. 175, Yokohama, Japan.
- PENHALE, HEDLEY J. D., Min. Engr., Mine Mgr.,
Sociedad Estanifera Totoral Consolidada, Mina La Riena, Oruro, Bolivia, S. Amer.
- PERKINS, FRED H. Genl. Mgr. Shamrock Cons. Min. Co., R. R. No. 1, Glendale, Ariz.
- PICKETT, C. E.....Min. Engr., Sampler, Nevada Cons. Copper Co., McGill, Nev.
- POMMERANTZ, KIVA, Cons. Min. Engr., Ore Trading Co., Ltd.,
Casilla 226, Coquimbo, Chile, So. Amer.
- PRICE, WILLIAM R., Supt., Shannon Mine, Barnes King Development Co.,
Marysville, Mont.
- RAMSEY, ERNEST COPE, Engr. in charge of field pressure regulation,
The Ohio Fuel Co., 52 W. Gay Street, Columbus, O.
- REICHEL, DAN H.....Asst. Chemist, Miami Copper Co., Box 100, Miami, Ariz.
- ROBSON, HAROLD CARMICHAEL, Asst. Engr. in Smelter, Spassky Copper Mine, Ltd.,
Spassky Zavod, Akmolinsk, Siberia.
- ROGERS, GEORGE CLARK, Min. Engr., American Zinc Co. of Tenn., Mascot, Tenn.
- RUHL, OTTO.....Consulting Min. Engr., 609 Keystone Bldg., Joplin, Mo.
- SAPPER, C. FLOYD.....Engineering Staff, St. Joseph Lead Co., Herculaneum, Mo.
- SELDEN, WILLIAM H. JR.....Iron River, Mich.
- SILLARS, DANIEL, Met. Chemist, 18 Hilda Place, Saltburn-by-the-Sea,
Yorkshire, England.
- SINCLAIR, J. FRANKLIN, Underground Supt., Copper Queen Cons. Min. Co.,
Box 126, Bisbee, Ariz.
- SLATER, HENRY BYRON.....P. O. Box 515, Riverside, Cal.
- SMYTH, RAYMOND WEIR, Met. Dept., Youngstown Sheet & Tube Co., Youngstown, O.
- SNOW, FREDERICK W., Min. Engr., Mine Foreman, Magma Copper Co.,
Superior, Ariz.
- SOMMERCAMP, HENRY JOHN.....Weiser, Idaho.
- SPEED, FLETCHER B. JR., Min. Engr., Managing Engr., Virginia Barytes Co.,
39 National Bank Bldg., Charlottesville, Va.
- STANLEY, FRANK ARTHUR, Editorial Representative, Hill Publishing Co.,
10th Ave. and 36th St., New York, N. Y.
- STEVENS, GEORGE R., Supt.....Olympic Mines Co., Mina, Nev.
- STODDARD, A. C., Min. Engr.....Inspiration Cons. Copper Co., Miami, Ariz.
- STOUT, WILBER, Asst. Geol., Geological Survey of Ohio,
Orton Hall, Ohio State Univ., Columbus, O.
- STROMBERG, OSCAR, Cons. Engr.....Tribune Bldg., New York, N. Y.
- STUART, CHARLES E., Supt., Bonanza Unit, American Smelt. & Ref. Co.,
1112 Mills Bldg., El Paso, Tex.
- SUNDT, F. ALFREDO, Min. Engr., Genl. Mgr., Compania Corocoro de Bolivia,
Care C. G. Barahona, Arica, Chile, S. Amer.
- SUTOR, DONALD McDONALD, Southwestern Sales Mgr.,
Sullivan Machinery Co., 511 Mills Bldg., El Paso, Tex.
- WALTON, BRINSLEY M., Min. Engr.; Mgr., Porcupine Premier Min. Co.,
Ltd., P. O. Box 330, So. Porcupine, Ont., Canada.
- WALTER, HERBERT WINFRED.....International Nickel Co., Bayonne, N. J.
- WHITING, M. A., Electrical Engr., Power & Mining Engrg. Dept.,
General Electric Co., Schenectady, N. Y.
- WILKINSON, JOSEPH R., Testing Dept., Anaconda Copper Min. Co., Anaconda, Mont.
- WILLIAMS, GEORGE L.....Mech. Engr., Empire Zinc Co., Hanover, New Mex.

WILLIAMS, HOWARD EDWARD, Chief Draftsman, Calumet & Hecla Min. Co.,
1144 Calumet Ave., Calumet, Mich.
WILLIAMS, MAURICE E., Asst. Mill Supt., Hollinger Cons. Gold Mines,
Ltd., Box 121, Timmins, Ont., Canada.
WILSON, RIDGEWAY R., Min. Engr., Crows Nest Pass Coal Co., Ltd.,
Ferne, B. C., Canada.
WOOD, JAY PENDLETON, Min. Engr., Supt., The Mountain Top Min. Co., Ouray, Colo.
YEN, YEH-TZE, Engr., Blast Furnace Dept., Hanyang Iron and Steel Works,
Hanyang, China.

Associate Members

MILLER, W. W. Sales Engr, General Electric Co., Schenectady, N. Y.
PAYNE, J. HOWARD, Mining Representative, Westinghouse Electric & Mfg., Co.,
165 Broadway, New York, N. Y.
STANSEL, N. R., Local Mgr. Southwest General Electric Co., El Paso, Tex.
ZANG, ADOLPH F., Vice-Pres. and Treas., The Vindicator Cons. Gold Min. Co.,
603 Symes Bldg., Denver, Colo.

Junior Members

ABOUCAR, SYLVAIN SLEYMAN, Senior Student, School of Mines,
Columbia Univ., New York, N. Y.
BARTON, JOE C., Student. Missouri School of Mines, Rolla, Mo.
BOWLES, MARTIN F., Student. Missouri School of Mines, Rolla, Mo.
DALE, RALPH, Student. Missouri School of Mines, Rolla, Mo.
HARDINGE, HARLOWE, Mech. Engr., Hardinge Conical Mill Co.,
120 Broadway, New York, N. Y.
HERMSDORF, RICHARD PAUL ERNST, Student, Columbia University,
School of Mines, New York, N. Y.
HIPPARD, CLEMENCE W., Student. Missouri School of Mines, Rolla, Mo.
KRAUS, EDGAR, Student. Columbia School of Mines, New York, N. Y.
LINN, THEODORE I, Student, Lehigh Univ., 209 Warren Square, So. Bethlehem, Pa.
LYNCH, WILLIAM W., Student. Hammond Laboratory, New Haven, Conn.
NG, SYDNEY S. H., Mining Student, Fumald Hall, Columbia Univ., New York, N. Y.
PELTON, EDWARD FRANKLIN, Mine Supt., Burro Mountain Copper Co.,
Tyrone, New Mex.
RIDDLE, DONALD D., Junior Student, Colorado School of Mines, Golden, Colo.
SCHLOSS, JOHN MALCOLM, Senior Student, School of Mines, Columbia Univ.,
New York, N. Y.
WILLIAMS, CHARLES FRANCIS, Min. Engr., Cananea Cons. Copper Co.,
Cananea, Son., Mex.

Total Membership, Jan. 31, 1917. 5,949

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

Applications Lacking Endorsement

Applications for membership have been received from Mr. Koch, Mr. Lehman and Mr. Trench, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

Otto Koch, Chuquicamata, Chile, South America.

Proposed by Henry Hay, Huntington Adams.

Born 1884, Bodo, Norway. 1890-1902, High School, Bodo, Norway. 1902-03, Military Academy, Kristiania, Norway. 1903-06, Chemical-metallurgical courses, Technical College, Trondhjem, Norway. 1906-07, Government Experimental Laboratory, Bergen, Norway. 1907-08, Chief Chemist, Min. and Smelt. Co., Norway. 1909-14, Chief Chemist and Smelter Supt., Kristiansands Nikkelraffineringsverk, Norway.

Present position—1914 to date: Supt., Smelt. & Mill. Dept., Chile Exploration Co.

Andrew William Lehman, El Nilhne, Chagres, Chile.

Proposed by Mark R. Lamb.

Born 1878, Paris, France. 1890-95, Dulwich College, London. 1896-1900, Royal School of Mines, London, where I obtained a Royal Exhibition and later became an Associate of the Royal School of Mines (A. R. S. M.). Also a member of the Inst. of Min. and Met., London. 1900, Entered the firm of Bewick, Moreing & Co., and went out in their employ to Western Australia, East Africa, Sinai Peninsula and N. Brazil. 1904, Went to W. Africa as Surveyor and Assayer for the Himan Concessions. 1905, Went to Peru for the Huinac Copper Mines. 1906, Came to Chile as Min. Engr., Société des Mines des Cuivre de Catemon.

Present position: Genl. Mgr., Société des Mines de Cuivre de Catemon.

Robert Hamilton Trench, Rangoon, Burma, India.

Proposed by A. W. G. Bleek, John Cadman.

Born 1880, Liverpool. 1902, Univ. of Cambridge, England, 2d class Mechanical Science. 1902-04, British Westinghouse Electric & Mfg. Co., Trafford Park, Manchester. 1904-08, Asst. Mgr. and Mgr., London & Pacific Petroleum Co. 1908-12, general reporting and executive work on properties in Russia, West Indies, East Indies, Salina, United States, Messrs. Thompson and Hunter, London.

Present position—1912 to date: Genl. Mgr., British Burmah Petroleum Co.

The following persons have been proposed during the period Jan. 1, 1917, to Feb. 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

George K. Anderson, Jr., Longdale Mines, Va.

Proposed by R. A. Lipscomb, F. U. Humbert, R. B. Ladoo.

Born 1892, Clifton Forge, Va. 1907-10, Randolph-Mason Academy. 1912-13, Virginia Military Institute. 1910-14, Foreman, Asst. Supt., R. R. Construction, W. W. Baxley & Co. 1914, Asst. Chemist, Low Moor Iron Co., Virginia. 1914, Supt., Coke ovens, Kay Moor, W. Va. Three months, Supt. construction, Supt. of dismantling operation, Low Moor Iron Co. of Va.

Present position: Supt. of Longdale Mines.

Thomas Childrey Baker, Pachuca, Hidalgo, Mexico.

Proposed by C. A. Lantz, T. B. Stearns, William J. Cox.

Born 1886, Stribling, Tenn. 1905-09, Leland Stanford Jr. Univ., A. B. 1909-10, Mine Surveyor, Miami Copper Co., Ariz. 1910-11, Mine Surveyor, Cia. Minera de Penoles, Ojuela, Dgo., Mex. 1911-16, Chief Surveyor, Shift Boss, Mine Foreman and Mine Supt., Cia. Santa Gertrudis, S. A., Pachuca, Hidalgo, Mex.

Present position: Mine Supt., Cia. Santa Gertrudis, S. A.

Harry Lowell Barr, Terry, So. Dak.

Proposed by Ernest R. Graham, Charles A. Chase, A. E. Sargent.

Born 1891, Moberly, Mo. Prior to 1904, F. P. Blair School, St. Louis, Mo. 1904-08, Yeatman High School, St. Louis, Mo. 1914-15, Univ. of Missouri. 1911-14, Four months as miner, four months as solution man in cyanide plant, Trojan Min. Co. and remaining time as an assayer.

Present position—1915 to date: Assayer in charge, Mogul Min. Co.

Joseph Ramsey Blackburn, Austen, W. Va.

Proposed by E. V. d'Invilleers, J. B. Dilworth, Walter Gilman.

Born 1888, Staunton. 1910, Washington and Lee Univ., B. S. in Chem. Engrg. 1911, Washington and Lee Univ., Lexington, Va., B. S., in Min. Engrg. Prior to 1911, during vacation, worked on Engrg. Corps in coal fields and iron fields, Tennessee, Virginia and West Virginia. 1911, with Mr. E. V. d'Invilleers, Geol. and Min. Engrs., Philadelphia, Pa. 1912-16, Engr., Consolidation Coal Co., Somerset, Pa. 1916, Chief Engr., Quemashoning Creek Coal Co., Somerset, Pa.

Present position—1916 to date: Vice-Pres. and Genl. Mgr., Austen Coal and Coke Co.

Frank P. Botsford, Gilbert, St. Louis Co., Minn.

Proposed by Stacy H. Hill, Edwin J. Collins, Dwight E. Woodbridge.

Born 1878, L'Anse, Mich. 1896, Grad., L'Anse High School. 1897, Grad., Hancock High School. 1900, Michigan College of Mines, E. M. 1900-01, Miner, Arcadian Min. Co., Houghton Co., Mich. 1901-07, Chief Engr., Pickands, Mather & Co. on the Mesaba Range. 1907-13, Supt. of Bangor, Syracuse and Mohawk Min. Companies—Pickands, Mather & Co.

Present position—1913 to date: Asst. Genl. Supt., Pickands, Mather & Co. on the Mesaba Range.

Spruille Braden, Santiago, Chile, South America.

Proposed by B. B. Thayer, William Braden, W. D. Thornton.

Born 1894, Elk Horn, Mont. 1904-09, Various preparatory schools. 1910, Montclair Academy. 1914, Sheffield Scientific School, Yale Univ., Ph. B. in mining. 1911-12, three months in Steptoe Concentrator, Nevada Cons. Copper Co. Various positions remainder of year as mucker, miner and timberman in Cumberland Ely Mine. 1913, summer, Experimental Plant, Chile Exploration Co., Perth Amboy, N. J.

Present position—1914 to date: Engr., Andes Exploration Co. of Maine.

Clyde L. Brown, Goldfield, Nev.

Proposed by J. G. Scragham, Henry W. Rives, John K. Turner.

Born 1882, Cleveland, O. 1899, Chemistry and physics, Butte High School. 1900-05, Practical experience in mining and electricity. 1905-16, a course in the Scranton School, in mechanical and electrical engineering. Also studies through the various current publications in assaying, cyaniding and mineralogy. 1900, Mining, Anaconda Copper Co. 1901-05, Smelting and electricity, Colorado Smelt. Co., also Pillsburg and Montana Min. and Smelt. Co., Butte, Mont. 1905-06, Humboldt Smelt. Co., Humboldt, Ariz.; Minas Penoles Co., Durango, Mex.; Esperanza Min. Co., El Oro, Mex.; El Oro Min. Co., Ltd., El Oro, Mex. 1907-16, Goldfield and vicinity.

Present position: Goldfield Merger Mines Co.

Stanley Brown, Chuquicamata, Chile, South America.

Proposed by James S. Wroth, James E. Bell, Fred MacCoy.

Born 1887, Swansea, So. Wales. 1900, Swansea Grammar School. 1901-05, Apprentice and Chief Asst. to J. E. Penrose, Public Assayer, Swansea, So. Wales. With Technical training at Swansea Metallurgical College. 1906-09, Chief Chemist, Huanchaca Min. Co., Pulasayo, Bolivia, So. Amer. 1910, Topographer on Survey, Bolivia Railway Co., Bolivia. 1910-11, Asst. Engr., Mulville Bros., Buenos Aires. 1912-13, Supt., Aranayo Francke Mines, Ltd., Chocaya, Bolivia. 1913-14, Supt., Oploca Min. Co., Chocaya, Bolivia.

Present position—1914 to date: Mine Chemist and Charge of Vein Mining, Chile Exploration Co.

David Gayley Browne, New York, N. Y.

Proposed by Lewis G. Rowand, George C. Stone, A. D. Beers.

Born 1892, Copper Cliff, Ont., Canada. Elementary education in Canada and United States. 1915, Queen's Univ., Kingston, Ont., Canada, B. S. 1909-15, summers, general work in construction of buildings. 1915, on construction of mills, New Jersey Zinc Co., Franklin Furnace, N. J. 1916, Construction of oil shale refining plant, for R. M. Catlin, Elko, Nev.

Present position: Engrg. Dept., New Jersey Zinc Co.

Daniel Worth Butner, Santiago, Cuba.

Proposed by Max Roesler, A. F. Duggleby, Paul S. King.

Born 1890, Winston-Salem, N. C. 1907-11, Colorado Springs High School, Colo. 1911-15, Colorado School of Mines, E. M. 1915-16, Asst. Engr., Utah Metal and Tunnel Co., Bingham Canyon, Utah. 1916, Asst. Engr., Cinco Minas Co., Jalisco, Mex. 1916, Chemist, Molybdenum Products Co., Tucson, Ariz.

Present position: Chemist, Juragua Iron Co.

Frederick E. Calkins, Miami, Ariz.

Proposed by Walter Harris, W. B. Cramer, W. S. Sultan.

Born 1883, Petoskey, Mich. 1900-01, Univ. of Michigan. 1904-08, Michigan College of Mines. 1909, Engr., Boston-Miami Copper Co. 1910, Field Engr., Venture Exploration Co. 1911, Min. Engr., Summit Copper Co. 1912-13, Asst. Geol. to George H. Garrey. 1914-15, Mine Inspector, Detroit Copper Co. 1915-16, independent mine examination work.

Present position: Min. Geol. and Engr., Independent work.

Henry L. Christensen, Juneau, Alaska.

Proposed by B. Britton Gottsberger, Frederick W. Solomon, H. P. Bowen.

Born 1889, Manti City, Utah. Public School education.

Present position—1916 to date: Mill Supt., Alaska Juneau Gold Min. Co.

Willis B. Clemmitt, Baltimore, Md.

Proposed by Joseph W. Richards, Benjamin L. Miller, H. Eckfeldt.

Born 1887, Richmond, Va. 1906, Grad., Baltimore Polytechnic Institute. 1910-13, Grad., Lehigh Univ., E. M. 1906-07, Laboratory Asst., Baltimore Polytechnic Institute. 1907-10, Surveying, drafting, and Asst. Engr., under O. F. Tackley, Harbor Engr. 1913-14, Blast furnace dept., furnace operation and metallurgical work, Republic Iron and Steel Co. 1914, first helper, Open Hearth, Inland Steel Co. 1914-15, Bridge and Highway Construction, Maryland State.

Present position—1915 to date: Met. Engr. in charge, heat treatment, chemical work and metallography, Cartridge Coal Dept., Baltlett Hayward Co.

John Coolbaugh, Allentown, Pa.

Proposed by Irving A. Stearns, R. V. Norris, Morris Williams.

Born 1871, Middletown, Pa. 1888, High School, Middletown, Pa. 1900, International Correspondence School, Scranton, Pa. 1889-92, Carpenter trade with John Hull, Wilkes-Barre, Pa. 1892, with H. S. Reets, Min. Engr. 1902, Transitman in charge of work, Wilkes-Barre, Pa. 1902, Chief Coal Inspector. 1906, Susquehanna Coal Co., Wilkes-Barre, Pa. 1907-16, Contracting and Engineering for myself.

Present position: Contracting.

Lloyd D. Cooper, Grass Lake, Mich.

Proposed by C. K. Leith, W. J. Mead, John E. Hodge.

Born 1878, Grass Lake, Mich. 1905, Michigan College of Mines, E. M. 1905-08, Engr., E. J. Longyear. 1908-15, Chief Engr., E. J. Longyear Co. 1915-16, Mgr., in shaft sinking work, E. J. Longyear Co., Loppens Verk, Norway.

Present position: Chief Engr., E. J. Longyear Co.

Clifton Sherwin Corbett, Edwardsville, Ill.

Proposed by H. M. Roberts, W. J. Mead, C. K. Leith.

Born 1891, Edwardsville, Ill. 1908-10, Classical course, Ohio Wesleyan Univ. 1910-12, Major in Geology, Northwestern Univ., A. B. 1912-13, Post-graduate work in Geology, Northwestern Univ. 1913-15, Post-graduate work in Geology, Univ. of Wisconsin, A. M. 1911, summer, Illinois Geological Survey. 1912-14, summers, Geologic Aid, United States Geological Survey. 1915, summer, Wisconsin Geological Survey. 1915, autumn, Geol., Producers Oil Co.

Present position—1916 to date: Geol., E. J. Longyear Co.

Ernest R. Crutcher, Great Falls, Mont.

Proposed by W. N. Tanner, R. B. Caples, J. O. Elton.

Born 1891, Owensboro, Ky. 1908, Grad., High School, Salida, Colo. 1914, Grad., Colorado School of Mines, E. M. 1909, underground, Taylor Mountain Min. Co. 1912-13, Experiment work, chiefly on Cottrell treater, Ohio and Colorado Smelt. & Ref. Co. 1914, Dry chlorination, Northwestern Metals Co., Helena, Mont. 1915-16, Anaconda Copper Min. Co.

Present position: Zinc Plant, Anaconda Copper Min. Co.

Warren W. Currens, Ray, Ariz.

Proposed by L. S. Cates, W. S. Boyd, L. A. Blackner.

Born 1888, MaHoon, Ill. 1906, Grad., St. John's Military School, Salina, Kans. 1912, Grad., Colorado School of Mines.

Present position—1912 to date: Chemist, Ray Cons. Copper Co.

Edward W. Davis, Duluth, Minn.

Proposed by W. G. Swart, Fred A. Jordan, Theodore B. Counselman.

Born 1888, Cambridge, Ind. 1894–1907, Public Schools, Elkhart, Ind. 1907–11, Purdue Univ., B. S. 1911–12, Westinghouse Electric and Mfg. Co. 1912–13, Fort Wayne Electric Works, General Electric Co. 1913–16, Instructor in Department of Mine Plant and Mechanics, School of Mines, Univ. of Minn.

Present position—1916 to date: Experimental mining work with W. G. Swart.

Joe De Arozena, Suijotoa, Ariz.

Proposed by J. A. Ede, Thomas T. Read, William H. Shearman.

Born 1890, Tex. 1908, Grad., San Angelo High School, San Angelo, Tex. 1911, Engrg. Student, Texas A. & M. College, College Station, Tex. 1912–16, Univ. of Arizona, Tucson, Ariz., B. Sc. 1908–09, Tool dresser, Epes & McAlvain, Toyah, Tex. 1914–15, Mill testing, Arizona State Bureau of Mines, Tucson, Ariz. 1915, Old Yuma Min. Co., Tucson, Ariz. 1915–16, Assayer, Arizona State Bureau of Mines, Tucson, Ariz. 1916, Assayer, Penn-Arizona Development Co., Suijotoa, Ariz.

Present position—1916 to date: Supt. and Local Mgr., Penn-Arizona Development Co.

Percival St. John Dixon, New York, N. Y.

Proposed by William H. Merrett, Walter McDermott, L. H. Cooke.

Born 1881, Wandsworth, England. 1899, Matriculated, London Univ. 1905, Grad., Royal School of Mines, S. Kensington, England. 1905, Associate, Royal School of Mines in Mining. 1906, Associate, Royal School of Mines in Metallurgy. 1901, London Univ., in Engineering (Metallurgy), B. Sc. 1906–08, Akrokerri (Ashanti) Gold Mines, W. Af. 1908–09, Surveyor, Broomarri Mines, Ltd., Gold Coast, W. Af. 1910, Literary Collaboration with Prof. Cooke of Royal School of Mines. 1910, Reporting on mines with Norton Griffiths, Bruce Marriett & Co., S. Af. 1910–11, Asst., Naraguta (Nigeria) Tin Mines, Northern Nigeria. 1912, Reporting on copper mines with E. D. McDermott, Russia. 1913, Examination and reporting copper mines and mill with Pellea-Harvey & Co., Wales. 1914–16, Frontino and Bolivia (S. A.) Gold Min. Co., Republic of Colombia, S. Amer.

Present position: Disengaged.

Charles William Drury, Kingston, Canada.

Proposed by S. F. Kirkpatrick, William Campbell, E. J. Hall.

Born 1887, Kingston. 1900–05, Kingston Collegiate Institute (Honour Matriculation). 1905–09, Queens Univ., B. Sc. 1909–10, Columbia Univ., A. M. 1912 and 1914, summers, Columbia Univ. 1915–16, Columbia Univ. 1908, summer, Chemist, Canadian Copper Co., Copper Cliff, Ont. 1909, Chemist, Lake Superior Power Co., Helen Mine, Ont. 1907–08 and 1908–09, sessions, Demonstrator in Chemistry, Queen's Univ. 1910, Supt., Electro Steel Co., Welland, Ont. 1911–15, Lecturer in Mining and Metallurgy, Queen's Univ.

Present position—1916 to date: Asst. Prof. of Metallurgy, Queen's Univ.

George Hill Dudley, Morenci, Ariz.

Proposed by Frank W. McLean, R. W. Prouty, A. S. Livingston.

Born 1877, Killingsworth, Conn. 1897, Morgan High School, Clinton, Conn. 1899, Eastman Business College, Poughkeepsie, N. Y. 1899, short period, nights only, Pratt Institute, Brooklyn, N. Y. 1899–1901, Bookkeeper and paymaster, Stickney Heating Co. 1901–02, Manufacturer, Asbestos pipe and boiler covering, Brooklyn. 1902–03, Sampling and Assaying Dept., Arizona Copper Co., Clifton, Ariz. 1903–06, Bookkeeper, Arizona Copper Co., Clifton, Ariz. 1906–12, Mgr., Standard Copper Mines, Metcalf, Ariz. 1912–16, Div. Supt., Detroit Copper Min. Co.

Present position: Foreman, Ryerson and Yankie Mines, Detroit Copper Min. Co

Randolph Churchill Eisenhauer, Basin, Mont.

Proposed by Hennen Jennings, Charles Kammerer, H. C. Perkins.

Born 1884, Lockeport, Canada. 1896–99, Lockeport High School. 1909–14, Min. Dept., Univ. of Cal., B. S. 1899–1903, Instructor, Mathematics, High School. 1903–06, Salesman, J. D. Baker Co. and H. Wilkinson Co., Boston, Mass. 1906–09;

Construction work, Joseph McNeil, Supt. 1911-14, while attending college, Mathematics Instructor, Boon's Univ. School, Berkeley. 1912, summer, Drafting-room, Standard Oil Co., Point Richmond, Cal. 1913, summer, Miner, North Star, Grass Valley, Cal. 1914, Assayer and Surveyor, Midas Mine, Knob, Cal. 1914-16, Asst. Engr., Conrey Placer Mining, Ruby, Mont.

Present position: Asst. Engr., Comet Mine.

Howard Sheffer Estill, Wayland, Floyd Co., Ky.

Proposed by Howard N. Eavenson, E. O'Toole, John G. Smyth.

Born 1879, Lexington, Ky. 1895-98, Grad., Virginia Military Institute, Lexington, Ky. 1898-1900, Land surveying, Lexington, Ky. 1900-01, Lombard & Clay Lumber Co., locating and building railroad. 1900-02, Levelman, railway surveys, Chesapeake and Ohio Ry. Co. 1902, Draftsman, C. & O. Ry. Co. 1902, Transitman, location and construction railway, C. & O. Ry. Co. 1902-05, Asst. Res. Engr., Res. Engr., Asst. Div. Engr., Construction Dept., C. & O. Ry. Co. 1905-06, Asst. Engr., M. of W. Dept., C. & O. Ry. Co. 1906, Asst. Div. Engr., Div. Engr., M. of W. Dept., C. & O. Ry. Co., Huntington Div. 1906-07, Surveying coal lands, Eastern Kentucky and making coal mine surveys for Col. J. C. C. Mayo. 1907-08, City Engr., Paintsville, Kentucky, building brick streets. 1908-11, Pres. and Genl. Mgr., Mining Coal, Middle Creek Coal Co. 1911-13, Land surveying, mine surveying coal reports for J. C. C. Mayo. 1913-14, Chief Engr., Elk Horn Coal Corporation, Wayland Div.

Present position—1914 to date: Chief Engr., Elk Horn Coal Corporation.

Walter French Evans, So. Bethlehem, Pa.

Proposed by J. A. Van Mater, Lewis G. Rowand, J. H. Polhemus.

Born 1884, Pittsburgh, Pa. 1907, Lafayette College, E. M. 1907-16, Mine Surveyor, The New Jersey Zinc Co., Franklin, N. J.

Present position: Supt., The New Jersey Zinc Co. (of Pa.) at Friedensville, Pa.

John J. V. Forbes, Pittsburg, Kans.

Proposed by A. C. Terrill, V. H. Manning, W. R. Crane.

Born 1885, Shamokin, Pa. 1903-05, Grad., Keystone State Normal School, Kutztown, Pa. 1905-07, Teacher in schools, Shamokin, Pa. 1907-11, Grad., School of Mining, Pennsylvania State College, State College, Pa. 1903-05, summers, mined coal Burnside Colliery, Pittsburgh and Reading Coal & Iron Co., Shamokin, Pa. 1907-11, summers, worked on engineering corps, Pittsburgh and Reading Coal & Iron Co. 1911-13, Steam Engr., Carnegie Steel Co. 1913-14, Safety Engr. and Min. Engr., Provident Coal Co., St. Clairsville, O.

Present position: Junior Min. Engr., Bureau of Mines.

DeWayne Loomis France, West Haven, Conn.

Proposed by Arthur F. Taggart, J. F. McClelland, L. W. Bahney.

Born 1891, Paris, France. 1913, Sheffield Scientific School, Ph. B. 1915, Sheffield Scientific School, E. M. 1915, U. S. Metals Ref. Co., Chrome, N. J. 1915, Richards and Locke, Boston. 1916, Willet Sears Co., Boston.

Present position: Research Work, Scovill Mfg. Co.

Frederick Charles Frey, West Sumatra, Dutch East Indies.

Proposed by Pope Yeatman, H. H. Webb, R. M. Catlin.

Born 1872, Franktown, Nev. Public School education. 1894, Univ. of Nevada, B. Sc. 1894-97, Underground and surface surveyor, Creston-Colorada, Amarillas and Verde, Minas Prietas Sonora, Mexico. 1897-1905, Sampler, Surveyor, Mine Captain, Underground Mgr., Acting Mgr., Simmer and Jack Min. Co., Germiston, Transvaal, S. Af. During the Boer War was employed for a few months as assayer and surveyor with the Yellow Aster Min. Co., Randsburg, Cal. 1900, Mgr., Inez Mine, Hartly Hills, Rhodesia. 1902-05, assisted in restarting the Simmer and Jack Mine after it had been closed down during the war, and was employed there till latter part of 1905. 1905 (latter part), Mgr., South Rose Deep, South Geldenhuis Deep, Rand Victoria Mine and Rand Victoria East Min. Cos. 1906, the South Rose Deep, Ltd., South Geldenhuis Deep, Ltd., Rand Victoria Mines, Ltd., Rand Victoria East, Ltd. were amalgamated into the Simmer Deep, Ltd., and was appointed Genl. Mgr. of this amalgamation. 1911-13, Cons. Engr., Messrs. Erdmann & Sielaken, Batavia, Java.

Present position: Genl. Mgr., Mijnbouw Maatschappij Redjang Lebong.

Javier Gandarillas, Santiago, Chile, S. Amer.

Proposed by W. S. Bishop, William E. Milligan, John P. Chadwick.

Born 1875, Santiago de Chile. 1885-88, Secondary studies in Institute Nacional, Santiago, Chile. 1889-92, Id. in Lycé Fauson de Saily, Paris. 1892, Bachelier enseignement moderne. 1892-97, Entered the Ecole de Génie Civil of Ghent (Belgium), C. E. 1899-1904, interested in a farming business in Huasco, Atacama. 1904-05, Prospecting nitrate grounds in Taltal. 1906-09, Cons. Engr., Santiago. 1906, Member of the Board of Directors, Sociedad Nacional de Minería, Santiago. 1907, Published a translation, revised and enlarged of Sunper and Michels, "Nitrate Industry of Chile" (in German). 1906, interested in mining in the province of Santiago. 1909-15, Deputy to the Congress for the province of Atacama and State Minister of Industry and Public Works. 1915, Past President, Institute of Civil Engineers of Chile. Present position: Interested in mining business and farming in Vallenat.

Hugo E. R. Garde, Malmberget, Sweden.

Proposed by J. H. Jowett, George A. Howells, George T. Cousins.

Born 1889, Stockholm, Sweden. 1908-11, Grad., Royal Academy of Mines, Stockholm, Sweden, E. M. 1912, Asst. in mining, Royal Academy of Mines, Stockholm, Sweden.

Present position—1913 to date: Min. Engr., Luossavaara-Kirunavaara Co.

James Henry Gardner, Tulsa, Okla.

Proposed by H. A. Wheeler, A. G. Heggem, M. M. Valerius.

Born 1883, Sonora, Ky. 1904, Univ. of Kentucky, B. S. 1906, Univ. of Kentucky, M. S. 1910, George Washington Univ., Ph. D. 1904-06, Kentucky Geol. Survey. 1906-10, U. S. Geol. Survey. 1911, Kentucky Geol. Survey. 1912, Pennsylvania Geol. Survey.

Present position—1913 to date: Oil Geologist and Appraising Engr.

Harold Sinclair Gay, Alhambra, Cal.

Proposed by Walter A. Richelsen, M. J. Elsing, George Kingdon.

Born 1880, Hawaiian Island. 1895-99, Redlands Union High School. 1899-1904, Stanford Univ., B. A. in Mining. 1904, Tigre Poorman Mine, Burke, Idaho. 1905, working for myself. 1906, Assayer, Live Oak Min. Co., Globe, Ariz. 1906, Surveyor, Velardena Mines of American Smelters Securities Co. 1907, working for myself. 1907-09, Surveyor, Clerk, Paymaster, Sampler, Minas Tecolotes y Anexas, Santa Barbara, Mex. (Hcla Bonanza, Zac., Mex.) 1909-11, Supt., Minas Bonanza y Anexas. 1911, Foreman, Terneras Mine, Velardena, Durango, Mex., American Smelters Securities Co. 1911-15, working for myself.

Present position—1915 to date: Engr., Cananea Cons. Copper Co.

James T. Gratiot, Denver, Colo.

Proposed by M. S. MacCarthy, Walter E. Burlingame, P. M. McHugh.

Born 1882, St. Louis, Mo. Practical education. 1897-1904, Office Boy, Mine Clerk, Travelling Mine Clerk, Travelling Auditor, Supt. of Time Distributing Dept., Chief Clerk to Genl. Supt. and Travelling Salesman, Colorado Fuel & Iron Co. 1904-05, In retail coal business. 1905, Minnequa Steel Works, Colorado Fuel & Iron Co. 1905-06, McPhee & McGinnity Lumber Co. 1906 to date, Salesman, Mechanic, Mgr. of R. R. Dept., Supt. of Shop, Mgr. Electrical Dept. and for past four years as Mgr. Machinery Dept., embracing and having supervision over all the above and the mining dept., Fairbanks, Morse & Co.

Present position: Mgr., Machinery Dept., Fairbanks, Morse & Co.

George L. Grosvenor, Mayér, Ariz.

Proposed by F. K. Brunton, E. C. King, G. M. Colvocoresses.

Born 1874, Jackson, Mich. To 1890, High School. Balance study and practical experience. 1902-07, Cananea Cons. Copper Co. 1910-12, Cananea Cons. Copper Co. 1912-16, Contracting. 1916, Cons. Arizona Smelt. Co., Humboldt, Ariz.

Present position: In charge of engineering and construction, Great Western Smelters Corp.

William George Harris, Salino, B. C., Canada.

Proposed by W. S. Grether, A. H. Cronk, C. C. Conover.

Born 1878, California. Common Schools. 1893-94, High School, Chewelah, Wash. Several years home study regarding mines and milling. 1896-1909, Hydraulic and hydraulic elevator mining in summer time, Idaho and British Columbia. Underground work in winter time. Hand and machine work, timbering, tool sharpen-

ing, shift boss, etc. 1909-14, Timberman, machine man, shaft man, etc., Morning Mine, Mullan, Idaho.

Present position: General mine foreman, Leessee, Hudson Bay Zinc Co., Ltd.

Burton Hartley, Tulsa, Okla.

Proposed by A. F. Truex, L. C. Snider, John J. Doyle.

Born 1890, Philadelphia, Pa. 1908-12, Lehigh Univ., So. Bethlehem, Pa., E. M. 1911, Porcupine Gold Mines Co., Porcupine, Ont. 1912, American Blaver Co., Detroit, Mich. 1913, Compania Mexicana de Petroles, "El Aguila," S. A., Tampico, Mex. 1914, Consulting oil work in Pennsylvania, West Virginia, Ohio, Louisiana and Cuba. 1915-16, Roxana Petroleum Co. of Okla.

Present position—1916 to date: Geol., Cosden Oil and Gas Co.

Ira Sankey Heller, So. Orange, N. J.

Proposed by Lewis G. Rowand, Thomas T. Read, George C. Stone.

Born 1876, Philadelphia, Pa. Public Schools. Baltimore Polytechnic Inst. 1905, Newark Technical School. 1901-04, Wetherill Separating Co. 1904-05, Alexander Trand Co. 1904, The New Jersey Zinc Co. 1905-06, Elizabeth Copper Co.

Present position—1906 to date: Chief Draughtsman, Milling Dept., The New Jersey Zinc Co.

Richard H. Henderson, Bartlesville, Okla.

Proposed by Henry B. Taylor, Kurt Stock, Harold C. Price.

Born 1891, Buchanan, Mich. 1915, four years work, Univ. of Colorado.

Present position—1915 to date: Asst. Chemist, Bartlesville Zinc Co.

James V. Howe, Tulsa, Okla.

Proposed by C. R. Forbes, H. A. Buehler, H. A. Wheeler.

Born 1880, Rowlesburg, W. Va. 1904, Univ. of West Virginia, B. S. C. E. 1907-08, Special, Cornell Univ. 1904-06, Engr., South Penn Oil Co., Mannington, W. Va. 1907, Mine Transittman, Ellsworth Coal Co., Ellsworth, Pa. 1912-13, Engr., Penn-Mex. Fuel Co., Tampico, Tamps., Mex. 1913-15, Cons. Oil Geol., in private practice in W. Va. and Kentucky.

Present position: Geol., Oklahoma National Gas Co.

Joseph Clyde Hoyt, Jerome, Ariz.

Proposed by A. P. Thompson, Robert E. Tally, Charles P. Berkey.

Born 1882, Pawnee City, Nebr. 1906-10, Univ. of Arizona, B. S. 1900-06, General Mining, United Verde Copper Co., Jerome, Ariz. 1910-13, Min. Engr., United Verde Copper Co., Jerome, Ariz. 1913-14, Min. Engr., North Star Tunnel and Development Co., Tonopah, Nev.

Present position—1914 to date: Consulting Mining Engineer.

L. L. Hutchinson, Tulsa, Okla.

Proposed by H. A. Wheeler, G. H. Cox, M. M. Valerius.

Born 1876, Marion Cokes. 1907, Oklahoma Univ., A. B. 1908, Yale Univ., M. S. 1905-10, Oklahoma Geol. Survey. 1910-12, Cons. Geol., Tulsa. 1912, Independent Operator (Oil).

Present position: Oil Producer and Geologist.

Frank G. Janney, Garfield, Utah.

Proposed by C. M. Stimpson, O. J. Salisbury, Paul T. Boise.

Born 1885, Salt Lake City, Utah. 1902-04, Univ. of Utah, Salt Lake City. 1905, Asst. Chemist, Copperton Plant, Utah Copper Co. 1907, Chemist, Magna Plant, Utah Copper Co. 1908, Met. Engr., Utah Copper Co. 1909, Genl. Mill Foreman, Utah Copper Co. 1911, Supt. of Arthur Plant, Utah Copper Co.

Present position—1914 to date: Genl. Supt. of Mills, Utah Copper Co.

Joseph Roebling Jarvis, New York, N. Y.

Proposed by Alexander C. Humphreys, C. G. Roebling, F. W. Roebling, Jr.

Born 1882, Philadelphia, Pa. 1907, Grad., Stevens Institute of Technology, Mech. Engrg. One year, Biltmore Forest School, Bachelor of Forestry. Two years experience in forestry. Eighteen months with F. R. Meier, New York, N. Y., both in office and field experience on 4 forest surveys in Tennessee and Virginia and six months at Lytle Creek forest, planting station of the U. S. Forest Service, San Bernardino Co., Cal. Two years structural steel draughting room experience with New Jersey Bridge Co. and Post and McCord, New York City. Five years mining experience in Idaho, one year in employ of Meadow Creek Gold Placer Co.

Present position: Hay Fork Mine, Idaho City, Idaho.

Thomas B. Johnson, El Paso, Tex.

Proposed by D. W. Reckhart, William D. Gordon, W. W. Rose.

Born 1858, Monterey, Cal. 1872, Grad., High, Public Schools, Salinas City, Cal. Three years, St. Mary's College, San Francisco, Cal. Took a mining course for two years and then went to accept a position as Assayer and Chemist, Ventanas Min. Co., Durango, Mex. 1887, Assay and Chemical Assay Office for D. W. Reckhart. Government Assayer, U. S. Government. After the expiration of this term, went before a Board and passed a thorough examination. Became the U. S. Government Assayer at the Port of El Paso for twelve years. Genl. Mgr., Mgr. and Supt. on many mines, the last one, The Peabody Copper Co., Cochise Co., Ariz. At present, Pres., Las Ardorga Min. Co., Parral, Mex.; Vice-Pres., Monte Crisco Sonora Min. Co.; Pres., Cinco De Mayo Min. Co.; Genl. Mgr., Corona Min. Co. and part owner, San Ignacio Min. Co., Ahnuada, Mex. Sole owner, Johnson Assay Co., El Paso, Tex.

Present position: Mine Reporter and Mining Stock Broker.

Fred Lucian Johnston, Denver, Colo.

Proposed by P. M. McHugh, Fred H. Bostwick, George A. Kennedy.

Born 1878, Hellers Comers, Ind. 1905, Michigan Agricultural College, B. S. in Mech. Engrg. Two years, Draftsman and Inspector on construction work—W. G. Fargo, Jackson, Mich. Four years, Designer, structural steel, Purdy and Henderson, Chicago, Ill. One year, City Engineer's office, Albuquerque, N. M.

Present position: Mech. Engr., The Western Chemical Mfg. Co.

James Dyer Jones, Gary, Ind.

Proposed by Alexander K. Hamilton, H. J. Freyn, H. A. Brassert.

Born 1881, Pittsburgh, Pa. 1899, Pittsburgh Academy. 1907-09, Draftsman, Minnesota Steel Co. 1909-12, Engr., M. H. Treadwell Co. of Ill. 1912-15, Asst. Chief Engr., Algoma Steel Co., Sault Ste. Marie, Ont. 1915-16, Chief Engr., Algoma Steel Co., Sault Ste. Marie, Ont. 1900, Tracer, S. Anderson, Pittsburgh, Pa. 1900-01, Draftsman, S. V. Hubert & Co., Pittsburgh. 1901-05, Draftsman, National Tube Co., Wheeling, W. Va.; Illinois Steel Co., So. Chicago; Lackawanna Steel Co., Buffalo; Jones & Laughlin, Pittsburg; National Tube Co., McKeesport, Pa.; Milliken Bros., New York.

Present position: Chief Engr., Indiana Steel Co.

John Knox, Calumet, Mich.

Proposed by James MacNaughton, C. H. Benedict, Claude H. Cooper.

Born 1876, Kilwinning, Scotland. 1899, Michigan College of Mines, E. M. 1897, Asst. Engr. on construction of Canadian Pacific. 1899, Min. Engr., Centennial Min. Co., Calumet, Mich. 1900, Engr., Nickel Copper Co. of Ontario. 1901, Engr., Fimountain Min. Co., Fimountain, Mich. 1904, Engr., Sterling Salt Co., Cuylerville, N. Y. 1907, Underground Supt. of Conglomerate Mine, Calumet & Hecla Min. Co.

Present position: Genl. Supt., Calumet & Hecla Min. Co.

Arthur L. Lee, Homestead, Ore.

Proposed by Angus Mackay, Halstead Lindsley, Thayer Lindsley.

Born 1891, Loveland, Colo. 1908-11, High School, Greeley, Colo. 1911-15, Colorado School of Mines, Golden, Colo. 1915, Asst. Engr., Utah Fuel Co.

Present position—1916 to date: Engr., Homestead-Iron Dyke Mines Co., Inc.

Henry Leighton, Pittsburgh, Pa.

Proposed by H. Ries, H. B. Meller, S. L. Goodale.

Born 1884, Canandaigua, N. Y. 1902-06, Cornell Univ., A. B. 1906-08, special work, Cornell Univ. 1906-08, Instructor in Geology, Cornell Univ. 1907, summer, Field Asst., U. S. Geological Survey. 1908-10, Asst. in Economic Geology, New York State Museum. 1910-12, Instructor in Economic Geology, School of Mines, Univ. of Pittsburgh. 1912-13, Asst. Prof. of Economic Geology, Univ. of Pittsburgh.

Present position—1913 to date: Prof. of Economic Geology, School of Mines, Univ. of Pittsburgh.

Milton Sigfried Lindholm, Bisbee, Ariz.

Proposed by Arthur Notman, Roger T. Pelton, Gerald F. G. Sherman.

Born 1887, Ortonville, Minn. 1907-11, School of Mines, Univ. of Minnesota, E. M. 1911-12, Millman and Assayer, Miller Gold Min. Co., Leadville, Colo. 1912-13, Assayer and Chemist, Yak Min. Mill. and Tunnel Co., Leadville, Colo.

1913-14, Engr., Copper Queen Copper Co., Bisbee, Ariz. 1914-15, Engr., Société Internationale Forestière et Minière du Congo, Kasai, Congo Belge, W. Africa. 1915-16, Engr. in charge, Companhia de Pesquisas Mineiras de Angola, Lunda, Angola, W. Africa.

Present position: Engr., Copper Queen Copper Co.

Edgar W. McCrary, Tulsa, Okla.

Proposed by Anthony F. Lucas, H. Foster Bain, H. A. Wheeler.

Born 1879, Nashville, Ark. 1900-04, Univ. of Arkansas, Civ. Engrg. and Geology. 1904-07, Topographer, U. S. Geological Survey. 1907-08, In charge of drainage surveys, Illinois Geological Survey. 1908-09, Partner, W. T. Griswold, Oil and Gas, Geology. 1909 to present, Oil and Gas Geologist. Principal connections, Philadelphia Co., Pittsburgh, Pa., Guffey, Gillespie and Pitcairn, Pittsburgh, Pa.

Present position: Geol., Tulsa, Okla.

Elliott Steele McCurdy, San Francisco, Cal.

Proposed by Howard D. Smith, C. W. Merrill, F. W. Bradley.

Born 1887, Wooster, O. 1903, Two year Scientific at Princeton. 1905, Min. Engr., Columbia. 1903-05, Pennsylvania R. R. (P. B. & W. div.). 1906, Upper Potomac & Pittsburgh Coal Co., West Virginia and Pennsylvania. 1907, working in Cal., Colo., Minn., Mont. and Nev. 1908, leasing, Utica Min. Co., Cal. and Nev. 1909, Utica Min. Co., Cal. 1909, Bridger Coal Co., Mont. 1910, Treasure Min. Co., Cal. 1910-17, with Utica and Treasure Min. Cos. and private practice.

Present position: Engr. Utica Min. Co.; Secy. and Mgr., Treasure Min. Co.

Charles McKnight, Jr., Pittsburgh, Pa.

Proposed by Stephen L. Goodale, Robert M. Black, Fred. Crabtree.

Born 1891, Sewickley, Pa. 1909, Lawrenceville School, N. J. 1916, Univ. of Pittsburgh, M. E. 1909-10, Midland Steel Co. 1910, S. Jarvis Adams Co. 1911-12, Carnegie Steel Co. 1913, Youngstown Sheet & Tube Co.

Present position—1915 to date: Asst. Met., Carbon Steel Co.

William Laughlin McLaine, Lost Hills, Cal.

Proposed by Rea E. Maynard, E. B. Kimball, D. M. Folsom.

Born 1873, Volcano, Amador Co., Cal. St. Mathews School, San Mates, Cal. 1892-97, Stanford Univ., Mech. Engrg. and Geology. 1898, Engr., McLaughlin, Est. Co. 1899, Crocker National Bank, San Francisco, Cal. 1900-01, Vishner Oil Co., New River Fields, Cal. 1902, Assaying, Butte Co., Cal. 1903-04, Supt., National Oil and Transportation Co. 1905-06, Jober Martin & Co., Contractors and Engineers. 1906-07, Supt., Construction, California Petroleum Refineries. 1907-11, Supt. and Mgr., Associated Pipe Line Co. 1911-16, Supt., Universal Oil Co., Vulcan Oil Co.

Present position—1916 to date: Supt., Lost Hills Development Co.

John James Magan, Westminster, London, S. W., England.

Proposed by Harry March, Frederick C. March, R. O. Ahlers.

Born 1879, So. Kensington, London, England. 1886-94, Oratory Grammar School. 1894-98, Kings College, London. 1906-10, Mechanics and General Engineering Works, Nottingham. 1910-12, Min. Engrg., Clifton, Nottingham. 1912-16, Min. Engr. (General Managing on mines owned by Harding Bros.) (Engineers Westminster Ltd., of which company am now Director, embracing the mining of Copper-Wolfram-Tin and Uranium).

Present position: Consulting staff (Engineering Dept.) of Harding Bros. (Engineers Westminster) Ltd.

Joseph H. Manwaring, Butte, Mont.

Proposed by C. W. Goodale, F. A. Linforth, Paul Billingsley.

Born 1889, Menomonie, Wis. 1909-13, Grad., Montana State School of Mines, E. M. 1909-10, vacations and two nights a week, Mining, Butte-Ballaklara Mine. 1911-12, Mining, Pennsylvania Mine, Anaconda Copper Min. Co. 1913, Shift Boss, Butte-Ballaklara Copper Co.'s mine. 1914, Engr., Butte-Ballaklara Mine.

Present position—1915 to date: Supt., Butte-Ballaklara Copper Co.

E. N. Barbot de Marney, Petrograd, Russia.

Proposed by T. J. Jones, H. H. Knox, K. Koliashnikoff.

Born 1868, Petrograd. 1890, General—Larinsky States, Gymnasium at Petrograd. 1896, Technical—Imperial Institute of Mining Engineers, Petrograd. 1896-98,

Mgr. of the States Bakalsky Iron Ores Mines, South Oural. 1898-1901, Mgr. of the Gold and Platinum Mines of Count Chonvaloff, Oural. 1901-06, Asst. of Mineralogy and Geology of the Imperial Min. Inst. and Docent of Geology of Academy of Military Academy, Petrograd. 1906-08, Mgr., Director of Neviansky Iron Work and Gold Mines, Oural. 1908-12, Director of Dept. of Gold Mines, Ministerium of Trade and Commerce.

Present position—1912 to date: Member of the Board of Lenskoie Gold Mines, Chairman of Board of South Siberian Gold Mines Co., Managing Director of the Board of Olekma Gold Mines Co., Member of the Board of the Komarousky Iron Work Co., South Oural.

Ernest Marquardt, New York, N. Y.

Proposed by R. M. Raymond, Charles P. Berkey, W. A. Scheuch.

Born 1891, Elberfeld, Germany. 1904, Grad., Public School, New York. 1907, Grad., Townsend Harris Hall. 1907-09, Freshman and Sophomore years, College of the City of New York. 1909-12, Columbia Univ., School of Mines, E. M. 1912-14, Churn Drill Sampler, Churn Drill Helper, Chemist, File Clerk, and Asst. Min. Engr. and conducted several investigations for the water supply of the mine in Chile, Chile Exploration Co. 1914-15, Geological, Engineering, and Topographical work in connection with the exploration of certain China oil fields, Standard Oil Co. of New York.

Present position—1916 to date: Geo., Associated Geological Engineers.

Otto F. Meister, St. Louis, Mo.

Proposed by Herman Garlichs, Gustave C. Taussig, W. E. McCourt.

Born 1876, Washington Univ., St. Louis, Mo. 1877, Assayer, St. Louis Smelt. & Ref. Co., Leadville, Colo. 1879, Mining in Gilpin Co., Colo. 1880, with my brother, Herman C. Meister, started a zinc smelter, Collinsville, Ill. Prior to 1912, Secy. and Treas., Collinsville Zinc Co., Collinsville, Ill.

Present position—1912 to date: Pres., Collinsville Zinc Co.

Gilbert Frank Metz, Hannibal, Mo.

Proposed by Fred T. Agthe, Ray E. Hoffman, C. R. Forbes.

Born 1889, Omaha, Nebr. 1908, Grad., McK. High School, St. Louis, Mo. 1914, Grad., School of Mines, Univ. of Missouri, B. S. 1910-11, Chemist and Testing Engr., Union Electric Light & Power Co., St. Louis, Mo. 1911-14, summers, Foundation Co. of N. Y., Union Electric, St. Louis, State Geological Survey of Missouri. 1914-17, Atlas Portland Cement Co.

Present position: Plant Engr., Atlas Portland Cement Co.

Archie Leon Monteith, Low Moor, Va.

Proposed by R. A. Lipscomb, R. B. Ladoo, F. U. Humbert.

Born 1892, Goshen, Va. 1908, Clifton Forge Seminary. Similar to 4 years high school. Now taking mining engineering with International Correspondence Schools. 1912, Locomotive Crane Engr., Low Moor Iron Co., Low Moor, Va.

Present position—1913 to date: Supt., underground Limestone Quarry, Low Moor Iron Co.

M. J. Munn, Tulsa, Okla.

Proposed by C. W. Washburne, George Otis Smith, E. W. Shaw.

Born 1874, Bodcaw, Ark. Two years, Asst. Topographer, U. S. Geol. Survey. Asst. Geol., Geological Dept., U. S. Geol. Survey. Detailed for the investigation of the geology of oil and gas fields in Ohio, Pennsylvania, West Virginia and other parts of the Appalachian oil field.

Present position: Chief Geol., Gulf Oil Corp'n.

Komatsuchi Ohdaira, Niihama, Iyo Prov., Japan.

Proposed by Charles E. Locke, Edward E. Bugbee, Michiya Hiraoka.

Born 1859, Fukuyama, Japan. 1895, Imperial Univ. of Tokyo, B. L. 1895-99, Member of the staff of the Mine Inspection Office of the Japanese Government. 1899-1905, Chief of the Mine Inspection Office of the Japanese Government. 1905-11, Chief of the Min. Dept., Fujita Co., Osaka, Japan.

Present position—1911 to date: Mgr., Besshi Copper Mine, K. Sumitomo, Osaka, Japan.

Robert W. Olinger, New York, N. Y.

Proposed by Harris K. Masters, E. Gybbon Spilsbury, R. M. Raymond.

Born 1884, Emporia, Kans. 1899-1904, New Mexico College of Agriculture and Mechanic Arts, Mesella Park, N. M. 1904, Mgr., Ore Buying House of Gualterio C. Palmer, Zecatecas, Mex. 1912, Ore Buyer, American Smelt. & Ref. Co., Chihuahua, Mex.

Present position—1916 to date: Ore Buyer, W. R. Grace & Co.

Albert Lee Owen, Santa Rita, New Mexico.

Proposed by John M. Sully, Horace Moses, D. C. Jackling.

Born 1867, Chillicothe, Mo. High School education. 1889-99, Mine operator Joplin, Mo. 1899-1917, Mine operator, New Mexico.

Present position: Vice-Pres. and Supt., Royal John Min. Co., Black Range, N. M.

Daniel Palacios, (Olmedo) Chile, S. America.

Proposed by Mark R. Lamb, Thomas T. Read, William H. Shearman.

Born 1873, Valparaiso, Chile. 1908, Univ. of Chile, diploma of Topographical and Min. Engr. During 12 years was Manager of the Compania Establecimiento de Fundicion de Tiltit (Chile). The last five years was consulting engineer especially copper, gold and iron. Worked own copper-mine and owner of the Freirina iron-mines in the north of Chile. Member of the Institute of Engineers of Chile.

Present position: Consulting Engineer.

Leroy A. Palmer, San Francisco, Cal.

Proposed by Joseph Jensen, H. C. Parmelee, Arthur J. Hoskin.

Born 1879, Lockport, Ill. 1895, High School, Grand Rapids, Mich. 1903 to date, Correspondence courses in civil and mining engineering. 1903, Salt Lake R. R., Salt Lake City, Utah. 1904, Mucker, miner, etc., Continental Mines and Smelters, Salt Lake City, Utah. 1905-06, Millman, Newhouse Mines and Smelters. 1906, Millman, Honerise Min. Co., Stockton, Utah. 1908, Engr. Corps, Steptoe Smelt. & Min. Co., McGill, Nev. 1908, Millman, Boston Cons. Min. Co., Garfield, Utah. 1908, Solution man, Jennie Gold Min. Co., Gold Springs, Utah. 1909-10, Transitman, Engr. Corps, Salt Lake City, Utah. 1911-16, Mineral Examiner, U. S. General Land Office and U. S. Forest Service.

Present position: Mineral Examiner, U. S. General Land Office.

William Roy Palmer, Clarkdale, Ariz.

Proposed by T. C. Roberts, A. T. Coston, Robert E. Tally.

Born 1884, Fairbury, Nebr. 1890-1900, Public and High Schools, Liberty, Nebr. 1900-06, Academy and Univ. of Nebraska, B. Sc. in Electrical Engr., Lincoln, Nebr. 1916, Alexander Hamilton Institute. 1906-07, Testman, General Electric Co., Schenectady, N. Y. 1908-09, Designing, repairing and building autos, O. P. Fritchle, Denver, Colo. 1909, Detailing steel, Karl Burkhardt, Denver. 1909-10, Design irrigation, etc., Geo. Wall, Denver. 1910-11, Asst. Engr., Grand River, Meeker and Salt Lake. 1911, Design power plant, Pueblo Traction and Lighting Co., Pueblo. 1911-12, Construction Engr., Arkansas Valley Railway Light and Power Co., Pueblo. 1912-13, Construction Engr., Tennessee Power Co., Nashville, Tenn.

Present position—1913 to date: Asst. to Chief Engr., United Verde Copper Co.

George H. Parsons, Hayden, Ariz.

Proposed by D. C. Jackling, L. S. Gates, D. D. Moffat.

Born 1890, Salt Lake City, Utah. 1908-11, Univ. of Utah, School of Mines. 1911-12, Millman, Utah Copper Co., Magna, Utah. 1912-14, Asst. Chemist, Utah Copper Co., Magna, Utah.

Present position—1914 to date: Chief Chemist, Ray Cons. Copper Co.

Alfred Pearson, Jr., Somerset, Pa.

Proposed by John G. Smyth, Frank Haas, Robert E. Rightmire.

Born 1880, Newburyport, Mass. 1896-98, general science, Rhode Island State College. Technical knowledge acquired by home study assisted by International Correspondence Schools. 1901-02, Rodman, Prov. Div., New York, New Haven and Hartford R. R. 1902-05, Transitman, Clearfield Bituminous Coal Corp'n. 1905-08, Draftsman, H. C. Frick Coke Co. 1908-11, Chief Draftsman, Somerset Coal Co. 1911-13, in private civil practice, Putney, Vt.

Present position—1913 to date: Div. Engr., Penn. Div., The Consolidation Coal Co. R. R. M. of W.; triangulation property and topographical surveys in connection with 5000 A. coal field-opening and surveying of mines, prospecting and construction,

drafting, mine mapping and construction drafting and general oversight of field work as assistant; civil engineering and surveying, Div. Engr. in charge of maintenance of mines and construction in connection with new development.

Ralph Grover Perry, Lead, So. Dak.

Proposed by R. W. Mackey, A. J. M. Ross, Allan J. Clark.

Born 1892, Joliet, Ill. 1897-1909, Grammar and High School, Joliet, Ill. 1911-15, Univ. of Illinois, B. S. in mining engineering. 1914, summer, Laborer, Homestake Min. Co. 1915, General underground experience, Homestake Min. Co.

Present position: Mine Sampler, Homestake Min. Co.

John G. Pew, Pittsburgh, Pa.

Proposed by Forrest M. Towl, I. C. White, S. A. Taylor.

Born ———. Since 1886 holding different positions such as City Supt., Supt., Asst. Mgr., Vice-Pres. and Pres., Peoples Natural Gas Co. Asst. Mgr., Hope Natural Gas Co., W. Va. Vice-Pres., River Gas Co.

Present position: Pres., The Peoples Natural Gas Co.; Vice-Pres., Hope Natural Gas Co. of W. Va.; Pres., Marion Oil Co.

Grover C. Pidgeon, Gleeson, Ariz.

Proposed by Gerald F. G. Sherman, Y. S. Bonillas, J. B. Tenney.

Born 1884, Houtzdale, Pa. To 1901, Graded High Schools, Houtzdale, Pa. 1904, Mercersburg Academy. 1905-08, Pennsylvania State College. 1902-03, Electrical Dept., Pennsylvania Coal and Coke Co., Gallitzin, Pa. 1903, Weighboss, G. L. Whitehead Coal Co., Osceola, Mills, Pa. 1907, Engr., Cardiff Coal Co., Nettleton, Pa. 1908-09, Assayer, Calumet and Arizona Min. Co. 1909-10, Chainman and Instrument man, Arizona and Eastern R. R. 1910, Assayer, Inspiration Copper Co. 1910-11, Transitman on Preliminary R. R. Survey in Sonora. 1911, Assayer, Copper Queen Cons. Min. Co. 1911-12, Instrument man, E. P. and S. W. R. R. 1912-14, Engr. in charge of Construction and Location, A. T. and S. F. R. R., Coast Lines. 1914-16, Assaying Dept., Engrg. Dept. Efficiency Dept., Copper Queen Cons. Min. Co.

Present position: Supt., Tejon Min. Co.

William A. Pitt, Santiago de Cuba, Cuba.

Proposed by Charles Page Perin, Robert P. Bowler, C. M. Weld.

Born 1876, Gloucester, England. 1885, Chellenham High School. 1889-92, St. Johns College, London. 1893, Freemans School, Wales. 1894-95, Engrg. Dept., Taff Coal Co., N. Wales. 1895-97, British Colombia Co., Colombia, S. Amer. 1898-1901, Engrg. Dept. U. S. Government occupation, Cuba. From then to present date, Spanish American Iron Co., Cuba Eastern R. R. Co., Mantanamio Esetts Co., Eastern Steel Co., Juragua Iron Co.

Present position: Genl. Mgr., Cauto Min. Co.

Theodore Polhemus, Benton, Wis.

Proposed by J. H. Polhemus, J. A. Van Mater, J. N. Houser.

Born 1888, Roseville, N. J. 1907-11, Mass. Inst. of Tech., Civil and Sanitary Engrg. 1911-15, Contracting and Engineering work, Winnipeg, Manitoba. 1915-16, surveying, construction work, etc., Wisconsin Zinc Co., Platteville, Wis.

Present position—1916 to date: Supt., Langhorn Mine.

Lewis Braden Pringle, Bonne Terre, Mo.

Proposed by C. J. Adami, H. Rabling, L. A. Delano.

Born 1891, Quincy, Ill. 1897-1910, Public Schools, Quincy, Ill. 1910-12, Missouri School of Mines and Metallurgy, Rolla, Mo. 1912-13, Univ. of Wisconsin, Madison, Wis. 1906-07, Clerk, State Savings Loan and Trust Co., Quincy, Ill. 1912, summer, Draftsman and Topographer, Chi., Peoria and Quincy Traction Co., Quincy, Ill. 1913, Miner, Wisconsin Zinc Co., Platteville, Wis. 1913, Engr., Oliver Iron Min. Co., Hibbing, Minn. 1913-14, Asst. Geol., Cananea Cons. Copper Co. 1914, Oil Geol. & Leaser, independent and intermittently for Wm. C. Kennedy Oil Co., Sullivan, Ind. 1914, Draftsman, American Road Machinery Co., Fort Wayne, Ind. 1914-15, Draftsman, S. F. Bowser and Co., Fort Wayne, Ind.

Present position—1915 to date: Chemist, St. Joseph Lead Co.

Louis E. Reber, Jr., Jerome, Ariz.

Proposed by W. J. Mead, C. K. Leith, A. N. Winchell.

Born 1889, State College, Pa. 1906-07, Penn. State College, Engrg. 1909-14, General and Civil Engrg. and graduate work in geology, Univ. of Wisconsin, B. S.

in Civ. Engrg. 1914-16, Graduate work, Economic Geology and Mining, Yale Univ., Ph. D. 1910-12, Three field seasons as compassman and geologist, engaged in iron exploration work in the Lake Superior region for the Oliver Iron Min. Co. 1913, Field season as geologist with reconnaissance survey party in S. W. Oregon for Oregon Bureau of Mines and Geology (assisting Dr. A. N. Winchell). 1914, Field season as geologist in charge of copper exploration party in northern Wisconsin for Adair Development Co., Deerwood, Minn. 1915-16, about 10 months in Geol. Dept., Detroit Copper Co., Morenci, Ariz.

Present position: Geol., United Verde Copper Co.

Edwin R. Richards, Midvale, Utah.

Proposed by Corey C. Boyton, L. O. Anderson, E. A. Wall.

Born 1879, Palmer, Mich. 1899, High School, Idaho Springs, Colo. 1905, Colorado School of Mines, E. M. 1905-08, Chemist, Engr., Draftsman, etc. 1908-09, Supt., Big Stick Gold Min. Co. 1908-10, Engr. and Foreman, Pingineo Mines Co. 1909-10, Supt., Pingineo Mines Co. 1910-11, Supt., Guanajuato Amalgamated Gold Mines Co. 1911-12, various personal business. 1912-13 Engr., U. S. Smelt. Co. 1913-14, Supt., Churn Drilling, General Dev. Co. 1914-16, Supt., Buckhorn Mines Co.

Present position—1916 to date: Vice-Pres. and Mgr., Childers Leasing Co.

William Harry Risher, Huntington, Ark.

Proposed by D. H. Cadmus, H. A. Buehler, C. R. Forbes.

Born 1888, Brazil, Ind. 1906, Grad., Brazil High School. 1912, Grad., Purdue Univ., B. S. in Civ. Engrg. 1908-10, Asst. Engr., Coal Bluff Min. Co., Terre Haute, Ind. 1912-14, Asst. in Civil Engrg. Dept., Purdue Univ. 1914-16, Instructor in Civil and Min. Engrg. Dept., Missouri School of Mines. 1916, Resident Engr., Huntington District, Central Coal & Coke Co.

Present position: Resident Engr., Central Coal & Coke Co.

William E. Ruder, Schenectady, N. Y.

Proposed by C. V. Ferguson, W. R. Whitney, Henry M. Howe.

Born 1886, Stockdale, Pa. 1903, Grad., South Western State Normal of Pa. 1907, Pennsylvania State College, B. S. 1907, Research Laboratory, General Electric Co.

Present position: Met., Research Laboratory, General Electric Co.

William Edward Ryan, Independence, Colo.

Proposed by Walter H. Aldridge, Wilber Judson, Louis S. Noble.

Born 1883, Denver, Colo. 1905, Colorado School of Mines, E. M. 1910-16, Engr., Field Engr. and Mgr. of unit, Mines Company of America.

Present position—1916 to date: Genl. Supt., Vindicator Cons. Gold Min. Co.

Robert S. Schultz, Jr., La Salle, Ill.

Proposed by Fred T. Agthe, Roy E. Hoffman, C. R. Forbes.

Born 1884, Wilkensburg, Pa. 1893-95, Friends School, Wilmington, Del. 1895-1902, William Penn Charter School, Philadelphia, Pa. 1902-06, School of Mines, Columbia Univ., E. M. 1906-08, Engr., Victoria Copper Min. Co., Victoria, Mich. 1908-11, Chief Engr., Victoria Copper Min. Co., Victoria, Mich. 1911-12, Asst. Supt., Victoria Copper Min. Co., Victoria, Mich. 1912-14, Min. Engr., The Atlas Portland Cement Co., Hannibal, Mo. 1914-16, Supt. of Engrg., The Atlas Portland Cement Co., Hannibal, Mo.

Present position: Cons. Engr., Marguette Cement Mfg. Co.

Sydney S. Sellner, Humboldt, Ariz.

Proposed by F. K. Brunton, E. C. King, G. M. Colvocoresses.

Born 1887, Elmira, N. Y. 1909, Grad., Carnegie Inst. of Tech. 1909-13, Cyanide Mill Supt., La Luz Los Angeles Min. Co., Bluefields, Nicaragua, C. Amer. 1913-14, Assayer and Chemist, Mogollon Mines Co., Mogollon, Mex.

Present position: Cons. Arizona Smelt. Co.

Archibald Dalgliesh Shankland, So. Bethlehem, Pa.

Proposed by W. R. Shimer, F. O. Kichline, E. O'C. Acker.

Born 1892, Baltimore, Md. 1911-14, Virginia Polytechnic Institute, B. Sc.

Present position—1914 to date: Chief Chemist, Saucon Plant, Bethlehem Steel Co.

Harry G. Smith, Principulca, Nicaragua, C. A.

Proposed by H. A. Buehler, H. W. Edmondson, Jesse C. Scobey.

Born 1884, Chetopa, Kans. 1901-02, 1902-03, Drury College, Springfield, Mo. 1903-04, 1904-05, Missouri School of Mines. 1904, Engineer's Asst., Joplin, Mo. 1905, Shippers Rep. Torreon Smelter, Torreon, Coahuila, Mex. 1906, Assayer, Hidalgo Min. Co., Mesa Correo, Chih., Mex. 1906-07, Assayer and Cyanide man, Green Gold and Silver Co., Concheno, Chih., Mex. 1907-08, Clerk, Amer. Smelt. & Ref. Co., Chih., Chih., Mex. 1908, Mill Foreman, Ris de Plata Min. Co., Guazaparas, Chih., Mex. 1909, Assayer and Engr., Mary Min. Co., Chih., Mex. 1909-12, Mine Foreman, Mill Supt., Acting Supt., Concheno Min. Co., Concheno, Chih., Mex. 1912-13, Classifier man and Vanner Foreman, Ray Cons. Copper Min. Co., Hayden, Ariz. 1913, Assayer, Refiner, Mill Foreman, El Tigre Min. Co., Yzabal, Son., Mex. Present position—1914 to date: Supt., La Luz & Los Angeles, Min. Co.

Blanchard M. Snyder, Los Angeles, Cal.

Proposed by A. B. W. Hodges, Ralph Arnold, C. Colcock Jones.

Born 1877, Richland, Ia. 1892-97, Throop Polytechnic Institute, Pasadena, Cal. 1900, Custom Chemist and Assayer, Spokane, Wash. 1901-02, Asst. Chemist, Granby Cons. Min., Smelt. and Power Co., Grand Forks, B. C. 1903-06, Chief Chemist, British Columbia Copper Co., Greenwood, B. C. 1906-07, Smelter Supt., British Columbia Copper Co., Greenwood, B. C. 1907-15, Cons. Engr., Los Angeles, Cal. 1915-16, Mgr., Victor Reduction Co., Victor, Mont. 1916, Mill Supt., Comet Mine, Basin, Mont.

Present position: Cons. Engr., Los Angeles.

James Aguilla Stader, Joplin, Mo.

Proposed by E. Fleming L'Engle, H. A. Buehler, G. B. Corless.

Born 1882, Newtonia, Mo. 1900-01, Academic Student, Central College. 1902, Student, Univ. of Missouri, M. A. 1903-06, Special student, Chemical Engineering. 1907, Student, Officers School and War College, Manila, P. I. 1907, Southern Pacific R. R. 1907-11, Military Officer, P. I. 1912, Atlantic Gulf & Pacific Contracting Co. 1913-14, Engrg. Dept., U. S. Army. 1914, Survey of Oil and Gas Fields, Southern Mexico. 1915-16, Chemical Engr., Eagle-Picher Lead Co.

Present position: Engr., Eagle-Picher Lead Co.

George William Starr, Grass Valley, Cal.

Proposed by P. G. Spilsbury, E. Gybbon Spilsbury, F. W. Bradley.

Born 1862, San Francisco, Cal. 1881, Bates Univ., San Francisco, Cal. 1882, Millman, Assayer and Asst. Mgr., Empire Mines, Grass Valley, Cal. 1887, Genl. Mgr., Empire Mines, Grass Valley, Cal. 1893, Genl. Mgr., Kimberley Roodeport Mine, S. Af. 1894, Genl. Mgr., New Primrose Mines, S. Af. 1895, Cons. Engr., Barnato Bros., S. Af.

Present position—1898 to date: Cons. Engr., Empire Mines.

John Robert Suman, Houston, Tex.

Proposed by E. T. Dumble, W. E. Wrather, Robert B. Moran.

Born 1890, Daleville, Ind. 1908, Grad., Los Angeles High School. 1909-10, Univ. of Southern Cal. 1911-12, Univ. of Cal., College of Mining, B. S. 1909, P. A. Howard Construction Co., on Los Angeles Aqueduct. 1910, Office Engr., City of Pomona, Cal. 1911, Asst. Assayer and Surveyor, Yellow Astor Min. & Mill Co., Randsburg, Cal.

Present position—1912 to date: Asst. Geol. and Engr., Rio Bravo Oil Co.

Ernest M. Symmes, Bacchus, Utah.

Proposed by J. M. Callow, Ernest Gayford, C. W. Stimpson.

Born 1889, Winchester, Mass. 1911, Mass. Inst. of Technology, S. B. in Chemistry. 1911, Electrolytic Alkali and Chlorine, Rumford, Me. 1911, Bureau of Standards, Washington, D. C. 1912, E. I. duPont Powder Co., Gibbstown, N. J. and Bayonne, N. J. 1913, E. I. duPont Co., Kenvil, N. J. 1914, Hercules Powder Co., Kenvil, N. J.

Present position: Asst. Supt., Hercules Powder Co.

Oliver Redford Taggart, Collinsville, Ill.

Proposed by Herman Garlich, D. B. Followill, W. E. Newnam.

Born 1887, Salina, Kans. 1909, Grad., Colorado State School of Mines, Golden, Colo., E. M. 1906, summer, Snow Storm Placer Mine, Fairplay, Colo. 1908, summer, Silver Lake Mine, Silverton, Colo. 1909, summer, Golden Cycle Mine,

Victor, Colo. 1909, Fall, concentrating mill, Waldorf Cons. Min. Co., Waldorf, Colo. 1910-11, in stamp mill, Homestake Min. Co., Lead, So. Dak. 1911-12, mill foreman, Beck Min. Co., Atlantic City, Wyo. 1912, Spring, Prospecting trip through Porcupine district, Ontario, Canada. 1912, summer, concentrating mill, Union Basin Min. Co., Golconda, Ariz. 1912, Fall, stamp mill, Gaston Gold Min. Co., Gaston, Cal. 1912-15, Custom Assay office, Telluride, Colo. Nov., 1915, Milling, Utah Minerals M. & M. Co., Eureka, Utah.

Present position—1915 to date: Chemist, St. Louis Smelt. & Ref. Co.

Thomas James Taplin, Jr., Sara Su, Siberia.

Proposed by Edward T. McCarthy, Herbert C. Woolmer, C. J. Hall.

Born 1886, Walthamstowe, London. To 1904, Public Secondary School. Central Foundation, Cowper St., E. C. Particularly Mathematics, Physics and Chemistry. 1904, Evening Schools and private study. 1910, Students' examination, Institute of Civil Engineers. 1911, Associate Membership, Institute of Civil Engineers. 1905, Office of H. G. Humley, Vice Cons. Engr. to Natal Government. 1905-09, Leeds Forge Co., Leeds. 1910, work in connection with the battleship, H. M. S. "Thunderer," Thames Iron Works Co., Cuning Town, E. C. 1911-14, Asst. and Chief Draughtsman, Messrs. W. G. Perkins & Co., London Wall, E. C. 1914-16, Chief Draughtsman and Engineering Asst., Atbasar Mines, Spassky Copper Mine, Ltd., Siberia.

Present position: Sara Su Concentrator, Spassky Mine.

Hollinshead N. Taylor, Philadelphia, Pa.

Proposed by William W. Hearne, Henry Shriver, Isaac Harter, Jr.

Born 1879, Philadelphia. 1901, Univ. of Pennsylvania, B. S. 1901-17, Senior member of firm, N. & G. Taylor Co.

Present position: Pres., N. & G. Taylor Co.

Alfred Paul Thompson, Great Falls, Mont.

Proposed by E. S. Bardwell, Milo W. Krejci, W. T. Burns.

Born 1893, Florence, Colo. 1911-12, Iowa State College. 1912-15, Montana State College, B. S. in Chemical Engrg. 1915-16, Testing Dept., Washoe Reduction Works, Anaconda, Mont. 1916, Asst. Chemist, Washoe Reduction Works, Anaconda, Mont.

Present position—1916 to date: Asst. Chemist, Electrolytic Zinc Plant, B. & M. Reduction Works, Anaconda Copper Min. Co.

John M. Tippet, Colorado Springs, Colo.

Proposed by H. C. Parmelee, Harry J. Wolf, George J. Young.

Born 1872, St. Louis, Mo. 1890, Mining, Aspen, Colo. 1891-92, Sorting ore, Aspen, Colo. 1893, Colorado School of Mines. 1896, Assayer and Chemist for Mr. Clarence King, on Mine examination, Gilman, Colo. (Iron Mask Mine). 1897, Assay, Bi-Metallic Smelt. Co., Leadville, Colo. 1897-1900, Chemist and Assayer, Colorado Ore Reduction Co., Cripple Creek, Colo. 1901, Chief Chemist and Assayer, Union Gold Extraction Co., Florence, Colo. 1902, Portland Gold Min. Co., Colorado Springs, Colo.

Present position: Asst. Supt., Portland Gold Min. Co.

Herbert Rodney Tunnell, Butte, Mont.

Proposed by Chauncey L. Berrien, John Gillie, C. W. Goodale.

Born 1880, Georgetown, Del. 1896, Grad., Georgetown, Delaware High School. 1901, Grad., Delaware College, B. M. E. 1902, Delaware College, B. C. E. 1902-03, Engrg. Dept., Worth Bros. Steel Mill, Coatesville, Pa. 1903, Instrument man, Norfolk and Western R. R., Williamson, W. Va. 1904-05, Engr., Red Jacket Cons. Coal and Coke Co., Red Jacket, W. Va. 1906, Mill Construction, F. W. Tunnell, Co., Frankford, Pa. 1907, Anaconda Copper Min. Co., Butte, Mont.

Present position: Foreman, Pennsylvania Mine, Anaconda Copper Min. Co.

D. A. Vazquez, Firmeza, Oriente, Cuba.

Proposed by DeB. Whitaker, Max Roesler, A. F. Duggleby.

Born 1888, Santiago de Cuba. 1904, Finished Santiago High School. 1904-05, The Heffley School, Brooklyn, N. Y. 1905-07, American Collegiate Institute, Far Rockaway, N. Y. 1907-10, Civil Engr. course, Rensselaer Polytechnic Institute, Troy, N. Y.

Present position—1910 to date: Engrg. staff, Juragua Iron Co.

Henry Joseph Volker, Cerro de Pasco, Peru, S. Amer.

Proposed by J. T. Glidden, Ewing Carter, Myron R. Walker.

Born 1889, Ossining, N. Y. 1906, Grad., Ossining High School, N. Y. 1907-11, Grad., Columbia School of Mines, E. M. 1911-12, Milling, Nevada Cons., McGill, Nev. 1913, Engrg., Beatson Copper Co., Latouche, Alaska. 1914, Sampling, Timbering, Mining, Canadian Copper Co., No. 3 Mine, Ontario. To date, 5 months engineering, 1 month shift boss, Cerro de Pasco Min. Co.

Present position: Foreman, Excelsior Mine, Cerro de Pasco Min. Co.

Mark U. Watrous, Magdalena, N. M.

Proposed by Elmer Pfouts, George E. Farish, Harry J. Wolf.

Born 1897, Windsor, N. Y. 1914, Colorado School of Mines, E. M. 1912, Terrace Irrigation District. 1913-14, State Highway, Colo. 1915, State Highway and private survey. 1916, Ozark Smelt. & Min. Co.

Present position: Asst. Mine Supt., Ozark Smelt. & Min. Co.

George Clifton Whaley, St. Francois, Mo.

Proposed by Arthur P. Watt, George B. Rodgers, W. H. Comins.

Born 1883, Calloway Co., Mo. 1905, Grad., Univ. of Missouri, B. S. 1905-08, Draftsman, Heyl and Patterson, Pittsburgh, Pa. 1908, Draftsman, Jeffrey Mfg. Co., Columbus, O. 1909-13, Draftsman, St. Joseph Lead Co., Flat River, Mo. and Herculanum, Mo. 1913-14, Draftsman, Anaconda Smelter, Great Falls, Mont. 1915, Tennessee Copper Co., Copperhill, Tenn.

Present position: Draftsman, St. Louis Smelt. & Ref. Co.

Edward I. Williams, New York, N. Y.

Proposed by Rowland F. Hill, Robert K. Painter, W. H. Nichols, Jr.

Born 1892, Charleston, S. C. 1910-14, Columbia Univ., E. M. 1914-16, Asst. Supt., Modoc Min. Co., Goudreau, Ont., Canada.

Present position: Asst. to Chairman, Min. & Met. Dept., General Chemical Co.

Gerrit Van Wagener Wood, High Bridge, N. J.

Proposed by John H. Hall, Knox Taylor, Victor Angerer.

Born 1893, Geneva, N. Y. 1911-15, Union Univ., Schenectady, N. Y., B. E. 1913, summer, production department, mine motors and supplies, General Electric Co., Schenectady, N. Y. 1914, summer, New York State Engrg. Corps on Barge Canal, Stillwater, N. Y.

Present position: Representative, Taylor-Wharton Iron and Steel Co., San Francisco, Cal.

Price T. Wrigley, Raton, N. M.

Proposed by Bert Lloyd, T. H. O'Brien, J. L. Caruthers.

Born 1884, Springer, N. M. 1890-98, Public schools in Raton, N. M., Denver, Colo., Philadelphia, Pa. 1898-1900, Raton Coal & Coke Co. 1900-02, Engrg. Dept., A. T. & S. F. Ry. Co. 1902-05, Mex. Central Ry.—Sierra Madre Ry., Sierra Madre Lade Lumber Co. 1905-06, Barstow Min. & Mill Co., Silverton, Colo. 1906-07, Raton Water Co., Yankee Fuel Co., (with exception of 1st 2 employees as Levelman, Transitman, Draughtsman, Supt. of Const.). 1907-16, County Engr., Colfax County.

Present position: Engr., Yankee Fuel Co.

Associate Members

George A. Fagerberg, Malmberget, Sweden.

Proposed by W. L. Saunders, F. Wanjura, C. R. Miller.

Born 1880, Upsala. 1899, Student, Upsala. 1899-1903, Technical High School, Stockholm. 1904, Falun Copper Mine. 1905, Rapala Zinc Mine. 1906, Sala Silver Mine. 1906-08, United States of America. 1908-09, Study trip, Belgium, Germany, Austria-Hungary, France. 1910, Tuollavaara Iron Ore Mine. 1911, Kiruna Iron Ore Mine.

Present position—1912 to date: Min. Engr., Luessavaara Kiirunavaara Co.

George W. Mark, St. Louis, Mo.

Proposed by H. A. Wheeler, Arthur Thacher, W. E. McCourt.

Born 1868, St. Louis, Mo. Grammar and High School. Private course and instruction under Charles Luedeking, Ph. D., St. Louis, Mo. 1901-10, Assayer in charge, U. S. Assay Office, St. Louis, Mo. 1914-16, Mining, Mohave County, Ariz. 1916, Supt., Missouri-Mohave Min. and Mill Co.

Present position: Pres., U. S. Assay and Ref. Co.

Einar Christian Richter, Narvik, Norway.

Proposed by W. L. Saunders, George A. Howells, C. R. Miller.

Born 1877, Orkedal, Norway. 1889-92, Trondhjems Cathedralsskole. 1892-96, Trondhjems Tekniske Lærestad. 1896-97, Practice with the Christian Municipal Electro Power Station. 1897, The Water Power Plant, Hapland. 1898-1904, Engr. for several enterprises of submarine buildings and construction of railways and power plants. 1904-10, Engr., Luessavaara Kiirunavaara Co., Narvik, Norway.

Present position—1910 to date: Mgr., The Luessavaara Kiirunavaara Co.

Robert W. Weiss, Santiago, Chile, S. Amer.

Proposed by A. J. Eveland, Mark R. Lamb, Louis A. Wright.

Born 1886, Lima, Peru. 1908, Escuela de Ingenieros, Lima, Peru, M. E. 1909, Cerro de Pasco Min. Co., Cerro de Pasco, Peru. 1911, Braden Copper Co., Rancagua, Chile. 1912, Allis-Chalmers Mfg. Co., Santiago office.

Present position: In charge of Santiago office, Allis-Chalmers Mfg. Co., during the manager's absence in the States.

Junior Members

Gale Leslie Adams, Houghton, Mich.

Proposed by F. W. McNair, F. W. Sperr, W. E. Hopper.

Born 1890, Madison, Wis. 1904-06, Madison High School, Madison, Wis. 1908-10, Jackson High School, Jackson, Mich. 1910-13, Fireman, Michigan Central R. R. 1913, Machinist's helper, Santa Fe Shops, San Bernardino, Cal.

Present position: Student, Michigan College of Mines.

Russell Cornelius Boyles, State College, Pa.

Proposed by C. E. McQuigg, W. R. Crane, E. S. Moore.

Born 1894, Steelton, Pa. 1909-13, Steelton High School. 1909-12, summers, Open-hearth, Pennsylvania Steel Co., Steelton, Pa. 1913, summer, Merchant Mill, Pennsylvania Steel Co., Steelton, Pa. 1914, summer, Templet Dept., Pennsylvania Steel Co., Steelton, Pa. 1915-16, summers, Open-hearth Central Iron and Steel Co., Harrisburg, Pa.

Present position: Student, Pennsylvania State College.

John Constine, So. Bethlehem, Pa.

Proposed by H. Eckfeldt, Benjamin L. Miller, Henry S. Drinker.

Born 1895, Wilkes-Barre, Pa. 1912, Grad., Wilkes-Barre High School. 1913-14, Student, Wilkes-Barre Min. Inst. School. 1912-14, Asst. to Chemist, Lehigh Valley Coal Co. 1914, summer, Engrg. Dept., Lackawanna Division, Lehigh Valley Coal Co. 1915, summer, Lackawanna Division, Engrg. Dept., Lehigh Valley Coal Co.

Present position: Student, Lehigh University.

William W. Cowan, So. Porcupine, Ont., Canada.

Proposed by R. M. Raymond, Robert Peele, J. C. Pickering.

Born 1893, New York, N. Y. Public School, New York, N. Y. Townsend Harris High School. 1912-16, Columbia Univ., School of Mines, M. E.

Present position: Asst. Surveyor, Domes Mines.

Tom Ralph Crawford, Rolla, Mo.

Proposed by Horace T. Mann, Charles Y. Clayton, A. L. McRae.

Born ———. 1909-13, Poola High School, Poola, Kans. 1913-16, Kansas School of Mines, Weir, Kans. Work during the summer in the Joplin District, Joplin, Mo.

Present position: Student, Missouri School of Mines.

Henry William Doennecke, Rolla, Mo.

Proposed by Charles Y. Clayton, Austin L. McRae, H. A. Buehler.

Born 1894, Massena, Ia. To 1910, Davenport Grammar School, Ia. 1910-14, High School, Davenport, Ia. 1914-15, Missouri School of Mines. 1916, summer, Asst. Assayer, Hill City Tungsten Production Co., Hill City, So. Dak.

Present position: Student, Missouri School of Mines.

Charles F. Feledy, State College, Pa.

Proposed by W. M. Weigel, C. E. McQuigg, W. R. Crane.

Born 1892, Gönc, Hungary. 1904-06, Gymnasium, Rozsnyo, Hungary. 1909-13, Allegheny High School, Pa. 1913-14, summer, Chemistry Laboratory, Pittsburgh

Crucible Steel Co. 1915-16, summer, Asst. Yard Foreman, Blooming Mill Billet Yard, Pittsburgh Crucible Steel Co. 1916-17, summer, Shear recorder, Production clerk, Blooming Mill Dept., Pittsburgh Crucible Steel Co.

Present position: Student, Pennsylvania State College.

Mowry E. Goetz, Woodlawn, Pa.

Proposed by W. M. Weigel, C. E. McQuigg, W. R. Crane.

Born 1893, Cleveland, O. 1907, Sharon Public Schools. 1907-11, Beaver High School. 1908, Civ. Engrg., Chainman, Jones & Laughlin Steel Co. 1909, Civ. Engrg., Rodman, Jones & Laughlin Steel Co. 1910, summer, Civ. Engrg., Rodman, Jones & Laughlin Steel Co. 1911, Civ. Engrg., Transitman, Jones & Laughlin Steel Co. 1911-12, Open Hearth, Third and Second Helper, Jones & Laughlin Steel Co. 1913-14, Open Hearth, First Helper, Jones & Laughlin Steel Co. 1915 and 1916, summers, Open Hearth, First Helper, Jones & Laughlin Steel Co.

Present position: Student, Pennsylvania State College.

Clarence W. Hannon, San Francisco, Cal.

Proposed by Frederick K. Copeland, W. S. Weeks, R. P. McGrath.

Born 1889, Saginaw, Mich. 1908-12, Univ. of Michigan, B. S. 1912 to present time; the first year and a half of this time was devoted to shop work and other work preparatory to Salesman's duties.

Present position—1912 to date: Salesman, Sullivan Machinery Co.

Ross Glenn Hinman, Corry, Pa.

Proposed by C. E. McQuigg, W. R. Crane, E. S. Moore.

Born 1892, Corry, Pa. 1911, Corry High School. 1914, summer, Canadian Copper Co., Copper Cliff, Ont. 1916, summer, Canadian Copper Co., Copper Cliff, Ont.

Present position: Student, Pennsylvania State College.

Harry W. Hosford, Bethlehem, Pa.

Proposed by Joseph W. Richards, G. A. Roush, H. Eckfeldt.

Born 1887, Danville, Ill. 1905, Grad., Danville Illinois High School. 1910, Grad., U. S. Naval Academy, Annapolis, Md. 1916-17, Special course in Metallurgy, Lehigh Univ., So. Bethlehem, Pa. 1910-17, Officer, U. S. Navy.

Present position: Lieut., U. S. Navy.

George Dodd Morgan, State College, Pa.

Proposed by W. M. Weigel, W. R. Crane, C. E. McQuigg.

Born 1892, Ogden, Utah. 1906-09, Ashland High School, Ashland, Pa. 1909-13, Chainman, Philadelphia & Reading Coal and Iron Co., Ashland, Pa. 1914, summer, Miner, Berwind White Coal Co., Windber, Pa. 1915, summer, Office work, Lehigh Valley Coal Co., Centralia, Pa. 1916, summer, Laborer, The New Jersey Zinc Co., Franklin, N. J.

Present position: Student, The Pennsylvania State College.

Hoyt Sherman, Cambridge, Mass.

Proposed by Charles H. White, George S. Raymer, H. L. Smyth.

Born 1896, Chicago, Ill. 1905-08, Noble and Greenough, Boston, Mass. 1908-09, Institution Sillig, Veney, Switzerland. 1909-10, Pierce School, Brookline, Mass. 1910-14, Brookline High School, Brookline, Mass. 1913, summer, Lady Belle Lease Syndicate, Eagle, Colo. 1914, summer, Bellevue Mine, Empire, Colo. 1916, summer, Excelsior Mine, Frisco, Colo.

Present position: Student, Harvard Univ.

Sidney Smith Small, Golden, Colo.

Proposed by Harry J. Wolf, W. G. Haldane, George J. Young.

Born, 1892, Lincoln, Nebr. 1910, Surveyor, Costilla Estates Dev. Co., San Acacio, Colo. 1911, Surveyor, Denver and Salt Lake R. R. Co., Denver, Colo. 1914, Mill man, Mary Murphy Gold Min. Co., Romley, Colo. 1915, Assayer, Tomboy Gold Min. Co., Ltd., Telluride, Colo. 1916, Asst. Testing Engr., Anaconda Copper Min. Co., Anaconda, Mont.

Present position: Student, Colorado School of Mines.

Richard Walter Thomas, State College, Pa.

Proposed by W. M. Weigel, C. E. McQuigg, W. R. Crane.

Born 1892, Lebanon, Pa. 1898-1907, Public Schools, Steelton, Pa. 1907-11,

High School, Steelton. 1906-13, summers, Moulding apprentice, Steel Foundry Dept., Pennsylvania Steel Co. 1913-16, summers, Moulder, Steel Foundry Dept., Bethlehem Steel Co.

Present position: Student, Pennsylvania State College.

Hsiung Tsai, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, Benjamin L. Miller, H. Eckfeldt.

Born 1896, Chekiang, China. 1911-14, High School Dept., Tsing Hua College, China. 1914-15, Phillips Academy, Andover, Mass. 1916, summer, working with the mining engr. corps of the Lehigh Coal and Navigation Co., Lansford, Pa.

Present position: Student, Lehigh Univ.

Oliver L. Wolfard, So. Bethlehem, Pa.

Proposed by Joseph W. Richards, G. A. Roush, H. Eckfeldt.

Born 1888, Colfax, Wash. 1906, Grad., High School. 1911, Grad., U. S. Naval Academy, Annapolis, Md. 1916-17, Special course in Metallurgy, Lehigh Univ., So. Bethlehem, Pa. 1911-17, Officer, U. S. Navy.

Present position: Lieut., U. S. Navy.

CHANGE OF STATUS

From Associate Member to Member

Henry Ernst Emil Bornhoft, San Antonio, Tex.

Proposed by _____

Born 1877, Hamburg, Germany. 1915-16, Asst. Genl. Mgr., Penoles Min. Co., Mapimi. 1916 to date, Associate Mgr. Compania de Minerale y Metales, S. A., operating mines, smelters, concentrators and zinc calcining plants in Mexico, viz.: Cia. Mineral Paloma y Cabvillas, Higuera; Cia. Minera Providencia and Albarradon, Mazapil; Cerralvo Mines, Smelter and Concentrator; Guadalupe and Mina Viljas Reduction Plant and Mines; Cia. Metalurgica de Terreon, Smelter; Agujita Coal Mines and other properties under lease.

Present position: As specified.

From Junior Member to Member

Arthur Potter Allen, Ashcroft, B. C., Canada.

Proposed by Frederic Keffer, Fred B. Ely, A. G. Heggem.

Born 1892, Flint, Mich. 1911, Grad., Flint High School. 1915, Michigan College of Mines, E. M. 1911-12, Cost Accountant, Buick Motor Co., Flint, Mich. 1915-16, Chief Engr., Winona Copper Co., Winona, Mich. 1916, Efficiency Engr., Calumet and Hecla Min. Co., Calumet, Mich.

Present position: Mine Foreman, Highland Valley Min. & Dev. Co.

Frederick A. Johnson, Changsha, China.

Proposed by _____

Born 1891, New Castle, Pa. 1915, Univ. of Pittsburgh, School of Mines, Petroleum Engr. 1915, Standard Oil Co. of New York, Changsha, Hunan Prov., China.

Present position: On marketing staff, Standard Oil Co. of New York.

John Orth Liebig, Newark, N. J.

Proposed by _____

Born 1890, Sparrows Point, Md. 1914, Grad., Lehigh Univ., in Metallurgy. 1914-15, Blast Furnace Dept., Maryland Steel Co., Sparrows Point, Md. 1915-16, Asst. to Foreman, Silver Shop, Balbach Smelt. & Ref. Co., Newark, N. J. 1916, Shift Foreman, Ore and Bullion Dept., Balbach Smelt. & Ref. Co., Newark, N. J.

Present position—1916 to date: With Metallurgist, Hyatt Roller Bearing Co.

Bertram A. Reber, Corbin, Mont.

Proposed by Theodore Simons, C. H. Bowman, Bancroft Gore.

Born 1892, Lindsay, Nebr. 1907, St. Patricks Parochial School, Baltimore. 1911, High School, Butte. 1915, Montana State School of Mines, E. M. 1911-14, Underground, West Colusa, Butte. 1914-16, Underground Speculator, Butte.

Present position—1916 to date: Engr. and Assayer, Boston and Corbin C. & S. Min. Co.

CHANGE OF ADDRESS OF MEMBERS

The following changes of address of members have been received at the Secretary's office during the period Jan. 1, 1917 to Feb. 10, 1917.

This list, together with the foregoing list of new members, therefore, supplements the annual list of members published in the Year Book and brings it up to the date of Feb. 10, 1917.

- ADAMS, ARTHUR KINNEY, Andes Copper Min. Co., Casilla 230,
Antofagasta, Chile, South America.
- AGNEW, WILLIAM C. 518 Hawthorne Road, Duluth, Minn.
- ALDERSON, VICTOR CLIFTON, Cons. Min. Engr., 185 Devonshire St., Boston, Mass.
- ALLEN, ARTHUR WATTS 309 Broadway, New York, N. Y.
- ALSDORF, FREDERICK C, Min. Engr 434 W. 120th St., New York, N. Y.
- ANDERSON, WILLIAM G., Mgr., Ore Chimney Min. Co., Ltd.,
Northbrook, Ont., Canada.
- AUSTIN, JOHN F. American Smelt. & Ref. Co., Maurer, N. J.
- BAKER, HAMILTON W. Sunnyside Mines Co., Brunzell, Idaho.
- BARNARD, ENOCH A. Instructed to hold everything.
- BASSETT, THOMAS ERSKINE Box 277, Bisbee, Ariz.
- BATCHELLER, JAMES H., Mgr., Virginia Lead & Zinc Corp.,
Allah Cooper Operation, Mineral, Va.
- BATCHELOR, STILLMAN Mina La Mejn, Los Acostas, Pinar del Rio, Cuba.
- BECK, EDWIN LAWRENCE Sodaville, Nev.
- BEDWORTH, HERBERT ARTHUR Hammond Laboratory, New Haven, Conn.
- BILICK, DON CARLOS Latouche, Latouche Island, Alaska.
- BISSELL, ROBERT WILSON P. O. Box 172, Greensburgh, Westmorland Co., Pa.
- BLACKMER, WILLIAM DANIELS, Northern Ore Co.,
Edwards, St. Lawrence Co., N. Y.
- BLANDY, S. H. B. Ivythorne Manor, Street, Somerset, England.
- BORDWELL, EDWARD MEAD Mill Supt, Sarita Min. Co., Masonic, Cal.
- BOWMAN, FRANCIS C. 15 Washington St., Deadwood, S. Dak.
- BRADEN, WILLIAM, Casilla No. 83-D, Calle Teatinos 351,
Santiago, Chile, South America.
- BRADLEY, HENRY FRANKLIN New Park Hotel, Park City, Utah.
- BRADY, S. H. Room 302, Nixon Bldg., Reno, Nev.
- BRIGGS, CHARLES H. 315 W. 4th St., Sedalia, Mo.
- BRINSMADE, R. B. Las Minas de Zomelahuacan, Veracruz, Mexico.
- BROWN, DOUGLAS PHILIP, 61 Castletown Road,
West Kensington, London, W., England.
- BROWN, HOLCOMBE JAMES Box 101, Ridley Park, Pa.
- BROWN, JAMES FREDERIC, 2nd Box 1896, Goldfield, Nev.
- BRUNTON, FREDERIC KEMBLE, Consolidated Arizona Smelt. Co., Humboldt, Ariz.
- BRYANT, ALBERT D. 276 Patent Office, Washington, D. C.
- BULL, ROBERT ALEXANDER, Vice-Pres. and Genl. Mgr., Chicago Steel Foundry Co.,
Kedzie Ave. and 37th St., Chicago, Ill.
- BURG, ROBERT S. Box 598, Miami, Ariz.
- BURNETT, ERIC V., The Ore Trading Co., Ltd., Guayacan, via Coquimbo,
Chile, South America.
- BUSH, H. E. 721 Market St., Redding, Cal.
- BYRNE, HARRY BARRY, Care Hornblower & Weeks, 42 Broadway, New York, N. Y.
- CAMERON, WILLIAM MCC Savoy Hotel, Strand, London, W., England.
- CARLSON, ARTHUR EUGENE Shannon Copper Co., Florence, Ariz.
- CARLYLE, WILLIAM A. 507 Royal Bank Bldg., Toronto, Ont., Canada.
- CARNAHAN, GEORGE HOLMES, Pres., Intercontinental Rubber Co.,
120 Broadway, New York, N. Y.
- CARPENTER, HOWARD B., Supt. of plant, Colorado Fuel and Iron Co., Pueblo, Colo.
- CARPENTER, JAY A., Genl. Supt., Nevada-Packard Mines Co., Lower Rochester, Nev.
- CARSON, ELLARD W., Oceanic Quicksilver Mine, Cambria,
San Luis Obispo Co., Cal.
- CARTER, EDWARD E. Mgr., Gold Hill Mine, Quartzburg, Idaho.
- CHADWICK, JOHN P. Casilla 12, Valparaiso, Chile, South America.
- CHAMPION, J. R. Box 25, Arvada, Colo.
- CHAPMAN, LEWIS C. Empire Gas and Fuel Co., Box 578, Bartlesville, Okla.

- CHAPPELL, HOWARD F. 120 Broadway, New York, N. Y.
 CHASE, CHARLES A. 812 Cooper Bldg., Denver, Colo.
 CHEDSEY, WILLIAM R., Associate Prof. of Mining, Pennsylvania State College,
 State College, Pa.
 CLAGHORN, DAVID MONTGOMERY, Coal Dept., Puget Sound Traction,
 Light & Power Co., Renton, Wash.
 COE, HARRISON S. 307 W. 98th St., New York, N. Y.
 COLE, A. NEWTON, H. Koppers Co., Care Brier Hill Steel Co., Youngstown, O.
 COLE, WALTER B. The Daly Mining Co., P. O. Box 780, Park City, Utah.
 COLLINS, GLENVILLE ARTHUR Arctic Club, Seattle, Wash.
 CORBET, EDWARD B. 1201 First National Bank Bldg., San Francisco, Cal.
 CORTESE, EMILIO P. O. Box 829, Alexandria, Egypt.
 CRAIG, JOHN J. P. O. Box 942, Jerome, Ariz.
 CREER, GEORGE Missouri Metals Corp., Mine La Motte, Madison Co., Mo.
 CROSBY, FRED W., Min. Engr. 434 H. W. Hellman Bldg., Los Angeles, Cal.
 CROSBY, WILLIAM O. Mass. Inst. of Technology, Cambridge, Mass.
 CULLUM, J. BARLOW Pottsville, Pa.
 DAGGETT, ELLSWORTH 409 Commonwealth Ave., Los Angeles, Cal.
 DAVENPORT, JOHN Asst. Supt. Virginia Smelt. Co., West Norfolk, Va.
 DAVIES, ROBERT GEISER Box 184, San Francisco, Cal.
 DAVIS, SIDNEY HUGH Baxter Springs, Kans.
 DODSON, TRUMAN M. P. O. Box A, Bethlehem, Pa.
 DONNELLY, THOMAS F. 569 Ridgewood Road, Maplewood, N. J.
 DRESSER, CLARENCE G. Missouri Metals Corp., Mine La Motte, Mo.
 DUPUIS, OLIVER CAMILLE, Mandarin Mines Corp., Yellow Jacket, Lemhi Co., Idaho.
 DUTTON, CLARENCE BENJAMIN 331 Fourth Ave., Pittsburgh, Pa.
 EATON, ARTHUR First National Bank Bldg., San Francisco, Cal.
 EDELEN, A. W. Hotel St. Francis, Oatman, Ariz.
 ELLIOTT, STUART R. Cleveland Cliffs Iron Co., Ishpeming, Mich.
 ELLIS, HUBERT INGERSOLL Kellogg, Idaho.
 ELLSWORTH, LINCOLN 12 E. 69th St., New York, N. Y.
 EMMONS, N. H. 2d Whittle Springs, Tenn.
 ENOS, HERBERT CORY 906 Mills Bldg., El Paso, Tex.
 EVANS, GEORGE WATKIN 1011 North 47th St., Seattle, Wash.
 EVELAND, ARTHUR JOHN, Care Engineers' Club, 32 W. 40th St., New York, N. Y.
 FAULKNER, ROBERT Bethlehem Steel Co., Lebanon, Pa.
 FINKELDEY, WILLIAM H. 220 North 32d St., Camden, N. J.
 FISHER, HOWELL TRACY 8 Spring Lane, Englewood, N. J.
 FISHER, NORMAN RICHARD, Consulting Engr., Haileybury, Ont., Canada.
 FISHER, W. B. Box 1506, Boston, Mass.
 FLAGG, ARTHUR LEONARD Ray, Ariz.
 FLOSS, F. C. Perseverance, via Thane, Alaska.
 FORBES, JAMES HYDE Lindsay-Strathmore Irrigation District, Lindsay, Cal.
 FRANTZ, JOSEPH HENRY P. O. Box 1381, Columbus, Ohio.
 FULLAWAY, RICHARD M. Supt., Dewey Mines Co., Gazelle, Cal.
 FULTON, CHARLES H. 1903 Boatman's Bank Bldg., St. Louis, Mo.
 FURST, EDWARD W., The Grasselli Chemical Co., Guardian Bldg., Cleveland, O.
 GAENSLER, GEORGE R. Jerome, Ariz.
 GARRISON, MURRAY E. 415 West Galena St., Butte, Mont.
 GATES, ARTHUR O., Sales Engr., Dodge Sales and Engineering Co.,
 814 Newhouse Bldg., Salt Lake City, Utah.
 GATES, LAUREN A. Engr., Four States Coal Co., Dorothy, W. Va.
 GEORGE, PERCY WILLIAMS Federal Mining & Smelting Co., Wallace, Idaho.
 GRADY, WILLIAM HENRY Lock Box 15, Lost Creek, W. Va.
 GREER, HERBERT CHESTER P. O. Box 849, Morgantown, W. Va.
 GREYER, WALTER SCOTT Mgr., The Lead and Zinc Co., Metaline Falls, Wash.
 GRIEVE, RICHARD P. 168 Sterling St., Brooklyn, N. Y.
 GRISWOLD, CLYDE T. Sag Harbor, N. Y.
 GRÜBNAU, VICTOR C. Los Cerrillos, Santa Fe County, New Mexico.
 GUTERMAN, KENNETH S., American Smelt. & Ref. Co.,
 1st National Bank Bldg., Denver, Colo.
 HACKETT, WILLIAM H. Mgr., The Texas Copper Co., Benjamin, Tex.
 HAMILTON, THOMAS M. Andes Exploration Co., Casilla 83-D,
 Calle Teatinos 351, Santiago, Chile, So. Amer.
 HARROD, WAYNE A. Box 398, Golden, Colo.

- HAWKINS, TERRELL G., JR., Asst. Genl. Mgr., Alvarado Min. & Mill. Co.,
706 Mills Bldg., El Paso, Tex.
- HICKMAN, ELWOOD CHEYNEY.....Box 66, Murray, Utah.
- HIGGINS, THOMAS F.....34 Water St., Laurel Hill, Long Island, New York.
- HILL, FRANK ALBERT.....1305 Smith Bldg., Seattle, Wash.
- HILL, ROWLAND F.....General Chemical Co., 25 Broad St., New York, N. Y.
- HILL, WILLIAM J.....Engrg. Dept., Ray Cons. Copper Co., Ray, Ariz.
- HOFFMAN, JOHN F., Supt., Western Nevada Mine,
Nevada-Douglas Cons. Copper Co., Mason, Nev.
- HOFFMANN, JOHN S.....Tyrone, New Mexico.
- HOFFSTOT, F. N.....350 Broadway, New York, N. Y.
- HOHL, LEONHARD JOHN.....251 Kearny St., San Francisco, Cal.
- HOWE, ALBION S.....Hotel Lorenz, Redding, Cal.
- HOWRY, HAMILTON HUBBARD.....616 W. 113th St., New York, N. Y.
- HUDSON, ALBERT WILLIAM.....P. O. Box 707, Warren, Ariz.
- HULL, DANIEL R.....The American Brass Co., Waterbury, Conn.
- HUTCHISON, ROBERT M.....354 Locust St., Steelton, Pa.
- HYDER, CHARLES A.....904 Hobart Bldg., San Francisco, Cal.
- INGERSOLL, JULIUS C.....Smuggler Union Mining Co., Telluride, Colo.
- ISHIHARA KIUKICH.....Kenjiho Iron Works, Mitsubishi Co., Okaido,
Chosen, Korea.
- ISHIWATA, KINNOSUKE, Care Sakito-Kogyosho, Sakito-mura, Nishisnoki-gun,
Nagasaki-ken, Japan.
- JARVIS, ROYAL P.....Cananea Cons. Copper Co., Cananea, Son., Mexico.
- JOBES, CHARLES T.....Stotts City, Mo.
- JOHNS, ALFRED L.....Box 607, Jerome, Ariz.
- JOHNSON, GUY R., JR.....Room 715, Frisco Bldg., Joplin, Mo.
- JOHNSON, WOOLSEY MCA.....69 Vernon St., Hartford, Conn.
- JONES, CHARLES HENRY, Met., 8 Bellevue Mansions, Forest Hill, S. E.,
London, England.
- JONES, THOMAS J.....7 Gracechurch St., London, England.
- JORDAN, FRED A.....1517 E. 4th St., Duluth, Minn.
- JORDAN, KNIGHT STARR.....Knight Investment Co., Provo, Utah.
- KAEDING, CHARLES D.....43 Exchange Place, New York, N. Y.
- KELLY, WILLIAM.....Vulcan, Mich.
- KENNEDY, GEORGE ADAMS The Western Chemical Manufacturing Co., Denver, Colo.
- KERR, GUY MANNING.....The Tacoma Smelt. Co., Tacoma, Wash.
- KLINE, ALLEN HAROLD.....234 Pacific St., Franklin, Pa.
- KOCH, ARTHUR WILLIAM.....American Zinc Lead & Smelt. Co., Granby, Mo.
- KOLENSKY, ALEXANDER VLADIMIROVICH, Min. Engr., Mgr., The Pitoeff
Naphtha Production and Distributing Co., Moika 30, Petrograd, Russia.
- KOMAR-OF-SKYE, GEORGE.....999 Aldus St., New York, N. Y.
- KRAUSE, WALTER B.....Instructed to hold everything.
- LADD, DAVID H.....Milford, Mich.
- LAMB, MARK R., Care American Institute of Mining Engineers,
29 W. 39th St., New York, N. Y.
- LANDON, STEPHEN L.....P. O. Box 256, Prescott, Ariz.
- LANG, HENRY.....36 Hawthorn Place, Montclair, N. J.
- LANGENBERG, FREDERICK C.....Met., Watertown Arsenal, Watertown, Mass.
- LASELLE, BEACH A.....512 Pacific Bldg., Vancouver, B. C., Canada.
- LEACH, ROBERT H.....Care Handy & Harman, Bridgeport, Conn.
- LEANING, EUGENE HOOKER.....Racket Brook Coal Co., Carbondale, Pa.
- LE BOUTILLIER, CLEMENT.....Katonah, N. Y.
- LEE, MONTROSE LUCIUS, Care S. Pearson & Son, Ltd., 82 Broadway, New York, N. Y.
- LI, KUO CHING.....Wah Chang Min. & Smelt. Co., Ltd.,
2283 Woolworth Bldg., New York, N. Y.
- LINDEMUTH, L. B.....Bethlehem Steel Co., Bethlehem, Pa.
- LIVERMORE, ROBERT, The Goodrich Lockhart Co., 60 Broadway, New York, N. Y.
- LIVINGSTON, IVOR.....First Creek Coal Co., Typo, Ky.
- LOVE, JAMES W.....Leonard Hotel, Butte, Mont.
- LUCKE, P. K.....136 E. Magnolia Ave., San Antonio, Tex.
- LUNDY, WILSON THOMAS.....1812 E. 24th St., Oakland, Cal.
- LUPTON, CHARLES T.....Drawer P, Tulsa, Okla.

- LYMAN, ROBERT H., Pres. and Managing Director,
 Elliott-Kirkland Gold Mines, Ltd., Cobalt, Ont., Canada.
 LYON, GEORGE J.....5 Ardsley Road, Schenectady, N. Y.
 MCAULIFFE, EUGENE.....Vice-Pres., West Kentucky Coal Co., Paducah, Ky.
 MCCOY, JAMES COMLEY....Colorado Mining Co., 95 Liberty St., New York, N. Y.
 MCCRYSTAL, JOHN HENRY...Supt., Gemini Mining Co., P. O. Box F., Eureka, Utah.
 MCCREERY, JAMES HAROLD, Secy. and Treas., Ophuls, Hill & McCreery, Inc.,
 112 W. 42nd St., New York, N. Y.
 MACFARLANE, RIENZI W.....P. O. Box 674, Morenci, Ariz.
 MACGOWAN, JOHN K.....120 Broadway, New York, N. Y.
 MANNING, JOHN FRANCIS.....Care American Legation, Peking, China.
 MARCH, ALBERT T.....48 Highland St., Hammond, Ind.
 MARSHALL, GEORGE B.....Abangarez Gold Fields, Abangarez, Costa Rica, C. A.
 MARSHALL, NEWTON C., Care Francisco Menotti, Buenaventura, Colombia, So. Amer.
 MARSHALL, STEWART M.....14 E. 8th St., New York, N. Y.
 MARSHALL, STUART BRADFORD, Aluminum Co. of America, Badin,
 Stanly Co., No. Carolina.
 MATTESON, WALLACE G.....Geol., Producers Oil Co., Houston, Tex.
 MEMMINGER, C. G.....Pres., Coronet Phosphate Co., Lakeland, Florida.
 MILLARD, WILLIAM J.....201 Texas Bldg., Tulsa, Okla.
 MILLS, EDWIN WALTER.....Care American Legation, Peking, China.
 MILLWARD, WILLIAM.....Hotel Columbia, Columbia, Pa.
 MIXNER, A. F.....The Silver Peak Mines, Ltd., Yerranderie, N. S. W., Aust.
 MOLENGRAEFF, G. A. F.....Voorstraat 60, Delft, Holland.
 MOORE, GEORGE.....28 Dock St., Yonkers, N. Y.
 MORAN, ROBERT B.....52 Union League Bldg., Los Angeles, Cal.
 MORRISON, HAROLD A.....Oak Hill Mine, La Grange, Tuolumne Co., Cal.
 NAGEL, HENRY P., JR., Min. Engr.....Victor, Colo.
 NEEL, CARR BAKER.....Box 253, Menlo Park, Cal.
 NEELD, HARPER C.....Zinc Co., Ltd., Montauban, Quebec, Canada.
 NEWBERRY, ANDREW WHITE....Care W. R. Grace & Co., Lima, Peru, S. Amer.
 NEWELL, ARTHUR.....P. O. Box 37, Anaconda, Mont.
 NEWELL, GEORGE STRIBLING.....American Smelt. & Refining Co., Saso, Ariz.
 NEWMAN, M. H.....903 South Main St., Carthage, Mo.
 NICOLSON, CLYDE W., The Engineering Dept., Cleveland-Cliffs Iron Co.,
 Ishpeming, Mich.
 NORRIS, ROBERT VAN ARSDALE, JR.....90 Pinehurst Ave., New York, N. Y.
 NORSWORTHY, HOWARD R.....130 Claremont Ave., New York, N. Y.
 NORTH, MORTIMER S.....Victoria Copper Min. Co., Ontonagon Co., Mich.
 OFFICER, HERBERT G., Andes Exploration Co., Teatinos 351, Santiago,
 Chile, S. Amer.
 OLIVER, EARL.....William Muir Oil Co., 202 McClure Bldg., Tulsa, Okla.
 OTTO, HENRY H.....Lehigh Valley Coal Co., 76 Sullivan St., Wilkes-Barre, Pa.
 PACKARD, HENRY J.....363 City Hall, San Francisco, Cal.
 PALMER, IRVING ALLSTON.....Altoona, Kans.
 PARRISH, SAMUEL F.....4619½ Melbourne Ave., East Hollywood, Los Angeles, Cal.
 PEARCE, WILLIAM C. WALWORTH, Wondecla, Herberton, North Queensland, Aust.
 PERRY, JOHN G., Genl. Supt., The Leyden Coal Co., 503 Gas & Electric Bldg.,
 Denver, Colo.
 PETERS, RICHARD, JR.....Rogers Brown & Co., Morris Bldg., Philadelphia, Pa.
 PIERCE, TALBOT E.....Semet-Solvay Co., Lock Box 399, Cleveland, O.
 PIGOTT, CURTIS.....Asst. Supt., U. S. Smelt. Co., Midvale, Utah.
 PLATT, JAMES M.....Cayllona, Peru, So. Amer.
 POIRIER, C. HERBERT., Min. Engr.....63 Wall St., New York, N. Y.
 PORTER, JESSE C.....11 First St., New Brighton, N. Y.
 PYNE, WALTER F.....524 N. Negley Ave., Pittsburgh, Pa.
 RAGAZ, IVAN.....1008 Mills Bldg., P. O. Box 777, El Paso, Tex.
 RAIBER, NICOLAUS H.....127 Madison St., Wilkes-Barre, Pa.
 REEDER, EDWIN C.....900 Fisher Bldg., Chicago, Ill.
 RENO, CHARLES A.....Mine Surveyor, Magma Copper Co., Superior, Ariz.
 RHOADS, ALBERT E.....U. S. Bureau of Mines, Golden, Colo.
 RICHARDS, ADELBERT HARRY.....Garfield Smelter Co., Garfield, Utah.
 RICHOLSON, WALTER A., Engineering staff, Cananea Cons. Copper Co.,
 Cananea, Sonora, Mexico.
 RICKARD, BRENT N.....American Smelt. & Ref. Co., Murray, Utah.
 RIEBEL, OTTO F.....Newhouse, Utah.

- RIGBY, WILLIAM ARTHUR, Feldspars, Ltd., Hartington R. R. No. 1, Ontario, Canada.
 RINEHART, CHARLES RAMSEY, Fuller Engineering Co., R. 2079,
 50 Church St., New York, N. Y.
 RING, AMBROSE ELY.....Fredericktown, Mo.
 ROBBINS, HALLET RICE.....813 Birke Bldg., Vancouver, B. C., Canada.
 RODGERS, C. EARL, Chief Assayer and Refiner, The Dome Mines Co.,
 So. Porcupine, Ont., Canada.
 RODGERS, JOSEPH H.....Supt., Comet Mine, Basin, Mont.
 ROSE, HUGH.....Mills Bldg., San Francisco, Cal.
 ROSE, CLYDE P.....21 Crownshield Road, Brookline, Mass.
 RUETSCHI, RUDOLF.....35 Ruth St., Hammond, Ind.
 SALISBURY, GEORGE W.....Jerome-Victor Extension Copper Co., Jerome, Ariz.
 SAMPSON, CARLOS E.....Box 608, Maricopa, Cal.
 SANDERS, W. MURRAY.....Carlisle Min. Co., Steeple Rock, Grant Co., New Mexico.
 SAYRE, HAL.....815 Logan St., Denver, Colo.
 SCHINDLER, DONALD FRANKLIN.....San Ignacio 25, Havana, Cuba.
 SCHULTZ, C. I.....Moctezuma Copper Co., Douglas, Ariz.
 SCHULTZ, ROY WILSON.....Calumet Hotel, Calumet, Mich.
 SCHUYLER, ARENT H.....International Nickel Co., Bayonne, N. J.
 SEARS, MORTIMER ANDREWS, Field Service, General Land Office, Washington, D. C.
 SIMKINS, WILLIAM A.....647 Mills Bldg., San Francisco, Cal.
 SIMPSON, K. M., Min. Engr.....57 Post St., San Francisco, Cal.
 SLATER, JOHN L., JR.....1002 Lake Shore Ave., Oakland, Cal.
 SMITH, CECIL W.....353 Fisk St., Pittsburgh, Pa.
 SMITH, ELIAS A. CAPELEN, Cons. Met. Engr., Guggenheim Bros.,
 120 Broadway, New York, N. Y.
 SMITH, HENRY P.....1101 Fremont Ave., So. Pasadena, Cal.
 SMITH, JAMES H., JR.....Care Melones Min. Co., Melones, Calaveras Co., Cal.
 SMITH, JAMES W.....16 Hamilton Road, Brookline, Mass.
 SMITH, W. HINCKLE.....201 Liberty Bldg., Philadelphia, Pa.
 SMYTH, JOHN G.....R. F. D. No. 1, Fairmont, W. Va.
 SNOW, FREDERICK WILLIS.....51 Autumn St., East Lynn, Mass.
 STACK, FRANK LAWRENCE, Care Compania Minera, Fomento,
 Prov. of Santa Clara, Cuba.
 STAYER, WILLIAM H., Min. Engr.....75 St. James Terrace, Yonkers, N. Y.
 STEWART, HOWARD R.....Room 1945, Railway Exchange Bldg., St. Louis, Mo.
 STIFEL, CARL GODFRIED, Brewer, Vice-Pres., Otto & Stifel Union Brewing Co.,
 St. Louis, Mo.
 STOCKWELL, RUPERT K.....910 Walker Bank Bldg., Salt Lake City, Utah.
 STODDART, A. W.....1251 St. Andrews Place, Los Angeles, Cal.
 STRANEY, JOHN WILLIAM.....Midland, Pa.
 STRINGHAM, J. S.....1860 Penobscot Bldg., Detroit, Mich.
 STROHECKER, JOHN WALLACE.....Masonia, Idaho.
 STROUT, ERNEST A.....405 South Van Ness Ave., Los Angeles, Cal.
 SWANQUIST, G. A.....Supt., Comacaran Gold Min. Co., Salvador, C. A.
 SWEENEY, HARRY PATTERSON.....Fort Montgomery, N. Y.
 TACKMANN, HENRY.....Baxter Springs, Kans.
 TAYLOR, WILLIAM WALTER.....Braden Copper Co., Rancagua, Chile, S. Amer.
 TEDDY, FREDERICK T., Adventure Cons. Copper Co., Box 350,
 Greenland, Ontonagon Co., Mich.
 THAME, JAMES.....Thorney Mill House, Iwer, Bucks, England.
 THOENEN, JOHN R., Supt. of Garson Mine, The Mond Nickel Co., Ltd.,
 Garson, Ont., Canada.
 THOMPSON, WARREN D.....Chile Exploration Co., Chuquicamata, Chile, S. Amer.
 THOMSON, HERBERT G., Min. Engr., Nevada Packard Mine Co.,
 Lower Rochester, Nev.
 THURSTON, E. C.....Ross, Marin Co., Cal.
 TOP, GRANT H.....Coast Mfg. & Supply Co., Livermore, Cal.
 TOENGES, ALBERT L.....Box 644, Fredricktown, Mo.
 TRUEX, ARTHUR FULLER.....506 Daniels Bldg., Tulsa, Okla.
 TRUSCOTT, S. J., Royal School of Mines, So. Kensington, London, S. W., England.
 TUSCHKA, OTTO.....Box 1718, Globe, Ariz.
 URQUHART, CHARLES HOWARD.....275 Belleville Ave., Newark, N. J.
 VAN BARNEVELD, CHARLES E., Min. Engr. and Met., U. S. Bureau of Mines,
 Tucson, Ariz.

VARIAN, JEAN PHILIP.....	420 So. Washington St., Iola, Kans.
VAUCLAIN, ANDREW CONSTANT.....	2416 N. 54th St., Philadelphia, Pa.
VENABLES, HARRY L.....	P. O. Box 215, Oruro, Bolivia, S. Amer.
VOORHEES, FREDERICK A.....	Grass Valley, Cal.
WAKABAYASHI, YAICHIRO, Mitsu Bishi & Co., Metal Min. Dept., Marunouchi, Tokyo, Japan.	
WALLACE, LOUIS R., Genl. Mgr., Andes Copper Mining Co., Casilla 230, Antofagasta, Chile, So. Amer.	
WALTER, RAYMOND A.....	83 West Main St., Frostburg, Maryland.
WARFORD, NORMAN L.....	1903 McCormick Bldg., Chicago, Ill.
WARNER, CHARLES M., Hotham St. and Melby Ave., East St. Kilda, Melbourne, Aust.	
WEIGEL, WILLIAM M., International Molybdenum Co., Ltd., Renfrew, Ont., Canada.	
WEINBERG, GEORGE S., Cons. Min. Engr., Mgr., Worthington Pump Co., Ltd., Nevsky 6, Petrograd, Russia.	
WEISENBURG, ANDREW.....	Hope and Palmer Streets, Philadelphia, Pa.
WELD, CHRISTOPHER M., Mining Engineer.....	2 Rector St., New York, N. Y.
WEMPLE, LELAND E., American Zinc, Lead and Smelt. Co., 1012 Pierce Bldg., St. Louis, Mo.	
WHEELWRIGHT, O. W.....	819 Tower Ave., Superior, Wis.
WHERRY, HENRY P.....	University Club, Chicago, Ill.
WHITAKER, CHARLES NEWTON, JR.....	1209 Garfield St., Denver, Colo.
WHITE, ROBESON T.....	American Smelt. & Ref. Co., Caldera, Chile, S. Amer.
WHITNEY, JOHN DRAYTON.....	Instructed to hold everything.
WILLIAMS, GARDNER F.....	Pacific-Union Club, San Francisco, Cal.
WILLIAMS, JOHN CHARLES.....	Apt. 6, 1760 Franklin St., Denver, Colo.
WILLIAMS, WILLIAM A.....	Bartlesville, Okla.
WILMOT, H. CLIFFORD.....	Instructed to hold everything.
WILSON, R. T., JR.....	120 Broadway, New York, N. Y.
WINSLOW, FRANCIS.....	1711-18th St., Washington, D. C.
WOLFE, CONRAD.....	300 Title Block, Spokane, Wash.
WOOD, ALEXANDER HAMILTON, Genl. Mgr. and Secy., King-Harlan Min. Co., Kildav, Harlan Co., Ky.	
WOODWARD, FRANK A.....	Iron Cap Copper Co., P. O. Box 309, Globe, Ariz.
WORMSER, FELIX EDGAR.....	Surveyor, Cornucopia Mines Co., Cornucopia, Ore.
WRAY, DAVID COWDEN.....	Mineral Point Zinc Co., Depue, Ill.
WRIGHT, H. F.....	William Muir Oil Co., 202 McClure Bldg., Tulsa, Okla.
WYER, SAMUEL S.....	Cons. Engr., Huntington Bank Bldg., Columbus, O.
WYBOR, D. C.....	Care General Chemical Co., 25 Broad St., New York, N. Y.
YAKE, E. E.....	Works Engr., Gilbert & Barker Mfg. Co., Springfield, Mass.

MEMBERS' ADDRESSES WANTED

Name.	Last address of Record from which Mail has been returned.
ABBOTT, JAMES W.....	2253 Calumet St., Los Angeles, Cal.
AHIER, PHILIPPE D., Anglo Greek Magnesite Co., Ltd., Afrati, Chalois, Euboea, Greece.	
BALFOUR, A. M.....	P. O. Box 301, Republic, Wash.
BYERS, WHEATON B.....	50 Hampden Hall, Cambridge, Mass.
CHURCH, JOHN L.....	Box 1123, Los Angeles, Cal.
DOUTHETT, GEORGE M.....	Crescent Metals Co., Lock Box 8, Glenwillard, Pa.
DEFARIA, CICERO C.....	Rua Conde do Bomfim, No. 46, Rio de Janeiro, Brazil.
HOBART, EDMUND NORRIS.....	Box 621, Nogales, Ariz.
JAMES, CYRIL H.....	Kingman, Ariz.
LINHARDT, HARVEY W.....	Columbus Iron and Steel Co., Columbus, O.
LIVINGSTON, ADDISON S.....	Box 138, Morenci, Ariz.
MACDONALD, AUGUSTUS.....	Box 207, R. F. D., San Gabriel, Cal.
NANCE, W. FRED.....	Belknap Coal Co., New Castle, Pa.
PARKER, JOSEPH EDMUNDSON, Rhodesia Exchange and Dev. Co., Box 213, Bulawayo, Rhodesia, S. Africa.	
READ, NORMAN HATFIELD.....	R. F. C., Care War Office, London, England.
SCHNEIDER, ALBERT F.....	Care Engineers' Club, 32 W. 40th St., New York.
SIBBALD, ALEXANDER.....	Box 1091, Globe, Ariz.
WINNE, ROLLA D.....	17 College Ave., Adrian, Mich.
ZACH, LOUIS M.....	U. S. Smelt., Ref. & Min. Co., Salt Lake City, Utah.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Jan. 1, 1917 to Feb. 10, 1917.

Date of Election.	Name.	Date of Death.
1915	*Dougall, Ralph.....	
1888	*Gifford, Stanley D.....	Dec. 31, 1916.
1901	*Hay, Alexander M.....	Jan. 20, 1917.
1890	*Jaques, William Henry.....	Dec. 23, 1916.
1912	*Jones, Lewis M.....	Oct. 20, 1916.
1886	*McKinney, R. C.....	Oct. 3, 1916.
1886	*Raht, August.....	Dec. 25, 1916.
1875	*Scranton, W. W.....	Dec. 3, 1916.
1912	*Sheadle, Jasper Holmes.....	
1884	*Thomas, Fred F.....	Aug. 6, 1916.
1882	*Vivian, George G.....	Oct. 11, 1916.
1912	*Wolf, Otto C.....	Dec. 19, 1916.

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.

DAVID H. BROWNE, *Chairman*. PERCY E. BARBOUR, *Vice-Chairman*.
 A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
 C. A. BOHN, *Treasurer*.

JOHN V. N. DORR,

LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.

W. E. C. EUSTIS, *Chairman*. R. L. AGASSIZ, *Vice-Chairman*.
 E. E. BUGBEE, *Secretary-Treasurer*, Mass. Inst. of Technology, Boston, Mass.
 ALBERT SAUVEUR, H. L. SMYTH.

Columbia

Holds four sessions during year. Annual meeting in September or October.

W. H. LINNEY, *Chairman*. OSCAR LACHMUND, *Vice-Chairman*.
 LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2164, Spokane, Wash.
 STANLY A. EASTON, S. SHEDD.

Puget Sound

Meets second Saturday of each month

SIMON H. ASH, *Chairman*. I. F. LAUCKS, *Vice-Chairman*.
 CHARLES SIMENSTED, *Secretary-Treasurer*, 425 Lyon Bldg., Seattle, Wash.
 GLENNVILLE A. COLLINS, JOHN N. POTT.

Southern California

C. COLCOCK JONES, *Chairman*. PHILIP WISEMAN, *Vice-Chairman*.
 ALVIN B. CARPENTER, *Secretary-Treasurer*, 508 Union League Building, Los Angeles, Cal.
 A. B. W. HODGES, R. A. PEREZ.
 E. A. MONTGOMERY, WILLIAM F. STAUNTON.

Colorado

CHARLES LOUGHRIDGE, *Chairman*. GEORGE M. TAYLOR, *Vice-Chairman*.
 FRED CARROLL, *Secretary-Treasurer*, State Capitol, Denver, Colo.
 F. H. BOSTWICK, P. M. McHUGH,

Montana

W. C. SIDERFIN, *Chairman*. OSCAR ROHN, *Vice-Chairman*.
 E. B. YOUNG, *Secretary-Treasurer*, 526 Hennesey Building, Butte, Mont.
 F. W. BACORN, C. D. DEMOND.

San Francisco

Meets second Tuesday of each month.

W. H. SHOCKLEY, *Chairman*. FRANK H. PROBERT, *Vice-Chairman*.
 C. E. GRUNSKY, JR., *Secretary-Treasurer*, 57 Post St., San Francisco, Cal.
 C. E. BRAYTON, J. F. NEWSOM.

*Pennsylvania Anthracite*R. V. NORRIS, *Chairman*.

CHARLES F. HUBER, *Vice-Chairman*. EDWIN LUDLOW, *Vice-Chairman*.
 W. J. RICHARDS, *Vice-Chairman*. ARTHUR H. STORRS, *Vice-Chairman*.
 PAUL STERLING, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
 DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP,
 RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

C. J. ADAMI, *Chairman*. HERMAN GARLICH, *Vice-Chairman*.
 F. W. DeWOLF, *Vice-Chairman*. M. M. VALERIUS, *Vice-Chairman*.
 WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
 A. W. DICKINSON, CHARLES T. ORR, ARTHUR THACHER.
 C. R. FORBES, F. D. RASH,

Chicago

CHARLES H. MacDOWELL, *Chairman*. LUTHER V. RICE, *Vice-Chairman*.
 HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
 ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
 GEORGE F. HULST, FREDERICK T. SNYDER.

Utah

C. W. WHITLEY, *Chairman*. WILLIAM WRAITH, *Vice-Chairman*.
 ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
 CECIL FITCH, E. R. ZALINSKI.

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GERALD SHERMAN, *Chairman*.
 NORMAN CARMICHAEL, 1st *Vice-Chair*. B. BRITTON GOTTSBERGER, 2nd *Vice-Chair*.
 ARTHUR NOTMAN, *Secretary-Treasurer*, Bisbee, Ariz.
 W. L. CLARK, J. C. GREENWAY,
 W. G. McBRIDE, FOREST RUTHERFORD.

Nevada

J. W. HUTCHINSON, *Chairman*. FRANCIS CHURCH LINCOLN, *Vice-Chairman*.
 HENRY M. RIVES, *Secretary-Treasurer*, Reno, Nevada.
 W. H. BLACKBURN, E. A. JULIAN,
 EMMET D. BOYLE, JOHN G. KIRCHEN,
 FREDERICK BRADSHAW, C. B. LAKENAN,
 TASKER L. ODDIE.

STANDING COMMITTEES

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GEORGE D. BARRON,
SIDNEY J. JENNINGS,

EDWIN LUDLOW,
L. D. RICKETTS,

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, September, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Methods for Determining the Capacities of Slime-Thickening Tanks

BY R. T. MISHLER,* ESQUEDA, SONORA, MEX.

(St. Louis Meeting, September, 1917)

I WISH to express my keen appreciation of the article on the above subject by Coe and Clevenger.¹ It has been doubly interesting to me, for the reason that the experience recorded and the principles evolved practically parallel the results of similar investigations made by myself and others at The Lucky Tiger Mine, El Tigre, Sonora, Mex. An outline of the work at El Tigre may aid in confirming the principles expressed by Coe and Clevenger.

El Tigre has the unenviable distinction of possessing a slime with a much lower settling rate than any recorded in the article discussed. Under most favorable conditions the settling rate is about one-third that reported for the Liberty Bell mill, and one-tenth that reported for the Nipissing, Golden Cycle, and Presidio mills. This slow settling characteristic of Tigre ore has necessitated careful investigation of settling phenomena throughout the various evolutions of the plant.

PREVIOUS SETTLING INVESTIGATION AT TIGRE

During construction work at the mill, the Metallurgical Engineer in charge, D. L. H. Forbes, ran a series of experiments on settling. His tests proved that deep settling tanks were not ordinarily necessary. The use of tanks 2 or 3 ft. in depth was suggested.²

Later experiments demonstrated that the following principles of settling applied to Tigre pulp.³

1. In settling dilute pulp, settling rates are independent of pulp depth.

2. In settling thick pulp, settling rates are approximately proportional to pulp depth.⁴

¹ H. S. Coe and G. H. Clevenger: *Bulletin* (March, 1916), 111, 597.

² D. L. H. Forbes: Settling of Mill Slimes, *Engineering and Mining Journal*, (Feb. 24, 1912), 93, 411.

³ R. T. Mishler: Settling Slimes at the Tigre Mill, *Engineering and Mining Journal*, (Oct. 5, 1912), 94, 643.

⁴ Coe and Clevenger express this principle in the words: "After pulp reaches the consistency where the flocs touch each other, further elimination of water becomes approximately a function of time."

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3. The critical dilution, or the boundary between dilute and thick pulp, in the application of the above principles, is the highest dilution at which channels form in the pulp; clear liquid reaching the surface through the channels.

It is gratifying to note that these principles, tentatively announced 4 years ago as applying to Tigre ore, are now proven, by the comprehensive tests of Coe and Clevenger, to have universal application.

The formula, suggested at that time for solving problems of area, was likewise similar to that adopted by these gentlemen.

$$\text{Area} = \frac{0.0222 (\text{Dry Tons}) (R_1 - R_2)}{\text{Settling Rate, in Feet per Minute}}.$$

R_1 and R_2 were the moisture ratios respectively of feed and discharge. The settling rate was that of the feed, as determined by laboratory tests. This method of calculation was shown to give results which closely approximated those of actual settling operations in the plant.

In subsequent investigations at El Tigre, improved methods have been devised for the laboratory determination of settling rates, and for the calculations of the required area and depth of settling tanks. Also many interesting new principles have been developed concerning the effect on settling phenomena of various changes in temperature and percentage of lime in solution. The details of these investigations are discussed at length below.

DEFINITIONS

In order to avoid confusion, the nomenclature employed by Coe and Clevenger is used as far as possible in this paper.

Settling rate is expressed in terms of the depth in feet of clear liquid formed per hour; or the rate in feet per hour at which the pulp surface moves downward.

Dilution is expressed in terms of the ratio by weight of liquid to solids, and is termed, for short, "The L:S Ratio." In tables, charts and formulas, the first factor only of the L:S ratio is used. Thus the figure 5 designates a dilution of 5:1.

Natural slime is defined as minutely divided particles of pulp resulting from the disintegration of kaolin, talc, or other similar constituents of ore which occur minutely subdivided in nature.

Slime is empirically defined, for the purpose of this discussion, as the finer particles of pulp that will remain suspended for 5 min. in a 100:1 mixture of fresh water and ground ore; the temperature of separation being 60° F. (16° C.). The term covers all the natural slime occurring in the pulp and also a considerable portion of extremely fine sand.

Sand⁵ is the relatively coarse material of the pulp, resulting from grinding quartz, calcite, feldspar, or other massive gangue matter. It may vary from the size of a pea to impalpable powder.

Flocs or floccules are aggregates of slime particles, caused by attraction or cohesion between the particles. Flocculation⁶ may be increased by the addition of electrolytes, such as lime hydrate, magnesium sulphate, glue, etc., and may be decreased by the introduction of deflocculating agents (sodium carbonate, sodium hydrate, etc.). Temperature and dilution have marked effects on the rate at which flocs form.

The free-settling zone is the horizon of evenly flocculated dilute pulp in the upper part of a continuous-settling tank. In this zone, the flocs do not press one upon the other, and settling rates are independent of pulp depth.

The critical dilution is the dilution at which flocs begin to touch. It is the highest dilution at which channels or tubes form in the pulp, the channels or tubes furnishing passageways to the surface for the clear liquid set free in the bottom of the vessels.

The thickening zone is the zone of thick pulp in the bottom of a continuous-settling tank. In this zone, the flocs rest one upon the other, and liquid is eliminated chiefly through the channels mentioned above. Settling rates, in the thickening zone, are approximately proportional to pulp depth. The same principle is expressed by Coe and Clevenger in the words: "Elimination of water is a function of time" (in the thickening zone). The term, "zone of compressions" is used by Coe and Clevenger, on the assumption that the introduction of the time element, as a function of settling, is due to compression caused by the depth of pulp. I believe that this assumption is open to dispute and have hence adopted a more general term for designating the zone.

⁵ It has been demonstrated by Free (*Engineering and Mining Journal*, (Nov. 6, 1916), 101, 250, that as regards the relation of mineral particles to liquid, there is no dissimilarity in the physical characteristics of slime and sand; any variation in behavior being explained on the basis of the difference in size of the particles, and the consequent effect of surface tension, surface adsorption and other similar forces. Free also shows that it is doubtful if sand is ever ground sufficiently fine, in ordinary mill practice, to cause it to assume the colloidal properties of natural slime. On this account it is necessary for the hydrometallurgist to distinguish between sand and natural slime. Regardless of its degree of fineness, sand has little effect on settling phenomena. It is the natural slime in the ore which determines settling rate, critical dilution and other phases of settling. It is likewise the natural slime in ore which determines the rate of filtration and the dilution best suited for classification, flotation, or gravity concentration.

⁶ This phase of settling is well covered by Free, and also by Ralston (*Engineering and Mining Journal*, (May 20, 1916), 101, 894.

THEORETICAL CONSIDERATIONS

Investigations at El Tigre indicate that cohesion or attraction between the flocs is responsible for the fact that elimination of water in the thickening zone is a function of time. Only a positive tensile force, like cohesion, could form channels and keep them open afterward. In the absence of channels, elimination of water in the thickening zone is a function of area and not of time. This fact is illustrated in Table 3.⁷

Cohesion plays an important part during all settling operations. In the free-settling zone it causes the slime particles to aggregate into flocs. In the first stages of thickening, it causes the flocs to draw together, thus developing tensile strains in the pulp which are relieved by cracks and channels (the clear liquid reaching surface through these avenues). During the final stages of thickening, especially in the presence of high lime, cohesion no longer causes closer union between slime particles, or between flocs; but it still strongly resists any force tending to change the shape of the flocs.⁸ This resistance reduces settling rate by opposing the compression, due to pulp depth, which would otherwise force the flocs to displace the liquid in the interfloccular spaces. The greater density attained by the deeper pulp columns in Table 3 can best be explained on this basis.

It has been noted by Coe and Clevenger that after settling has ceased in the shorter columns, it may be caused to continue by decanting the clear liquid and agitating the thick pulp. This behavior is quite as apparent in the absence of classified sand as when classified sand occurs in the pulp. The cause of the action appears to be the rearrangement of the slime particles. Before agitation, a slightly flocculated structure is noted, a close inspection disclosing clear liquid between the flocs. After agitation, no flocculation is apparent. Evidently the pulp flocculates "en masse" after agitation, and settling is resumed because cohesion aids instead of resists compression.

The rate at which flocs form in the free-settling zone is an item of importance in the operation of continuous-settling tanks. When pulp first leaves the feed column of a tank, flocs are entirely unformed and the settling rates are low or nil. The dilute pulp spreads out laterally—over the top of the denser pulp in the bottom of the tank. As the lateral

⁷ The failure of the channels in the latter part of the tests of Table 3 was due to high lime, combined with the complete absence of classified sand. This combination of conditions seldom occurs in plant practice. Hence the data of Table 3 have little practical value, except to aid in understanding the principles of settling. Classified sand, when it occurs in the pulp, furnishes passageways for the liquid after the collapse of the channels. Under such circumstances, settling rates remain proportional to depth throughout the entire thickening period.

⁸ It is in the stage of thickening that the channels collapse (see Table 3).

movement lessens, flocs begin to form and the settling rate increases correspondingly. The rate at which flocs develop depends upon temperature, dilution, and strength of electrolytes. During warm weather, and in the presence of high lime, the slime becomes fully flocculated almost immediately after leaving the feed column. Under such circumstances the average settling rate, over the entire area of the tank, is high. During cold weather, and in the presence of low lime, partially flocculated pulp may spread practically to the tank periphery, forming a blanket over the pulp surface, which retards egress of liquid from the more completely flocculated pulp in the lower horizons of the tank.

The various stages in the development of flocs are well represented in laboratory tests at free-settling dilutions. When tests are first started, flocculation is imperfect and settling rates are low, this stage representing conditions near the feed column in continuous-settling tanks. As the tests continue, flocs become more perfectly developed and settling rates increase. When lime, dilution, and temperature are high, flocculation is completed and maximum settling rates are attained, before the pulp surface has settled $\frac{1}{8}$ in. When lime, dilution and temperature are low, complete flocculation and maximum settling rate are attained only after the pulp surface has settled $\frac{1}{4}$ or $\frac{1}{2}$ in. The complete flocculation and consequent maximum settling rates, finally attained in the tests, correspond to the conditions existing near the tank periphery and also in the lower free-settling horizons of the tank.

Laboratory tests, for determining the settling rates corresponding to the various horizons in a continuous-settling tank, should be so performed that the degree of flocculation in the tests is approximately the same as occurs at the various depths in the settling tanks. Tests at the dilution of feed should aim to represent the imperfectly flocculated condition occurring near the feed column as well as the more complete flocculation at the tank periphery, the average settling rate over the entire pulp surface being the result desired. Experience at Tigre, under varying conditions of dilution, lime and temperature, indicates that this result can best be accomplished when the first 0.05 ft. settled in the tests furnishes the basis for the calculation of the settling rates corresponding to feed dilutions.

In case doubt exists that area might be governed by the permeability of some horizon below that of the feed, a series of settling tests at lower free-settling dilutions are run. In this series, observations are not started until after flocs are fully developed and maximum settling rates are attained. This condition is usually fulfilled when the second 0.05 ft. settled in the tests furnishes the basis for calculating the settling rates corresponding to the lower dilutions in the depths of the settling tank.⁹

⁹ In order to maintain uniform dilution at the pulp surface the depth of pulp in the tests should be 1 ft. or more.

At Tigre, area is invariably governed by the settling rate of the feed horizon. Hence, the tests at lower dilutions are usually omitted. In Table 2, a series of settling rates, experimentally determined at feed dilutions, are compared with actual settling rates in a 32-ft. continuous-settling tank, a considerable range in lime, dilution and temperature being obtained in the comparison. The experimentally determined settling rates will be observed to check closely with those occurring in actual practice. Minor differences are probably due to changes in the character of ore from day to day, and to the difficulty of properly correlating dilutions of feed and discharge. The tank was operated at maximum capacity throughout the tests, so errors introduced by changes in pulp level were slight.

Table 3 also illustrates the effect of deficient area on thickening operations below critical dilution. In all cases recorded in the table, the thickness of discharge was limited by settling rates in the free-settling zone, the area being inadequate to secure any benefit from the depth of tank. Even when the feed entered the tank below critical dilution, it was diluted to critical dilution by the excess of liquid rising from the lower depths, the bulk of pulp in the tank being practically at critical dilution. Final thickness in such cases was limited by the settling rate at critical dilution.¹⁰

PRACTICAL PRINCIPLES

The foregoing discussion of settling phenomena differs somewhat from that of Coe and Clevenger. The differences are mostly theoretical in character and are not insurmountable in a practical consideration of the subject. For the application of experimental data to the solution of practical problems, fundamental principles are all-important. In the conception of fundamental principles, Coe, Clevenger and myself are fully agreed.

The principles underlying settling operations may be briefly stated as follows:

1. Settling rates in the free-settling zone are independent of pulp depth. Stated in other words: At dilutions above critical dilution, elimination of water is a function of settling area, and is independent of pulp depth.

2. Settling rates in the thickening zone vary directly with pulp depth. Stated in other words: At dilutions below critical dilution, elimination of liquid is a function of pulp volume, being dependent on both area and depth. Or, in the words of Coe and Clevenger, "elimination of water in the zone of compression is a function of time."

¹⁰ Settling rates, corresponding to dilutions of feed below critical dilution, were regarded as proportional to differences in dilution, the calculations being based on the experimentally determined settling rate at critical dilution.

3. Settling rates vary with the dilution of the natural slime in the pulp, being usually greatest for high dilutions and smallest for low dilutions.

4. Settling rates are practically independent of the amount of sand occurring in the pulp (regardless of the fineness of the sand).

5. Settling rates increase as the temperature increases.¹¹

6. Settling rates vary with the degree of flocculation of the slime particles.

7. Degree of flocculation depends upon:

(a) The nature and preceding treatment of the ore.

(b) The character and concentration of electrolyte.

(c) The dilution and temperature of the pulp.

(d) The time elapsed after inception of flocculation.

8. Critical dilution¹² depends primarily upon the character and proportion of natural slime in the ore; secondarily upon the strength of flocculating agents in solution; and thirdly (only slightly) upon temperature.

The above principles furnish the basis for the solution of settling problems. The first two principles are of direct importance in the determinations, by laboratory tests, of the settling equipment required in metallurgical plants. The remaining principles enumerate the effects of other influences upon settling operations and illustrate the importance of maintaining these influences the same in the tests as in actual practice.

TESTS FOR DETERMINING SETTLING EQUIPMENT

In designing or remodeling hydro-metallurgical plants, questions frequently arise regarding the number and size of settling tanks to be installed. Occasionally the tanks are required for some definite service in a predetermined flow sheet of the plant. In such cases, the conditions of temperature, electrolyte, tonnage, dilution of feed, and desired thickness of discharge, are known beforehand, the problem being to provide sufficient settling equipment to fulfil these conditions. Usually, the flow sheet depends to some extent upon settling operations, rendering it necessary to determine approximately the settling equipment required under varying conditions, in order to decide intelligently upon the conditions best suited for a well-balanced plant.

The first step, in the solution of such problems, is to secure a repre-

¹¹ This phase of settling is covered by Ashley, *Mining and Scientific Press*, (June 12, 1909), 98, 831.

¹² The critical dilution, in ordinary "all-slime" plants, is usually about 2 : 1 when lime is low and 3 : 1 when lime is high. Tigre ore contains an unusually large proportion of natural slime. The critical dilution is therefore high, ranging from 4 : 1 with low lime to 6 : 1 with high lime. The critical dilution of Tigre slime (with all possible sand removed by classification) varies between 7 : 1 and 12 : 1 according to the lime content.

sentative sample of the pulp to be settled. The sample should contain the maximum proportion of natural slime that would occur in the mill pulp during any single day of operation. It should secure as nearly as possible the same preliminary treatment as would occur in mill practice. Above all, the sample should not be dried at a high heat, since heat changes the character of the slime, greatly increasing subsequent settling rates. The importance has been pointed out, by Coe and Clevenger, of not allowing the sample to remain a greater time in contact with water than would occur in plant operations; the degree of flocculation, and consequently the settling rate, being changed by this treatment. The electrolytes present in the water should be the same as would occur in plant practice. Often, electrolytes are derived partly from the ore¹³ and can be introduced in proper proportions only by bringing the water into repeated contact with fresh samples of ore, the procedure being the same as in the return of water to the head of the mill. The entire object of these precautions is to secure a sample that will represent the pulp to be fed to the tanks.

This sample is settled, preferably over night. The clear water is decanted and set aside for diluting the pulp in the various tests. The thickened pulp is mixed, and tested for liquid to solid ratio.

The following procedure in testing has given most uniformly reliable results at Tigre:

For determining critical dilution, glass graduates are filled to a depth of 1 ft. with varying consistencies of the pulp to be tested. After 10 or 15 min. of undisturbed settling, channels develop in the columns of thicker pulp, clear liquid rising through the channels. If no channels are observed, or if they develop first in the bottom of the vessel, the critical dilution is lower than that of the test. If channels develop first near the pulp surface, or if transverse cracks form in the pulp, the critical dilution is higher than that of the test. The exact critical dilution is indicated by the simultaneous formation of channels in all parts of the settling column, the channels in the upper part of the column being usually replaced later by the extensions of the channels from the bottom of the vessel.

For determining settling rates in the free-settling zone, cylinders are filled to a depth of 1 ft. with various dilutions of pulp, one sample being at exactly critical dilution, the remaining samples representing an ascending scale of dilutions above the critical point. The $L:S$ ratio of each sample is determined by test or by calculation. Each sample is violently agitated and the settling tests started.¹⁴ From the first 0.05 ft. settled

¹³ Calcium sulphate, derived from the action of lime hydrate on iron sulphate, is an active flocculating agent.

¹⁴ In determining minimum settling equipment, it is essential for temperature and electrolyte in the tests to represent minimum conditions in the plant.

in the tests, are calculated the settling rates corresponding to the feed horizons of continuous-settling tanks. These may be termed "the settling rates of feed." From the second 0.05 ft. settled in the tests (of lower dilutions) are calculated the settling rates corresponding to the various free-settling dilutions in the depths of continuous-settling tanks. These may be termed the "free-settling-rates."

Settling rates in the thickening zone are determined in a single test. For this purpose a cylindrical graduate, of 1 liter, or greater, capacity, is marked at the following depths:

Marks on Testing Cylinder¹⁵

Depths in Feet	Depths in Feet
1.160	0.636
1.050	0.576
0.950	0.521
0.860	0.471
0.778	0.427
0.704	0.386

Each depth is $\frac{95}{105}$ of the previous one. The distance between any two consecutive depths is one-tenth the average of the two depths. Hence the distance settled between any two consecutive marks represents $\frac{1}{10}$ ft., per foot depth of pulp. This principle is of great assistance in calculating settling rates in the thickening zone.

The specially marked cylinder is filled to the 1.05-ft. mark with pulp at exactly critical dilution. Solution is added until the combined pulp and solution reaches the 1.16-ft. mark. The mixture is then agitated and the test started. Due to the addition of solution, the test begins in the free-settling zone, and flocs are at least partially formed before the critical dilution is reached. This somewhat parallels usual plant practice and gives more representative results than are obtained when the test is started at exactly critical dilution. The time is first noted as the pulp surface passes the 1.05-ft. mark, and is thereafter noted for each subsequent mark until settling ceases. This test indicates the lowest dilution that can be obtained in the tank discharge, and furnishes the basis for calculating the depth of tank required to produce that dilution, or any other dilution in the thickening zone. The settling rates in the thickening zone must represent the average rate of subsidence of pulp surface, per foot depth of pulp, in settling from the critical dilution to the various possible dilutions of discharge.

The average settling rate, corresponding to any dilution of discharge, may be quickly calculated by considering each space as $\frac{1}{10}$ ft., and by dividing the aggregate space settled (so considered) by the time required for the pulp surface to settle from critical dilution to the dilution con-

¹⁵ In case there is a contraction in the bottom of the cylinder, spaces are proportioned to volume rather than depth.

sidered. The method for tabulating observations and calculating settling rates is illustrated in Table 1.

The dilutions of pulp, corresponding to the various marks on the thickening cylinder, may be successively calculated from the critical dilution by the following formula:

$$D = 0.905 C - \frac{0.095}{G} \quad (\text{Formula 2})$$

D and C are the $L:S$ ratios corresponding to any two consecutive depths.

G is the specific gravity of dry pulp.

This method for calculating dilutions may be extended to the free-settling tests by similarly marking all cylinders, the range of free-settling dilutions being secured by filling the various vessels to the different depths with pulp at critical dilution and adding solution to each vessel until the 1.05-ft. mark is reached.

Determination of Area and Depth of Continuous-Settling Tanks

The determination of the settling area, required for any definite settling operation, is based on the principle that the elimination of water in the free-settling zone is a function of settling area and is independent of pulp depth. Expressed as a formula, this law is:

$$A = \frac{1.34 (F - D)}{S} \quad (\text{Formula 3})$$

Where

A = Area in square feet per ton of dry pulp settled daily.

F = $L:S$ ratio of the governing dilution.

D = $L:S$ ratio of the dilution of discharge.

S = Settling rate of governing dilution (as determined by laboratory test at free-settling dilutions).

The numerical constant (1.34) is the factor introduced by reducing the quantity of water eliminated, from cubic feet per hour to tons per day.

The derivation of this formula is explained more fully by Coe and Clevenger, and also in a former article on the settling of Tigre slime.¹⁶ The idea of a governing dilution (introduced by Coe and Clevenger) generalizes the formula, rendering it applicable, should area be governed by some free-settling rate in the depths of a settling tank.

As an example of the application of the above formula, the area of tank required to settle Tigre pulp from a dilution of 14:1 to a dilution of 2.3:1, may be calculated from the data presented in Table 1. The settling rate of the feed, determined by interpolation in the table, is 0.78 ft. per hour. Substituting in Formula 3:

¹⁶ *Engineering and Mining Journal*, (Oct. 5, 1912), 44, 1643.

$$A = \frac{1.34 (14 - 2.3)}{0.78} = 20 \text{ sq. ft. per dry ton settled daily} \\ \text{(based on the settling rate of the feed).}$$

When the same formula is applied to the various free-settling dilutions between 14 : 1 and 5.1 : 1 (critical dilution), the largest area required is 16 sq. ft.—corresponding to the dilution of 13.1 : 1. This is considerably less than the area determined by the feed dilution. Hence in this case, the settling rate of the feed is the governing factor, the required area being 20 sq. ft. per ton of dry pulp settled daily.

The determination of the depth required in continuous-settling tanks depends on the law that the elimination of water in the thickening zone is a function of both area and depth. This law is expressed in the following formula:

$$\text{Depth} = \frac{1.34 (C - D)}{A \times T} \quad (\text{Formula 4})^{17}$$

Where

C and D are respectively the $L : S$ ratios of critical dilution and of discharge.

A is the settling area (as previously determined by Formula 3).

T is the average settling rate, per foot depth of pulp, in settling from dilution C to dilution D (determined in laboratory tests, previously described).

In the problem to be solved:

$C = 5.1$; $D = 2.3$; $A = 2.0$; and $T = 0.03$ (from Table 1).

Substituting in Formula 4:

$$\text{Depth} = \frac{1.34 (5.1 - 2.3)}{20 \times 0.03} = 6.3 \text{ ft.}$$

From the above calculations, it appears that in order to settle Tigre pulp from a feed dilution of 14 : 1 to a dilution of discharge of 2.3 : 1 (the lime being 0.3 lb. per ton and the temperature 75° F.) it is necessary to provide a minimum area of 20 sq. ft. per ton of dry pulp settled daily, and a minimum effective depth of 6.3 ft. This checks closely with actual requirements in the plant.

The method outlined above is applicable for any other dilutions of feed and discharge within the limits of the tests.

Problems, involving the determination of dilutions of feed or discharge when area and depth are known, may be solved by transposing or combining Formulas 3 and 4.

¹⁷ The formula may be mathematically derived by applying Formula 3 to the horizon of critical dilution.

This method for calculating depth, and the method suggested by Coe and Clevenger, were developed quite independently. Each is based on a different viewpoint of the same law. Hence the results from both methods are identical.

GRAPHICAL METHOD FOR SOLVING SETTLING PROBLEMS

In designing settling equipment it is often necessary to consider numerous combinations of interdependent settling operations. The algebraic solution of such problems is cumbersome, involving a great bulk of calculations for each combination considered. A graphical method, designed to meet the needs at Tigre, might be suggested for use in such cases.

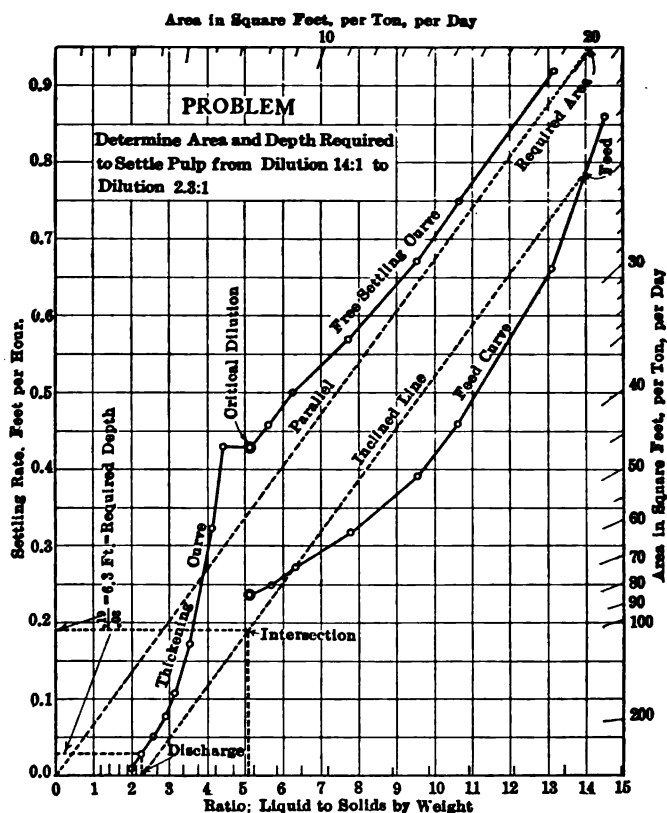


FIG. 1.

For the graphical method, settling tests are performed as previously described, the specially marked cylinders being used in all tests. The settling rates, corresponding to the various dilutions, are plotted on a printed chart (see Fig. 1). The vertical axis of the chart represents settling rates. The horizontal axis is calibrated for both $L:S$ ratios and divisions marked on the cylinders (the short marks above the axis representing the marks on the cylinders). With the $L:S$ ratio of the critical dilution known, lower or higher dilutions may be determined by

counting divisions to left or right, thus obviating the necessity for calculating the ratios of liquid to solids, corresponding to the various dilutions. The radial calibrations at the upper and right-hand margins of the chart are for the purpose of graphically solving problems of area. The problem solved algebraically above is solved graphically in Fig. 1.

Following is the standard procedure for determining area and depth, when dilutions of feed and discharge are known:

1. From the point on the horizontal axis representing dilution of discharge, draw an inclined line to the point on the curve representing settling rate of feed. In case the inclined line intersects the free-settling curve, it is rotated downward until tangent to the lowest depression on the curve. In this case, the point of tangency represents the dilution which governs area.¹⁸ If the inclined line does not intersect the free-settling curve, area is governed by the settling rate of feed.

2. Parallel to the inclined line, draw a line from the zero point of the axis to the margin of the chart and read the area required for each ton of dry pulp, settled daily.

3. Observe the settling rate indicated by the intersection of the inclined line with the ordinate through the critical dilution. Divide this settling rate by the settling rate shown on the curve, corresponding to the dilution of discharge. The quotient is depth required.

Problems involving the dilution of feed or discharge, when depth and area of tank are known, may be solved by reversing this procedure.

The fundamental principles underlying the graphical calculations are identical with those upon which the algebraic determinations are based. Results are the same when calculated by either method. The advantages of the graphical method are greater simplicity, greater rapidity, and the possibility of solving a larger number of problems from a single set of observations. The graphical method is of most service in solving complex problems, in which various settling operations are interdependent.

SCOPE OF LABORATORY TESTS

The laboratory methods for ascertaining the required dimensions of continuous-settling tanks are by no means perfect. Small errors are undoubtedly introduced by the empirical manner of experimentally determining the effects of the rate of flocculation on settling. Other errors possibly enter on account of the failure to determine the extent to which the pulp in the upper part of the tank is diluted by water rising from below. These errors are not serious. Quite possibly they may be altogether eliminated by future investigations. In the meantime, the methods already developed may be employed with the assurance of obtaining reasonably accurate results.

¹⁸ The graphical method is based on the principle that the cotangent of the angle, between the horizontal axis and the inclined line, is equal to the factor $\frac{F - D}{S}$ of Formula 3.

The particular advantage of laboratory determinations lies in the rapidity with which they may be performed. A complete series of tests, covering all necessary dilutions of feed and discharge, usually requires 3 hr. of constant attention, followed by 6 hr. of intermittent attention, and perhaps a couple of observations the next day. In 3 or 4 days it is often possible to cover, in a general way, all conditions of dilution, electrolyte, and temperature, that might occur in the plant.

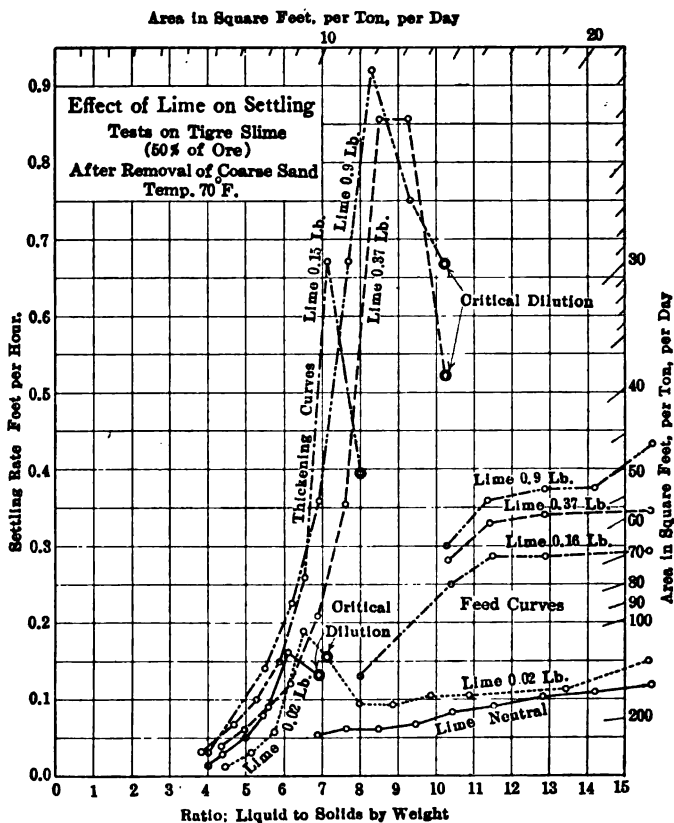


FIG. 2.

Laboratory tests are most useful for preliminary investigations prior to mill construction. They furnish means for roughly estimating the settling equipment necessary for various requirements and under varied conditions in plant practice. These data, combined with similar information regarding dependent operations, enable the mill designer to decide intelligently upon the conditions to be maintained in the plant, and the duty to be performed by the settling equipment. It is as an aid in deciding upon the flow sheet of a contemplated plant, that laboratory settling tests are particularly useful. The approximate determination of the

settling equipment necessary to fulfil the requirements imposed by the flow sheet, is a secondary advantage.

UNIT TESTS

Before proceeding with extensive installations, it is usually advisable to check, by unit tests, the laboratory determinations of the required settling equipment. Unit tests should preferably be performed in tanks of full height, though the area may be considerably less than that of the

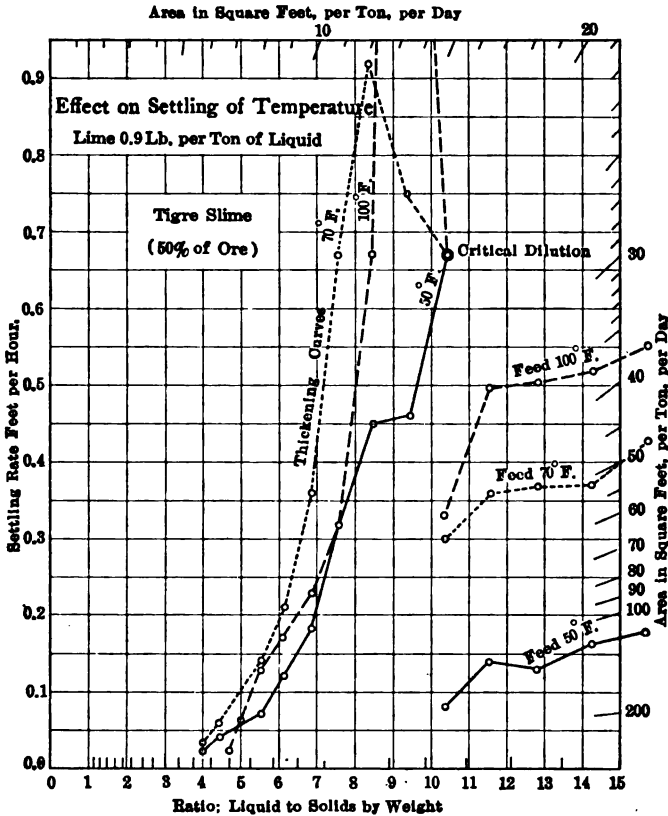


FIG. 3.

tanks to be installed. Test tanks, for determining the size of small installations, may be constructed of 12- or 16-in. pipe, suitable overflow, feed column, cone bottom, and continuous discharge, being provided. For determining extensive installations, the tests should preferably be conducted in standard settling tanks. Observations in unit tests may be started as soon as the required dilution is secured in the discharge, and may be continued over periods ranging from a day to a month, the dura-

tion of the tests depending upon the accuracy desired and the importance of the equipment to be installed. It is important that the required conditions of dilution, electrolyte, temperature, and pulp depth, be maintained throughout the tests; and that the capacity, under these conditions, be accurately ascertained. In determining settling equipment from unit tests, capacity is regarded as proportional to settling area.

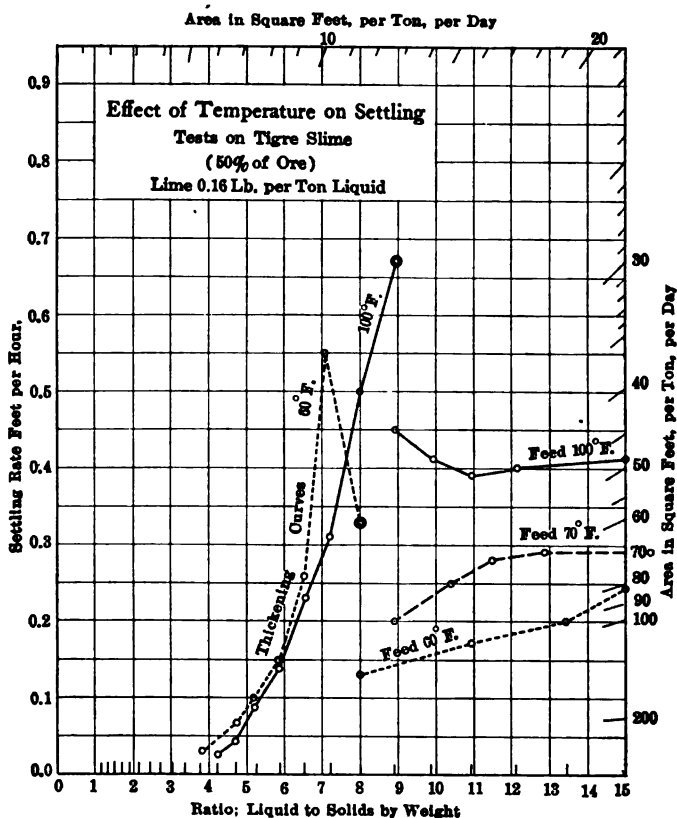


FIG. 4.

VARIOUS FACTORS AFFECTING SETTLING

Repeated mention is made in the foregoing discussion of the importance, in settling operations, of dilution and temperature. The effects of variations in these conditions, upon sundry phases of settling at Tigre, are graphically illustrated in Figs. 2, 3 and 4. The data, presented in these figures, may be briefly summarized as follows:

Effects of Dilution

In the free-settling zone, settling rates are greatest at high dilutions and lowest at low dilutions.

In the thickening zone, settling rates are greatly increased at dilutions just below critical dilution, reaching a maximum when the dilution is four-fifths that of critical dilution. Below this dilution, settling rates rapidly decrease as the dilution is reduced.

Effects of Lime

In the free-settling zone, settling rates of feed are trebled by increasing lime from 0 to 0.2 lb. per ton of solution, and are quadrupled by increasing lime from 0 to 0.7 lb. Further addition of lime above 0.7 lb. produces little advantage.

The critical dilution is doubled by increasing lime from 0 to 0.4 lb. Further addition of lime produces little effect on critical dilution.

At the higher dilutions in the thickening zone, lime up to 0.4 lb. greatly increases settling rate.

At the lower dilutions in the thickening zone variations in lime have little effect on settling rate, except in very shallow columns, or in the absence of classified sand.

Effects of Temperature

In the free-settling zone, an increase in temperature from 40° to 80° F. trebles settling rates.

Critical dilution increases slightly as the temperature is raised.

At the higher dilutions in the thickening zone, increase in temperature from 40° to 100° doubles settling rates.

At the lower dilutions in the thickening zone, changes in temperature have little effect on settling rates.

GENERAL CONCLUSIONS

High dilution, low lime and low temperatures predicate extensive settling area and tanks of shallow depth.

Low dilutions, high lime, and high temperatures predicate less area and comparatively deep tanks.

Adequate area is the first consideration in settling. Any advantage from increased depth can also be obtained from increased area. On the other hand, when area is inadequate, tank depth has no effect on results of settling.

SETTLING PROBLEMS AT EL TIGRE

Recent changes in treatment at El Tigre necessitated the reduction of lime in the mill water, from 1.5 lb. per ton to 0.02 lb. Before the changes were made, it was realized that this reduction in electrolyte

would greatly decrease the already low settling rate of Tigre pulp. Hence all conditions affecting settling were carefully reëxamined.

The ore at Tigre is crushed in stamp batteries to pass a 2-mesh screen. It is then ground to 20-mesh in a Hardinge mill. After concentration, it is further reduced, in tube mills, until 85 per cent. will pass a 200-mesh screen. The all-slime product is then settled to the *L:S* ratio of 5:1.

Tigre ore contains 25 per cent. natural slime. In the presence of this proportion of slime, classification of fine sand in the tube-mill circuit can be accomplished only at dilutions above 15:1.

To settle Tigre pulp, in the absence of strong electrolytes, requires extremely large settling area. This is especially true during the cold winter months. Preliminary laboratory tests indicated that in order to settle average Tigre ore from a dilution of 15:1 to a dilution of 5:1, the lime being 0.02 lb. and the temperature 45° F., it would be necessary to provide 110 sq. ft. of settling area per ton of ore settled daily.¹⁹ On this basis, 61 settling tanks, each 24 ft. in diameter, would be required to handle the Tigre tonnage of 250 tons per day.

The problem was solved, not by supplying this excessive equipment, but by modifying conditions elsewhere in the plant.

The dilution at which classification of sand can be accomplished in mechanical classifiers depends upon the coarseness of the sand to be separated and the proportion of natural slime in the pulp.²⁰ To separate 20-mesh sand, dilutions between 4:1 and 1:1 are usually required. The classification of finer sand necessitates higher dilutions. With 25 per cent. natural slime in the pulp, 200-mesh sand can be separated only at dilutions above 15:1. When the pulp contains 8 per cent. natural slime, satisfactory classification of 200-mesh sand can be effected at the dilution of 6:1. In the presence of 1 per cent. natural slime, classification is satisfactory at the dilution of 1:1. It thus appears that classification is a function of the dilution of the natural slime and is practically independent of the proportion of sand occurring in the pulp.

Settling rates, also, vary inversely with the percentage of natural slime in the pulp. It follows, therefore, that in order to obtain a thick, "all-slime" product from an ore containing an excessive amount of natural slime, it is important to separate the natural slime and sand, while the slime is undiluted and the sand is still coarse. The coarse sand can then be diluted sufficiently to insure good classification in the fine-grinding department of the plant, without necessitating extensive equipment for dewatering the product.

These objects are accomplished at El Tigre by modifications in the

¹⁹ This estimate has been subsequently verified by plant operations.

²⁰ Classification depends also on temperature and electrolyte, being most satisfactory when temperature and strength of lime are low. Any condition, tending to retard or prevent flocculation, aids classification.

stamp mill. The batteries are divided into two units, a mechanically operated classifier receiving the discharge from each unit. The sand from both classifiers is sluiced to the Hardinge mill with fresh water. Attrition in the Hardinge mill produces a certain amount of natural slime—from the disintegration of clay balls, etc. This is washed from the sand in hydraulic classifiers. The sand from the hydraulic classifiers is concentrated, and flows by gravity to the tube-mill room.

The slime overflow from the hydraulic classifiers is elevated and used as feed water in the first unit of batteries. The slime overflow from the classifier below the first unit of batteries is elevated and employed as feed water in the second unit. Thus the battery feed water is used in three operations, acquiring a load of slime in each operation. The final slime product, overflowing from the classifier below the second unit of batteries, has a dilution of 6:1 and requires no further thickening.

On account of the low dilution of the final slime product, much fine sand is carried over with the slime overflow of the second classifier. This is separated on Deister slime tables, which are equipped with deep riffles for the purpose. The shallow flow of slime across these tables and the sharp movements imparted by the mechanism prevent flocculation and effect a good separation of sand from the thick pulp. The slime tailing from the Deister tables tests 87 per cent. through 200-mesh, which is satisfactory. The sand product is concentrated on a recleaning table, equipped with standard riffles. Improved concentration is secured on the Deister tables by this roughing and recleaning process. The final sand tailing from the recleaning table joins the sand from the upper part of the mill and flows by gravity to the tube mills.

The low dilution in the battery discharge necessarily reduces tonnage through the batteries. This effect is counteracted by employing coarser screens and by using the same feed water successively in the two sets of batteries. An average daily tonnage of 12 tons per 1,200-lb. stamp is maintained.

Low efficiency in the batteries is offset by improved efficiency in the Hardinge and tube mills.

The cost of the successive elevations of slime in the stamp mill is offset by reduced cost of pumping from the bottom of the mill.

The sand entering the tube-mill room still contains some natural slime. More slime is liberated in the tube mills by the disintegration of partially kaolinized feldspar. The final overflow from the tube-mill classifiers contains 7 per cent. natural slime, and 93 per cent. fine sand. This product tests 85 per cent. through 200-mesh. The dilution is 7:1. It is settled to a dilution of 4:1 in 12 shallow settling tanks, each 24 ft. in diameter.

The combined dilution of final sand and slime products is 5:1, which is the dilution required.

Lime, 0.3 lb.
Temp., 73° F.

TABLE 1.—*Laboratory Tests on Tigre Pulp*

Test No.	L : S		D	T	Settling Rate
	Beginning of Test	End of Test	Depth of Clear Liquid Formed per Foot of Depth, Feet	Duration of Test, Minutes	$\frac{60 D}{T}$, Feet per Hour
1	14.5		0.05	3.50	0.86
2	13.1		0.05	4.50	0.66
4	10.7		0.05	6.50	0.46
5	9.6		0.05	7.75	0.39
7	7.8		0.05	9.50	0.32
9	6.3		0.05	11.00	0.27
10	5.7		0.05	12.00	0.25
11	5.1*		0.05	12.50	0.24

Settling rates of feed:

1	14.5		0.05	3.50	0.86
2	13.1		0.05	4.50	0.66
4	10.7		0.05	6.50	0.46
5	9.6		0.05	7.75	0.39
7	7.8		0.05	9.50	0.32
9	6.3		0.05	11.00	0.27
10	5.7		0.05	12.00	0.25
11	5.1*		0.05	12.50	0.24

Rates of free settling:

1	14.5		0.05	2.50	1.20
2	13.1		0.05	3.25	0.92
4	10.7		0.05	4.00	0.75
5	9.6		0.05	4.50	0.67
7	7.8		0.05	5.25	0.57
9	6.3		0.05	6.00	0.50
10	5.7		0.05	6.50	0.46
11	5.1*		0.05	7.00	0.43

Thickening test:

12	5.1*	4.6	0.10	14.00	0.43
13	5.1	4.1	0.20	37.00	0.32
14	5.1	3.7	0.30	108.00	0.17
15	5.1	3.3	0.40	223.00	0.11
16	5.1	2.8	0.50	383.00	0.08
17	5.1	2.5	0.60	720.00	0.05
18	5.1	2.3	0.70	1,380.00	0.03
19	5.1	2.0	0.80	4,520.00	0.01

* Critical dilution.

Thus the problem was solved by the installation of one-fifth the settling equipment estimated as necessary, and without sacrificing any element of efficiency in mill operations.

In the above discussion, an effort has been made to submit, as briefly as possible, any results of investigation at El Tigre which might have application elsewhere. Since tests have been limited to one class of ore, many of the observations can have only local importance, or at the most,

TABLE 2.—*Actual Settling Rates in Tank Compared with Settling Rates Determined by Test*

Tests on Tigre pulp (50 per cent. fine sand, 50 per cent. slime).

Observations in plant, over a period of 3 months.

Observations are averaged for days having similar temperature, dilution of feed, and amount of lime in solution.

No. of Days Averaged	Lime in Solution, Pounds per Ton	Temperature ° F.	Square Feet Settling Area per Ton per Day	R_1 L : S Feed	R_2 L : S Discharge	Settling Rate, Feet per Hour $\frac{1.34(R_1 - R_2)}{A}$	Settling Rate from Laboratory Test
1	0.15	45	14.0	4.75	3.70	0.10	0.11
7	0.15	55	14.9	4.75	2.75	0.18	0.15
3	0.15	55	15.5	5.48	3.30	0.19	0.17
1	0.15	55	23.0	7.00	3.87	0.18	0.21
2	0.15	65	9.5	3.44	3.12	0.05	0.06
2	0.15	65	12.4	4.46	2.80	0.18	0.17
1	0.15	65	13.6	5.16	3.16	0.20	0.19
4	0.25	45	9.5	4.23	3.10	0.16	0.11
6	0.25	55	12.2	4.76	3.10	0.18	0.15
3	0.25	55	12.7	5.23	3.46	0.19	0.16
2	0.25	55	12.4	7.05	4.21	0.31	0.22
8	0.25	65	11.2	4.26	2.88	0.16	0.16
10	0.25	65	11.1	4.73	2.83	0.23	0.20
5	0.25	65	12.2	5.41	3.25	0.24	0.24
1	0.25	75	9.1	3.85	3.17	0.10	0.13
3	0.25	75	10.8	4.64	2.69	0.24	0.22
6	0.25	75	12.0	5.52	3.69	0.21	0.25
3	0.25	75	13.0	6.68	4.30	0.24	0.28
1	0.35	55	9.2	3.48	3.08	0.06	0.06
1	0.35	55	22.0	6.40	3.00	0.21	0.19
1	0.35	65	14.2	3.64	3.16	0.05	0.08
8	0.35	65	11.4	4.56	2.81	0.20	0.19
5	0.35	65	11.5	5.29	2.74	0.30	0.26
1	0.45	65	10.8	3.86	2.84	0.13	0.12
5	0.45	65	11.3	4.61	2.72	0.22	0.22
Average.....						0.18	0.17

NOTE.—Critical dilutions: 4.0 : 1 at 0.15 lb. lime.

4.7 : 1 at 0.25 lb. lime.

5.1 : 1 at 0.35 lb. lime.

5.2 : 1 at 0.45 lb. lime.

serve to indicate what may be expected under like conditions elsewhere. In other instances, the investigations at El Tigre and the more comprehensive investigations of Coe and Clevenger, are mutually confirmatory,

TABLE 3.—*Thickening Test on Tigra Slime (47 Per Cent. of Ore)*

(All possible sand removed by previous classification)

Temperature 75°

Saturated lime solution

Tests begun below critical dilution and continued until settling ceased.

Tests performed in glass cylinders (1 ft. diam.).

Test No.	L : S Ratio		Average Settling Rates: Feet per Hour for Different Column Depths				
	Beginning of Observation	End of Observation	3 in.	6 in.	12 in.	24 in.	48 in.
1	7.7	6.9	0.060	0.110	0.180	0.340	0.340
2	6.9	6.3	0.040	0.060	0.080	0.300	0.600
3	6.3	5.6	0.030	0.040	0.070	0.140	0.680
4	5.6	5.0	0.021	0.037	0.047	0.066	0.424
5	5.0	4.5	0.015	0.028	0.033	0.056	0.088
6	4.5	4.1		0.023	0.022	0.022	0.020
7	4.1	3.6			0.010	0.009	0.009
8	3.6	3.2				0.004	0.005

No classification of sand observed.

No channels in 3-in. column.

Short channels in upper part of 6-in.

Strong channels in 12-in., 24-in. and 48-in. columns, but these closed during last part of test.

Only short channels (less than 2 in. long) occurred near the surface in the observations below the heavy line.

Settling rates below heavy line are independent of pulp depth.

No arching of the slime occurred and no cracks or pockets of clear liquid could be detected at the end of the tests.

making it possible to formulate, and employ with confidence, many general principles of settling.

Although all phases of settling have by no means been investigated, sufficient work has been done to render possible the formulation of certain standard methods for the solution of settling problems. Undoubtedly these methods will be simplified and rendered more accurate by future investigations.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, September, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Exploration of Metalliferous Deposits

BY W. H. EMMONS,* PH. D., MINNEAPOLIS, MINN.

(St. Louis Meeting, September, 1917)

Introduction

THE exploration of deposits of the metals will never become an exact science. There will always be an element of uncertainty in prospecting and developing mines. In countries where the surface has been closely scrutinized, most of the deposits whose outcrops contain valuable metals have probably been discovered. Many metalliferous deposits, however, are normally so much altered at the surface that the true significance of their outcrops is likely to be obscured.

If a deposit is not exposed, its presence may become known by some one of its characteristics that is different in kind or in degree from a similar feature of the associated rock. Thus, a deposit that is not exposed may be discovered because of its magnetic property. So few minerals are strongly magnetic that the value of magnetic surveys is limited, but the dip needle has nevertheless proved of great aid in locating magnetic belts, and these in some regions are associated with workable iron ores. The Cuyuna Iron Range of Minnesota, which does not exhibit any outcrop of iron ore, was discovered by drilling "areas of attraction" that are due to belts of magnetic rocks associated with the iron ores. In some regions the use of the dip needle is justified, even if it does no more than reveal the strike of the rocks, since a knowledge of the latter enables the driller to locate and point his holes to advantage.

The measures of gravity by the use of the pendulum are so accurate, or the limit of error is so small, that one might detect the presence of a concealed tabular deposit approximately at the surface, 34 ft. (10.36 m.) thick, having a density double that of the surrounding rocks. However, not many deposits have a density twice that of ordinary rocks and a thickness as great as 34 ft.; if they lie a short distance below the surface the density or the thickness must be even greater. Consequently, there is not much hope of sufficiently developing the gravity method of detecting deposits of heavy ores to make it of practical value. If the limit of error could be reduced to about one-tenth the present factor, and if the rather tedious and painstaking methods of determining gravity that are now in use could be simplified, the method would promise some degree

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of usefulness in the exploration of certain types of heavy ores. This, however, seems unlikely, since great refinement of method is necessary to determine gravity with the degree of error that now exists. In the future, as in the past, the explorer will probably have to rely upon what he sees rather than upon some peculiar characteristic of concealed deposits.

Certain relations of the distribution of lode ores to geologic structures have long been recognized. These ores are deposited principally in regions of complex faulting and fracturing and in areas of igneous activity. The earlier prospector found that it was profitable to scrutinize the mountainous regions, for these in general are most complexly faulted and fractured; and are more likely to have been the centers of great igneous activities. In some regions, it is true, flat-lying rocks far removed from igneous centers yield deposits of the valuable metals; these, however, are rarely of the lode type.

Certain features of a region other than its outcrops may lead one to suspect the presence of ore lodes. Thus, the presence of placers will lead to a search for the sources of the metals they contain. In some regions, waters containing iron sulphate suggest the presence of deposits containing iron sulphides. At some places, many feet above the present streams, gravels are cemented by iron oxides that were evidently derived from the weathering of deposits containing iron sulphides. At Cananea, Mex., at Bingham, Utah, at Lead, S. D., below the Homestake lode, and in many other regions, iron-cemented gravels are conspicuously developed. These gravels were consolidated, long before the mines above them were opened, doubtless by iron oxide that was deposited by hydrolysis of iron sulphate formed by oxidation of pyrite during the weathering of the pyritic deposits.

Lode ores, it is believed, are deposited principally by ascending hot waters. This inference is justified since they are almost universally associated with intrusive igneous rocks. The solutions that course through fractures and shattered zones soak into the country rock. In some districts these solutions have wandered far from the master fractures, profoundly altering great areas of country rock. Where the prevailing hydrothermal alterations are sericitic, as at Butte, Mont., and in many other regions, the country rock is bleached. Where the alterations are propylitic, as in many of the precious-metal deposits of Nevada, igneous rocks by development of chlorites from dark minerals, become pale green. Prospectors, even those without academic training, know the significance of hydrothermal metamorphism. Some designate the result of such alteration as the "kindly look" of the rock and contrast it with the "hungry look" of the fresh unaltered rock which they have often found to be barren of ores.

Primary ore deposits fall into a few well-defined groups, each with characteristic features. These in different districts have definite rela-

tions to the geologic structure depending on their genesis. The mapping of the structures and the investigation of the relations of the deposits to structures are essential features of rational exploration.

Of the discoveries made in the United States within the past 10 years, perhaps as many have resulted from a knowledge of the geologic conditions of the districts containing the deposits as from the exploration of outcrops that were supposed to be the altered cappings of valuable deposits. The exploration made with an adequate knowledge of the structure of an area and also with an understanding of the characteristic superficial alteration of its deposits is the most likely to succeed.

Some outcrops themselves contain the valuable metals; others have had the valuable metals leached from them, but may carry the alteration products of associated minerals and these frequently furnish the clue leading to the discovery of orebodies. Pyritic copper sulphide deposits and pyritic zinc sulphide deposits are generally leached of copper and zinc near the surface. In the earlier days of mining many of the deposits were discovered incidental to mining associated metals less readily dissolved at the surface. In general these deposits were stained with iron oxide. The copper deposits and zinc deposits of Butte, Mont., were discovered by exploration of silver ores in the upper oxidized parts of the veins. The United Verde mine, Jerome, Ariz., the Highland Boy mine, Bingham, Utah, and the Mount Morgan mine, Queensland, were worked first as gold mines. All subsequently developed great bodies of copper ores carrying noteworthy amounts of gold.

Stimulated by examples of iron-stained rocks passing downward into workable copper ores, explorers investigated ferruginous outcrops systematically and many deposits were discovered below the valueless cappings. Many of the disseminated copper deposits are barren of copper at the surface. The exploration of these deposits soon showed, also, that some are not everywhere capped by gossans heavily stained with iron oxide. The cap rock in general is composed largely of sericite, kaolin, and chalcedonic silica. Some contain also sulphates. Such associations, though not so easily recognized as limonitic areas, are nevertheless significant and the recognition of the origin of such an outcrop is certain to play an increasingly important part in the explorations of the future. Because such an association of alteration products is less readily recognized, outcrops of this character are the most likely to have been overlooked in an area that has not been fully explored.

The problems connected with the superficial alteration of ore deposits, although complex, are susceptible of analysis and experimental study. The changes are accomplished by water, air, and the compounds resulting from the action of water and air upon the ore itself. One may take the ore, expose it to water and air, and ascertain what products are formed. One may expose the ore further to the action of water and the

products of alteration in the absence of air, and ascertain the changes that take place. By analysis one may ascertain the composition of the waters of mines. There are now available more than 50 analyses of waters from mines of sulphide ores.¹ These are similar in composition and closely resemble solutions formed by placing sulphide ore in contact with pure water. It may be assumed with confidence that solutions that have accomplished superficial alteration of sulphide ores are systems of sulphates, carbonates, and chlorides, of heavy metals, of alkalis and of alkaline earths.

Near the surface solutions are acid and generally contain ferric sulphate. In depth, ferric sulphate is reduced to ferrous sulphate, acidity is decreased by reaction with minerals of the ore and wall rock, and at greater depths the solutions become neutral and ultimately alkaline. Subjecting for long periods various minerals and various combinations of minerals to solutions such as are known to accomplish superficial alteration, one may ascertain what changes take place and compare the reactivities of ores of the various metals and their reactivities in various mineral associations.¹

Some of the metals are easily dissolved near the surface where waters are acid and oxidizing. Of these some are precipitated in depth where acidity decreases and where solutions are reduced because oxygen of the air is excluded. Deposits of such metals are likely to be leached at the surface. If the metals are readily precipitated in depth the deposits become enriched below the surface. Other metals are very difficultly soluble and because associated materials are dissolved and removed deposits of such metals are likely to be enriched at the surface. Associated gangue minerals are important also because some metals dissolve readily in certain associations but not in others. One may separate the metals into groups—one of metals that dissolve readily, another of metals that dissolve slowly, and still another of metals that dissolve very slowly.

1. Dissolve Readily

copper
zinc
silver
gold (in part)
uranium
vanadium
iron
manganese
nickel
cobalt

2. Dissolve Slowly

mercury
lead
antimony
arsenic

3. Dissolve Very Slowly

gold (in part)
bismuth
tin
chromium
molybdenum
tungsten

Copper

Copper, because of its chemical relations, is easily leached from the surface and precipitated in depth. As already observed, the outcrops

¹ W. H. Emmons: The Enrichment of Ore Deposits, *U. S. Geological Survey*, (1917), *Bulletin* 625.

of a copper sulphide deposit may be so thoroughly leached that practically all the copper is removed and carried downward.

In base-leveled countries or in countries where the surface has remained nearly stationary for a long time, the outcrops are generally depleted of copper. Even in mountainous countries, where erosion is comparatively rapid, not many large deposits of copper are workable at the surface. Ferric sulphate hydrolyzes, depositing limonite, so the deposits of many iron-copper sulphide ores are marked by a gossan or "iron hat." As already noted, many copper deposits have been discovered by following downward a nearly barren gossan or by the downward exploitation of deposits of precious metals that are concentrated near the surface above deposits of copper ores in which the precious metals are present in small amounts.

In copper deposits that do not carry sulphides the downward transportation of copper is generally slow. The native copper deposits of Keweenaw Point, Mich., are workable at the surface, although the country has undergone erosion for a period so long that it has become nearly a peneplain.

Where the sulphides are present in subordinate quantities, copper carbonates and silicates may occur abundantly at and near the surface, as at Ajo, Ariz.,² where oxidized copper minerals are conspicuous in outcrops.

In limestone, copper will commonly segregate as carbonate at and near the surface, and many oxidized copper deposits in limestone have been worked by open pits. Limestones that have been altered by contact metamorphism are relatively impermeable, because their tough, heavy silicates, such as garnet, amphibole, and mica, are not readily fractured. Most deposits of this nature contain considerable calcite, and any copper-iron sulphide ore they carry will usually oxidize to carbonates, silicates, and oxides. The copper in such an ore is particularly stable and is likely to endure long weathering. Such deposits have stimulated deep prospecting in many districts where other types of deeper copper ores are present,³ and they have thus served as useful indicators of hidden wealth.

The copper lodes at Butte (in igneous rocks) are leached of copper, some of them to a depth of 400 ft. (121.9 m.). The disseminated ores in porphyry show great variation as to depth of leaching, but are commonly leached to depths of 100 to 300 ft. (30.5 to 91.4 m.) below the surface, and exceptionally to greater depths. Some of them show practically no copper at the surface. At Cananea, Sonora, and Morenci, Ariz., barren gossans that were explored to considerable depths have led to good

² I. B. Joralemon: The Ajo Copper-Mining District, *Trans.*, (1915), 49, 601.

³ W. H. Emmons: The Outcrops of Ore Deposits, in *Types of Ore Deposits*, Bain, H. F., and others, p. 318. San Francisco, 1911.

deposits of chalcocite ore. Copper was only sparingly present at most places in the outcrops of the great disseminated deposits at Miami and Ray, Ariz. At Bingham it was locally somewhat conspicuous as carbonates and silicates.

In making explorations for copper the question frequently is raised whether drilling is justified in an area that shows but little iron oxide at the surface. Nearly all copper deposits mined in North America do show ferruginous outcrops, but some gossans that cap valuable disseminated ores in porphyry are not heavily stained with iron. At Cananea, Sonora, valuable chalcocite deposits occur below outcrops that show heavy iron stain only here and there. As stated above, however, the outcrops show much silicification and kaolinization, and generally some limonite is present.

As a rule, the disseminated deposits of chalcocite ore will show outcrops more highly stained with iron in the earlier stages of their chalcocitization. As the country is eroded and the chalcocite zone descends, more and more pyrite and chalcopyrite are replaced by copper sulphides. A point may be reached where the chalcocite ore contains very little of the original iron sulphide. Obviously the oxidation of such an ore would yield but little iron sulphate. It would yield even less limonite, if the solutions were actively descending. If the process goes still further, the proportion of iron may become insufficient to dissolve all the copper in the chalcocite zone. The gossan then will generally carry oxidized copper minerals such as cuprite, the carbonates, basic sulphates, and chrysocolla.

Zinc

Zinc, like copper, is very readily dissolved from its sulphide deposits and in the presence of much pyrite or other iron sulphide, zinc near the surface is likely to be almost completely removed from its deposits. It forms the very soluble zinc sulphate. But zinc in depth is not so readily precipitated by sulphides as copper. It will remain dissolved in solutions from which copper may be precipitated. Thus secondary concentrations of zinc sulphide ores are rare compared with the secondary ores of copper.

In limestone, a solution of zinc sulphate precipitates zinc carbonate ores. These ores have lately become prominent, particularly in several districts of western America, where superficial changes have produced notable concentration of zinc.⁴

⁴ G. M. Butler: Some Recent Developments at Leadville, *Economic Geology*, (1913), 8, 1.

Adolph Knopf: Mineral Resources of the Inyo and White Mountains, Cal., *U. S. Geological Survey*, (1914), *Bull.* 540, 97.

G. F. Loughlin: The Oxidized Zinc Ores of Tintic District, Utah, *Economic Geology*, (1914), 9, 1.

A common type of ore in limestone consists of pyrite, argentiferous galena, sphalerite, a little chalcopyrite, and other sulphides in a gangue of quartz. The orebodies, like many deposits in limestone, are commonly large irregular masses. In the oxidation of such a deposit the lead and much of the silver remain essentially in place, the galena being in part oxidized to anglesite and cerusite. The oxidation of pyrite and sphalerite yields acid, and zinc and iron sulphates, which are carried out of the deposits in great quantities. A part of the iron remains behind as oxide, but in some deposits practically all the zinc is removed. When the solution, which is doubtless acid and carries ferric, ferrous, and zinc sulphates, moving along a water channel, encounters the limestone that surrounds the orebody, it will precipitate iron and zinc.

Under some conditions a zinc-iron carbonate or sideritic smithsonite, monheimite, is formed. This reaction has recently been investigated by Wells.⁵ Dilute solutions of two metallic salts in equivalent (molar) quantities were precipitated with only enough sodium carbonate for one metal. With equivalent quantities of zinc and calcium nearly all the zinc and only a trace of calcium are precipitated. With equivalent quantities of iron and calcium, nearly all the iron and only a trace of calcium are precipitated. Sideritic smithsonite, or monheimite, contains iron carbonate in varying proportions. Some smithsonite is nearly pure and some contains as much as 20 per cent. of iron carbonate, or even more.

After it is formed the smithsonite, or monheimite, with the progress of the erosion of the country, is exposed to more highly oxygenated waters. The iron carbonate then oxidizes and stains the ore brown so that it may easily be mistaken for iron-stained limestone. Thus deposits of this character, though exposed in underground workings, have been overlooked for years.

Silver

Silver, like copper, is readily dissolved in ground waters. If the chloride forms in the gossan its solution is delayed. The chloride, however, is to be regarded as a temporary mineral except in arid countries⁶ where chlorine is commonly present in considerable quantities in earth waters. In such countries more chloride forms and, moreover, its solution is prevented by chlorides present in ground waters. There may, therefore, be great enrichment of silver at and near the surface. In depth, silver is carried in acid solution. Air or ferric iron is necessary for its

⁵ R. C. Wells: The Fractional Precipitation of Carbonates, *Washington Academy Scientific Journal*, (1911), 1, 21.

⁶ R. A. F. Penrose, Jr.: The Superficial Alteration of Ore Deposits, *Journal of Geology*, (1892), 2, 288-317.

active solution in sulphate waters.⁷ However, the carbonate of silver is soluble and without much doubt some silver is carried downward in carbonated waters. Silver is precipitated from its sulphate⁸ and carbonate⁹ solutions in a sulphide environment. For this reason great bodies of secondary silver ore may be deposited below a leached or low-grade capping.

Gold

In acid waters that carry chloride in the presence of manganese oxides,¹⁰ gold is readily dissolved; by reduction of acidity, gold is precipitated. Frequently the manganese and gold are precipitated together; in deposits that contain manganese, gold may be carried downward in solution and accumulate below the outcrop. But gold is readily precipitated from its solutions by many minerals, and its migration is slow. In the presence of carbonates,¹¹ or of other minerals that reduce acidity readily, gold in manganiferous deposits tends to remain near the surface and in manganiferous carbonate gangue it may even accumulate as placers. If the metal were not of great value, its secondary concentration would be of little economic importance because it is dissolved near the surface only in chloride solution and in solutions that reconcentrate sulphide ores and chlorides are much less abundant than sulphates. In deposits which do not contain manganese, gold dissolves very slowly, if at all, and tends to accumulate at the surface. Such deposits are commonly enriched by removal of material other than gold.

Uranium and Vanadium

Uranium and vanadium are readily dissolved in sulphate waters and both are regarded as mobile metals. Both metals are precipitated from the soluble salts by organic matter.¹² The carnotite deposits of Colorado¹³ are believed to be concentrations from cold ground waters that dissolved the metals from associated rocks. In many deposits, vanadium minerals occur in relations that leave no doubt of their secondary origin.

⁷ H. C. Cooke: The Secondary Enrichment of Silver Ores, *Journal of Geology*, (1912), **21**, 9.

⁸ Chase Palmer and E. S. Bastin: Metallic Minerals as Precipitants of Silver and Gold, *Economic Geology*, (1913), **8**, 140.

F. F. Grout: *Idem.*, (1913), **8**, 407-433.

⁹ L. G. Ravicz: Enrichment of Silver Ores, *Economic Geology*, (1915), **10**, 368-392.

¹⁰ W. H. Emmons: The Agency of Manganese in the Superficial Alteration and Secondary Enrichment of Gold Deposits in the United States, *Trans.*, (1913), **42**, 3-73.

¹¹ A. D. Brokaw: The Secondary Precipitation of Gold in Orebodies, *Journal of Geology*, (1913), **21**, 251-268.

¹² F. L. Hess: An Hypothesis for the Origin of the Carnotites of Colorado and Utah, *Economic Geology*, (1914), **9**, 675-688.

¹³ W. F. Hillebrand and F. L. Ransome: On Carnotite and Associated Vanadiferous Minerals in Western Colorado, *U. S. Geological Survey*, (1905), *Bulletin* 262, 14.

Iron and Manganese

Iron and manganese are grouped with the metals that are readily dissolved in sulphide ores. In sulphide deposits containing iron and manganese these metals are generally present in ground waters. The iron and manganese solutions, however, are not very stable and the oxides of iron and manganese are readily precipitated by hydrolysis. Consequently deposits that contain appreciable amounts of iron and manganese will generally carry these metals at the surface. Secondary deposits of iron sulphide or manganese sulphide are almost unknown. At depth, iron and manganese are deposited along fractures as oxides and more rarely as carbonates, but these ores in lode deposits are of relatively small importance. Deposits of iron and manganese are commonly enriched by removal of other materials. In general, iron and manganese deposits carry rich ores at the surface.

Nickel

Nickel is similar to iron in its chemical activities, but unlike iron it does not oxidize in bivalent to form trivalent salts and its sulphate does not hydrolyze readily and deposit oxide. It is dissolved almost as readily as copper in its sulphide combinations and it is precipitated in depth as sulphide. It is not so easily precipitated as copper sulphide, however, and will not be thrown down in an acid environment. It therefore resembles zinc in its migrational reactivities more closely than it resembles copper. No large deposits of secondary nickel sulphide have been recognized. The best-known deposits of nickel sulphide ores have been glaciated and possibly secondary sulphide zones have been removed. At the Lancaster Gap nickel mine,¹⁴ Pennsylvania, secondary millerite has been present in bodies of economic value. The gossan of nickeliferous pyrrhotite is essentially limonite. In the weathering of nickeliferous basic rocks the nickel accumulates as silicates not far below the surface.

Cobalt

Cobalt salts are soluble and cobalt is dissolved readily from outcrops. No deposits of secondary sulphides are known. Its chemical behavior is closely similar to that of nickel and zinc and it should be grouped with the metals which are concentrated under favorable conditions. Like nickel, cobalt forms silicates and concentrates in the superficial zones of silicate deposits which are undergoing alteration. Cobalt, like nickel, forms a moderately insoluble arsenate in some outcrops. Cobalt is found in considerable quantities in the oxidized material which caps the altered peridotite of New Caledonia.

¹⁴ J. F. Kemp: The Lancaster Gap Nickel Mine, *Trans.*, (1894), **24**, 620.

Mercury

Mercury is dissolved and reprecipitated in underground waters. The reactions go on very readily with chloride waters but not in sulphate solutions. Mercuric chloride is soluble in water and does not give insoluble basic salts with water. Mercuric sulphate, on the other hand, is easily hydrolized and gives basic sulphate which is reduced to native metal. Experiments made over long periods by Broderick¹⁵ show that cinnabar is practically insoluble in sulphate waters although it is readily dissolved in hydrochloric acid, and more readily still in presence of manganese oxide. The secondary enrichment of mercury deposits is somewhat similar to that of gold deposits. Neither metal is appreciably dissolved except in presence of chlorides. Reconcentration of mercury is most marked in arid countries, where chlorides are present. Chlorine, however, is quite subordinate in underground waters compared with sulphates. Underground waters, therefore, can not move large masses of the metal and concentrate them as copper and silver are concentrated. Gold is so valuable that even small concentrations are important economically. Mercury is less valuable: a concentration of 2 oz. of gold per ton would be highly significant, but 2 oz. of mercury per ton would be unimportant. There is indisputable evidence that secondary mercury minerals result from processes of sulphide enrichment, but these are small in amount because chlorides in general are not abundant in waters of deposits of mercury ores. Mercury must be placed with metals that are not migratory.

Lead

Lead is one of the least soluble of the common metals, although the chloride is fairly soluble and the sulphide is dissolved in sulphuric acid in the presence of an oxidizing agent. The oxidation is attended by the formation of the relatively insoluble lead sulphate, anglesite. This coats the sulphide and tends to delay solution. Deposits with ores carrying galena will generally contain oxidized lead minerals near the surface. It is not uncommon to find galena partly altered to anglesite in the outcrops of deposits containing lead sulphide. Lead minerals, because of their low solubilities, are useful indicators of mineral deposits, and at many places have led to the discovery of silver, gold, and zinc deposits. Like deposits of other difficultly soluble metals, lead deposits are likely to be enriched by removal of valueless materials rather than by concentration in depth by the processes of solution and precipitation.

¹⁵ T. M. Broderick: Some Experiments Bearing on the Secondary Enrichment of Mercury Deposits, *Economic Geology*, (1916), 11, 645-651.

Antimony

Antimony dissolves very slowly in the oxidizing zone. Its sulphide deposits form the oxides which are about as stable as the secondary lead minerals, cerusite and anglesite. In depth, however, antimony minerals dissolve readily in alkaline solution. The antimony salts unite with silver-bearing solutions and precipitate antimony sulphosalts of silver. Antimony is not to be classed as a readily migratory metal though it plays an important part in the precipitation of the secondary silver minerals.

Arsenic

Arsenic dissolves rather readily in acid solutions, but if it is present as a salt of H_3AsO_3 , much water will hydrolyze it, and if present as salts of a base-forming element these also are hydrolyzed. In an acid solution its salts are oxidized but do not migrate extensively. In alkaline solutions, however, they are easily dissolved and in depth the activities of arsenic in connection with the secondary enrichment of silver are important.

Bismuth

Bismuth is more or less closely allied to antimony and arsenic. Like them it forms oxides near the surface in its deposits. There is no evidence of its extensive migration. Reasoning from its chemical relations it is not likely that important secondary zones formed by the migration of bismuth will be found. Its sulphate hydrolyzes readily in water and in weak acid, forming oxides, and its other salts also hydrolyze so readily that they would not be transported.

Tin

Tin is one of the inert metals. The stannous salts, both sulphate and chloride, are easily soluble, but the stannic salts are readily hydrolyzed. Thus, in an oxidizing environment such as obtains in sulphide ores near the surface of the earth, the solution of tin is exceedingly difficult. The stannous salts when formed are oxidized to stannic salts and stannic salts will break down almost at once to the insoluble oxide, cassiterite.

Thus tin, although it may go in solution, will be almost immediately precipitated. Experiments by J. P. Goldsberry in the geological laboratory of the University of Minnesota have shown that cassiterite and stannite are both practically insoluble in tenth normal sulphuric acid and hydrochloric acid. Doelter observes that tin oxide is slightly soluble in water. This statement was found by Goldsberry to be erroneous. The method of determination would have detected one part in a million. At

the end of one month, tenth normal sulphuric acid, in contact with stannite and cassiterite, showed only faint traces of tin dissolved. Tin is enriched but little by migration in its deposits, because of the hydrolysis of its stannic salts which, as above stated, are precipitated as the insoluble oxide.

Chromium, Molybdenum, Tungsten and Uranium

The metals—chromium, molybdenum, tungsten, and uranium—are closely affiliated chemically and all form acid trioxides like that of sulphur, SO_3 . Chromium forms also the very stable basic oxide Cr_2O_3 which resembles ferric oxide in its properties. It forms in silicate environment and is a constituent of igneous rocks. In sulphide deposits chromium is dissolved and reprecipitated, but so far as any evidence is available it is not very migratory. Molybdenum and tungsten oxidize slowly to secondary minerals which generally remain near the parent primary minerals.

Molybdenite is not dissolved in hydrochloric acid nor in sulphuric acid, nor, at the end of one month, in the presence of ferric salts.

Tungsten forms tungstic acid, $\text{H}_2\text{WO}_4 \cdot \text{H}_2\text{O}$. This is somewhat soluble in water. In moist climates, it could be leached out of deposits which were exposed long enough, but the reaction is exceedingly slow. Tungsten minerals are so insoluble that they commonly form placers. Both molybdenum and tungsten are classed with the non-migratory metals.

Uranium, on the other hand, is carried in underground waters and doubtless forms secondary deposits of considerable importance. The deposits of southwestern Colorado, as already stated, are believed to be reconcentrations by ground water near the present surface.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, September, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Tayeh Iron Ore Deposits

BY CHUNG YU WANG, HANKOW, CHINA

(St. Louis Meeting, September, 1917)

DURING the time I was in charge of this mine, from 1914 to 1915, I had occasion to read the interesting papers by T. T. Read and C. M. Weld about these deposits, to find how far their observations corroborated mine. To quote from the paper on "The Mineral Production and Resources of China" by T. T. Read:¹ "The iron ores at Tayeh * * * * lie along the contact between a marble and an intrusive body of a dark gray syenitic rock. * * * * At one place it is slightly magnetic, apparently having been partly reduced to the magnetic oxide by the action of reducing solutions, which have deposited small amounts of copper and iron sulphides along the foot wall." To quote from the paper on "The Tayeh Iron Ore Deposits"² by C. M. Weld: "I find the orebodies described as occurring along an approximately E-W contact between hornblende-granite and limestone. * * * * The granite is a granitoid rock consisting essentially of quartz, potash-feldspar and hornblende." These quotations show the differences of opinion entertained by two observers. My own study of these interesting deposits has been intermittent amid the manifold duties devolving upon a superintendent.

General Description of the Deposits

The deposits are situated to the west of the Yangtse River, being connected with the river by a standard-gage railroad of 56 li (16.16 miles, 26 km.) in length, as shown in Fig. 1. The general topography of the region is of mature age, showing broad valleys with sluggishly meandering streams. These deposits are worked as two mines, named the Tieh Shan Mining Department and the Teh Tao Wan Mining Department; the former consisting of three orebodies more or less connected, named Tieh Man Kan, Sha Mau Tze and Lung Tung, and the latter consisting of Chang Pei Shan, Sze Tze Shan, Ta Shih Men, Yeh Chih Ping and Tsin Shan, also more or less connected, as shown in Fig. 2.

¹ *Trans.* (1912), **43**, 29-30.

² *Trans.* (1912), **44**, 27-37.

near the partially decomposed rock next to the orebodies; the other, mottled white, which is farther off from the deposits and can be found as boulders and pebbles along the streams. The limestone occurs in various stages of crystallization, in which hardly any trace of contact minerals can be found. I am indebted to my assistant, W. A. Wong, for having found a layer of limestone thickly studded with crystals of garnet, not far from the orebody; the only layer of limestone with such contact mineral that I saw during my stay in Tayeh.

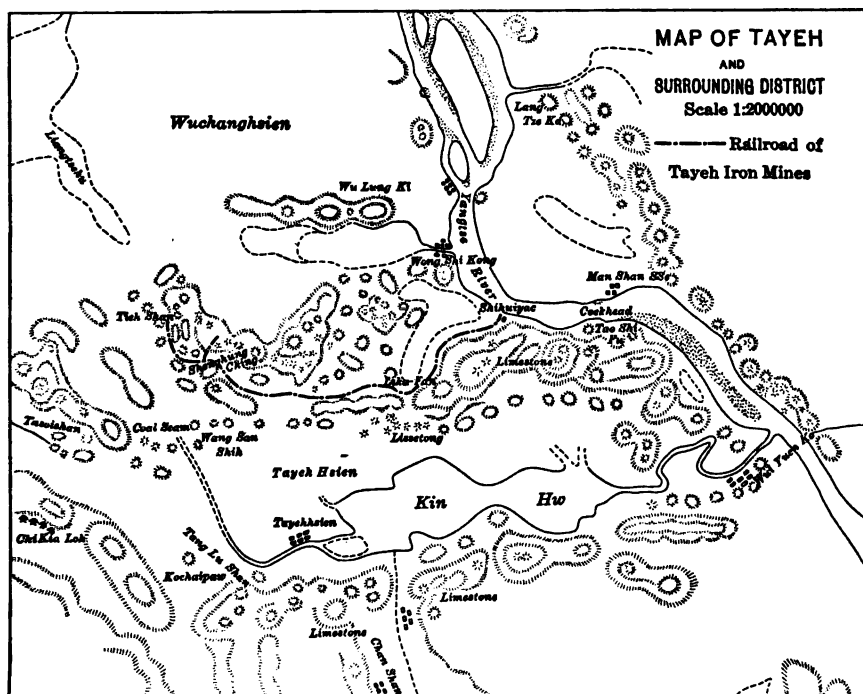


FIG. 2.

Description of the Deposits

From Fig. 2 it will be noticed that the deposits are more or less lens-shaped, having an approximate NW-SE strike and a NE dip of 60° to 70°, with the exception of the orebodies at Sha Mau Tze and Tsin Shan, which dip away from the igneous rock (SW dip); that is to say, the hornblende syenite forms the hanging wall, which is generally sharply distinct from the orebodies themselves, and the limestone the foot wall, which in many places is very irregular in outline, inclosing buncy pieces of limestone, ranging from 2 to 10 ft. in diameter. The orebody at Tieh Man Kan has been faulted from the main trend of the other orebodies in a southwesterly direction. The iron ore is essentially composed of a mix-

ture of magnetite and hematite, with occasional occurrences of malachite, chalcocite and pyrite and rarely epidote and mica. Table 1 shows average analyses of the ore and of the limestone.

TABLE 1.—*Average Analyses of Ore and Limestone*

Ore:

No.	Localities	For	Fe, Per Cent.	Mn, Per Cent.	S, Per Cent.	P, Per Cent.	Al ₂ O ₃ , Per Cent.	Cu, Per Cent.	SiO ₂ , Per Cent.	Sp. Gr.
1	Teh Tao Wan	Wakamatsu & Hanyang	62.93	...	0.029	0.049		0.238	5.42	4.64
2	Tieh Shan	Hanyang	59.45	0.63	0.319	0.109	1.05	0.292	9.16	4.13

Limestone:

SiO ₂ , Per Cent.	Al ₂ O ₃ Fe ₂ O ₃ , Per Cent.	CaO, Per Cent.	MgO, Per Cent.	Sp. Gr.
3.14	1.14	52.63	0.92	2.71

Analyses of the discarded ore dumped into the waste heap have been found to contain as much as from 1 to 2 per cent. S and from 0.5 to 3 per cent. Cu. All the orebodies are more or less jointed and the jointing planes are in a general way parallel to the hanging wall, along which is frequently found copper carbonate which must have been deposited and concentrated there from the leaching of the chalcocite found disseminated in certain portions of the orebodies.

Origin of the Deposits

I consider that these orebodies belong to the contact-deposit type and are the products of contact metamorphism; that the magnetite is primary—it has been in places partially or wholly altered to hematite, due to hydrothermal and pneumatolytic reactions, concomitant with the intrusion of the hornblende-syenite—and that the hematite thus originated has been carried forward by ascending telluric water, probably together with the aid of descending meteoric water, to replace metasomatically the intrusive and the limestone, especially along the contacts at certain particular spots. I have arrived at this conclusion from a consideration of the following facts:

1. As noticed above, the hornblende syenite along nearly the whole length of the contact is in a decomposed and altered condition, due undoubtedly to hydrothermal- and hydro-metamorphism, that involves the development of kaolin and chlorite with traces of epidote, pyrite, quartz and calcite. This may be the result of the process of propylitization.

2. At Ta Shek Man, the hornblende-syenite changes to a light-brownish and yellowish color and is rather soft, with scales of sericite, distin-

guishable megascopically. This is probably the result of sericitization, developing prominently in this particular locality.

3. The crystallization of the limestone to marble is undoubtedly due to its contact with the hornblende-syenite magma, with the development of a banded structure in the limestone, particularly discernible at Sha Mou Tze. This banded structure may probably be due to differing degrees of silicification.

4. Near the main orebodies at Teh Man Kan, irregular smaller bodies of ore rich in manganese are found in the limestone, which must owe their origin to the same magmatic emanation.

5. I regard the magnetite as primary and derived from magmatic emanation rich in iron, and the hematite as a secondary product from the



FIG. 3.—DECOMPOSED HORNBLLENDE SYENITE TOTALLY INCLOSED BY HEMATITE. DEPOSIT AT YEH CHIH PING.

oxidation of the magnetite. We can picture to ourselves how this process of hematization can be brought about. Chemically magnetite is composed of the sesquioxide Fe_2O_3 which is hematite, and the protoxide FeO , which, being unstable, would readily be oxidized to Fe_2O_3 in the presence of active free oxygen under suitable pressure and temperature. Undoubtedly the gaseous vapor, during the intrusion of the syenite, furnished such oxygen. O. Silvestri has found as much as 18.97 per cent. of oxygen in the gas issuing from Mt. Etna. It is generally known that chemical reaction may or may not be favored by an increase of pressure, and this is why magnetite cannot be hematized either at or near the surface, or at great depth. Probably the gaseous emanation rich in magnetite and oxygen, on coming up from below, reaches a particular zone where

the pressure and temperature are suitable to the combination of the oxygen and magnetite to form hematite.

6. Some of the hematite thus produced may be carried forward by telluric water, probably aided by descending meteoric water to replace metasomatically some portion of the rocks of both the hanging wall and the foot wall. This process of replacement is clearly shown from Figs. 3 and 4 where decomposed hornblende syenite is totally inclosed by hematite; from the occurrence of big bunchy pieces of limestone, as de-



FIG. 4.—FORMATION AT TSIN SHAN OF SAME NATURE AS THAT SHOWN IN FIG. 3.

scribed above, inclosed entirely by hematite, and from some hand specimens in which particles of altered syenite are in process of being replaced by hematite.

Production

The following table shows the output of iron ore from 1896 to 1914 inclusive, which, in round numbers, is about $3\frac{1}{2}$ million tons. The ore-in-sight of these deposits, including Chang Pei Shan, which, being Government property, has not yet been opened, has been estimated by three engineers to be 20, 30, and 100 million tons respectively. But according to my judgment, it would not be unreasonable to regard the ore-in-sight as 30 million tons and the probable-ore in depth as 30 million tons.

Year	Tons
1896.....	15,933
1897.....	20,545
1898.....	36,558
1899.....	24,765
1900.....	57,201
1901.....	109,215
1902.....	84,036
1903.....	107,794
1904.....	106,378
1905.....	151,168
1906.....	185,610
1907.....	174,612
1908.....	171,934
1909.....	309,399
1910.....	343,097
1911.....	359,467
1912.....	268,685
1913.....	416,342
1914.....	488,258
	3,430,976



Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

Discussion of the paper of MAX ROESLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1789 to 1839.

W. L. CUMINGS,* Bethlehem, Pa. (communication to the Secretary†). —In this discussion of Mr. Roesler's paper, I shall follow Kemp in using the term "granite" to refer to the acid rock called syenite by Kimball. Diorite, as I understand it, is the dark basic rock invariably closely associated with the orebodies. The types of rock of intermediate composition, mainly intrusives, are not under discussion.

It is difficult for me to conceive that this granite is the ore-bearing agent, for the following reasons:

1. No isolated orebodies in the granite are known. It would seem that if this rock is the mineralizer, some small bodies, or at least patches and small masses of ore so situated would exist.

2. In 1909 the writer conducted explorations on the Pequena and Grande claims about 1 mile north of West Mine No. 5. This work was done after a very careful magnetic survey had shown a large and uniform area of attraction. As exploration progressed, it became evident that in this case the magnetic attraction was due to disseminated magnetite. Much ore of 40 per cent. Fe was found, but very little commercial ore was shown up. This ore was entirely in the diorite and no granite outcrops are near. The occurrence could easily have been described as "ore formation but very little ore." I cannot conceive how a remote granite intrusion could impregnate a diorite mountain with 40 per cent. of Fe, and prefer to believe that this 40 per cent. of iron was in its real home, rather than in a sort of dioritic "boarding house" after having been evicted by granitic parents.

3. The Ocaña mine, which has produced nearly 500,000 tons, shows no evidence that the granite is the mineralizer. This is admitted by Mr. Roesler on p. 1827.

4. I believe it will not be disputed that all the iron-ore deposits extending along the south coast for 60 or 70 miles are of a similar origin. Evidence available at other points might properly be referred to in discussing the article under review. At the La Plata mines, about 60 miles west of Santiago, the writer has also made explorations. No granite is in evidence. At one point in the bed of a creek is a small body of magnetite about 10 ft. long, entirely surrounded by a hard fresh diorite. To me

* Geologist, Bethlehem Steel Co.

† Received Jan. 27, 1917.

it is difficult to believe that a granite, which was not in evidence, was responsible for depositing this ore in the diorite. Had the latter been altered to a degree that would permit the entrance of ore-bearing solutions, the granite theory would seem more plausible.

In a general way I would comment on the lack of credit given by Mr. Roesler to the magnetic surveying which has been done at Juragua. One might infer, from his recommendation of magnetic surveying on pp. 1838-39, that he was recommending something new. As a matter of fact, it is probable that no area in the United States had been mapped magnetically more thoroughly than Juragua at the time Mr. Roesler wrote his article. The first work was done in 1903 and the work was taken up again in more detail in 1906. This work, in connection with actual exploration, has continued ever since, as other work permitted. Finally, in 1915-16, N. M. Gibson was employed to go over the whole property in order to make sure that no area of attraction had been overlooked. The expression, "every foot of the ground has been gone over," is almost literally true in this case.

Magnetic surveying has also been done at Daiquiri.

In conclusion, it would seem that little of a practical nature has appeared in any of the four recent articles on Juragua and Daiquiri geology. If one studied the four articles carefully preparatory to planning exploration work, he would probably be badly confused in trying to reconcile statements and trying to find out just exactly what the different authors were really trying to prove.

In the meantime, the writer is of the opinion that about the only rule for exploration in the south coast of Cuba is, "stick to the diorite or 'piedra azul'" of the miners, and keep away from the granite. In exploring magnetic areas, be careful that the attractions are not caused by diorite impregnated with iron ore (iron-ore formation), or that they are not caused by areas of contact garnet rock, which, in places, is highly magnetic.

Recent Geologic Developments on the Mesabi Iron Range, Minnesota

Discussion of the paper of J. F. WOLFF, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1763 to 1787.

CARL ZAPFFE, Brainerd, Minn. (communication to the Secretary*).—Unless one has actually directed explorations for iron ore, it is doubtful whether the importance of Mr. Wolff's paper, the value of the information contained, and the large amount of detail work involved in what is set forth can be fully appreciated. It is indeed rare to get such a com-

* Received Dec. 18, 1916.

plete and masterful treatise on this subject, and Mr. Wolff is preëminently qualified to deal with it. The Oliver Mining Co. has a director of explorations of long experience, and he has ever maintained a competent staff to study in detail and correlate drilling exploration data, and the individuals have always been alert to recognize the value of and accept new suggestions and information given by others engaged in similar work. The application of these data in the construction of sections and interpretation of the Mesabi ore deposits by this company should be considered authoritative.

Mr. Wolff, a mining engineer, preaches a strong lesson—one of great interest to me as a geologist. In his opening paragraph he says (1) that during only the last 4 or 5 years has much been added to the detailed geologic knowledge of the district, and (2), that refinement in his work has become mandatory and a commercial necessity. It is noteworthy, then, that this district has now been a shipper for 25 seasons; that the Mesabi is well known to all men in the iron business; that the geology has generally been deemed too simple to afford a sufficient basis for consideration in the operation and development of a mine; that the geology has been well known for a long time (a comprehensive *U. S. Geologic Survey Monograph* on the geology of the Mesabi district was contributed by C. K. Leith in 1904, and in 1911 another monograph by Van Hise and Leith covered thoroughly the geology of the entire Lake Superior region, and in addition Leith has contributed much in special papers, and otherwise, on the geology and the ores of the Mesabi); and now, after these many years of activity, study, and publicity, the greatest and most influential operator in that district strongly indorses the fact that no matter how simple the geology is and how great the district is, geologic features, no matter how minute or inconspicuous, must be sought and taken into account and geologic principles must be applied in the work. Many have heard too often the old miner's classic remark that "the ore is where it is," but Mr. Wolff says that there is a reason for the ore being where it is but that the structural conditions governing are not readily discernible. Nothing additional to the subject can be advanced, and it is needless to comment further on the details presented other than to emphasize the remarkable regularity of the subdivisions of the iron-bearing formation throughout such great distances, the importance of this fact, and the value that could be derived from similar observations made in other iron-mining districts.

Mr. Wolff treads on less familiar ground when he discusses the Mesozoic fossils and questions the classification of the remnants of the Cretaceous formation. Even if other data were lacking, nothing that he says is a bar to correlating the fossils found as of Cretaceous age. Upper Cretaceous fossils are rather abundant in widely scattered places in the glacial drift in Minnesota, but never have I heard of Jurassic

fossils in Minnesota, not even in the drift. Only a short time ago I had occasion to discuss this personally with Dr. F. W. Sardeson, formerly professor of paleontology in the University of Minnesota, who years ago identified such a collection of teeth and shells sent him from the Coleraine district. Dr. Sardeson said that he visited the place as recently as one year ago and that these fossils are undoubtedly Cretaceous fossils and clearly belonged near the Niobrara. Such a fossil as Mr. Wolff shows (his Fig. 14) Dr. Sardeson says would come from the Colorado group of the Upper Cretaceous rather than from the Jurassic, and that the coarse-ribbed, senile appearance which Mr. Wolff uses to argue for a Jurassic age, indicates Cretaceous, and quoted as examples other coarse-ribbed types of the Cretaceous such as *Schloenbachia leonensis*, *Prionotropites woolgari*, *Mortoniceros shoshonense* (Mick), and *Prionotropites loevianus* (White).

I do not know of any Jurassic deposits in the northwestern part of Minnesota, nor is the Jurassic considered generally to have been present. I know of nothing that would indicate that the interior Jurassic sea ever got farther east than the east flank of the Black Hills in South Dakota and the extreme southwest corner of North Dakota, which is a long way from the western end of the Mesabi Range. This interior sea was up to that time but slowly encroaching eastwardly and not until the later part of the Cretaceous period was this interior sea extensive, and there is no record that even with so extensive a sea were "extensively developed shales" laid down in northwestern Minnesota, as stated by Mr. Wolff.

Mr. Wolff concludes by presenting some facts which, as he says, may be of some value to geologists who are engaged in the revision of the correlation of the Huronian series. He is timely in his suggestion because recently a notable attempt has been made to change the correlation of the Huronian groups in the entire Lake Superior region, and there seems to be some foundation for it. So far as the Mesabi is concerned, it affected only the nomenclature of the Upper Huronian group, which was to be changed to Middle Huronian. But when it is a question of separating the Virginia slate from the Biwabik iron-bearing formation by an unconformity, if cognizance is taken of the fact that slate formations are built up under vastly different conditions, deep-sea deposits on the one hand and delta deposits on the other, and if the more recent theory of the deposition of the Mesabi iron-bearing formation is well understood, it seems that the conditions cited by Mr. Wolff need not call for a stratigraphic break to explain them.

EDWIN J. COLLINS, Duluth, Minn. (communication to the Secretary*).
—This paper by Mr. Wolff is particularly valuable because it contains

* Received Jan. 20, 1917.

so completely the actual observations of the author extending over a period of many years.

As an engineer of the Oliver Iron Mining Co., Mr. Wolff has had access to all the mines of the Mesabi Range, on which that company operates about 40 properties. The Oliver company and the other operating companies on the Range are very liberal in giving each other information of mutual interest and permitting inspection of their mines.

The observations in his paper cover 110 or more operating properties and many others now inactive. The mines are located along about 80 miles of the present producing portion of the Range. As a result of his keen observations in the mines and the study of the records of several hundred miles of drill holes, he has correlated the structural formation of all portions of the Range and shows that although the detail of the different orebodies varies, the structural features are the same.

The engineers of all Mesabi properties should work out the structure of their respective mines following the broad classification made by the author, so that ultimately there will be a uniform detailed structural record of all the orebodies of this great iron-ore district.

The information contained in Mr. Wolff's paper has been obtained by laborious observations and comparisons. It is of much economic value and anyone interested in mining on the Mesabi Range is indebted to him for publishing the information that has taken so long to acquire.

Geology and Ore Deposits of Mohave County, Arizona

Discussion of the paper of FRANK C. SCHRADER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1935 to 1967.

J. DANA SPERR, Jerome, Ariz. (communication to the Secretary*).—Very little accurate information has been published about this district. Most of the geological data appearing in the technical press are based on "careful observations" made from an auto stage and a casual glance through the *U. S. Geological Survey Bulletins*, No. 340 of 1908 or No. 397 of 1909, both by F. C. Schrader. Mr. Schrader's report is still the best publication on the district. A really careful study of this report will uncover only a very few statements which have since proven questionable. This is a remarkable record when it is considered that only a short time was spent by Mr. Schrader in the field and that there was very little development work done at that time.

The mining world and especially Mohave County, Ariz., will feel a sense of gratitude to Mr. Schrader for revising his early report on this

* Received Jan. 17, 1917.

possible exception of the Big Jim, not a single commercial oreshoot is known to exist in the andesites which is not intimately associated with a latite. (In this district I prefer to use the name "latite" to classify any rock too acid to be an andesite and too basic to be a rhyolite. It is generally impossible to classify any of the intermediate rocks in the locality from hand specimens or even by the microscope. Mr. Schrader remarks that some of the rocks which he calls "green chloritic andesite" may be latite, so, what I refer to as a latite may, in some instances, be the equivalent of the "green chloritic andesite.")

It is impossible to classify positively the different rocks in the field, and nearly so with a microscope, owing to the extreme alteration which they have undergone. As engineer for the Tom Reed I had some 50 specimens examined by C. F. Tolman, Jr., of Stanford University and succeeded in getting three or four definite classifications. However, it is generally possible to map the different flows, at least over local areas.

There is in the district an andesite younger than the vein system, which is a little more basic than the biotite or chloritic andesite and resembles, from the descriptions, the older andesite. This later andesite is found as a capping to a depth of from 200 to 300 ft. on the Black Eagle claim of the Tom Reed. It probably belongs to the undifferentiated series mentioned by Mr. Schrader. It accounts for the impression, frequently gained, that the older andesite is sometimes younger than the green chloritic andesite. I have thought it possible that the occurrence of an andesite under the green chloritic andesite described by Mr. Schrader might possibly be due to a faulting which had placed this still later andesite in a relatively lower position over local areas. This possibility might be supported by the fault, which I am informed exists, between the two andesites described as found at the Leland.

It must be noted that the physical appearance of different parts of the same flow may vary considerably. This fact, I believe, has led to some very erroneous conclusions. My notes show that the foot wall of the United Eastern mine is a nearly normal biotite andesite from where the shaft first penetrated the ledge to at least 600 ft. in depth, although the physical appearance of the rock at 200 ft. and at 500 ft. is decidedly different. Probably the same condition has existed in other places where the ore is reported to be found within the older andesite.

I cannot admit the intrusive character of the andesite of the Tom Reed, as the bottom levels show a nearly horizontal contact between the andesite and a basic rock which may possibly be an earlier flow or a remnant of the Paleozoic sediments. But the latite on the hanging wall is almost certainly an intrusive. This latite is green and chloritic, but the microscope shows it to be a nearly normal biotite latite.

Mr. Schrader notes that the gangue was primarily mainly calcite and dolomitic carbonates, but these minerals have been largely replaced

by quartz and adularia. I have observed that the greater the amount of quartz after spar, as a rule, the richer the ore. Lower levels observed show a constantly decreasing amount of quartz with uniform decrease in the gold content. At depth the vein filling seems to grade into a pure spar. Bottoms of well-developed oreshoots show very little replacement. There is no questioning the continuance of the main fissures to greater depths than will ever be opened up. The real question is whether there will be a limit to the barren calcite zone and the conditions favorable to replacement again exist.

As to the vertical horizons, the richest ore ever mined in the district was found between 200 and 300 ft. in depth.

At the bottom of p. 1957, Mr. Schrader outlines a general rule to guide the operator in search of ore, with which I am heartily in accord.

If a given vein or vein system in the district shows the quartz pseudomorphic structures, etc., mentioned, its development is an excellent gamble. But so far the calcite veins have proven too uncertain to be very safe.

A condition frequently observed is the occurrence of basalt plugs, often found on the strike of the main fissures. These plugs break up the continuity of the veins, sometimes reversing the pitch and causing other freaks. Some of these plugs come to surface and may even spread as a capping over a considerable area. Others do not reach the surface. Basalts seem to be good things to stay away from.

The accompanying cross-section through the Tom Reed zone may be of interest.

It will be noted that the main fissure is the line of a strong fault. While the amount of throw of this fault is not known, it is probably about 900 ft. The physical character of the rib of andesite on the hanging wall of the ledge at the 900-ft. level is very similar to that of the foot wall at or near the surface. No further evidence is known to the writer as to the amount of the throw.

The fault plane marked through the vein filling is a post mineral slip, very regular and distinctly traceable. I recall the first time I observed this slip. A raise had been driven carrying this slip as a foot and as I climbed through the raise I thought it had been concreted, the wall was so smooth and even. Deep vertical striations are the only irregularities.

The latite shows an intense crushing action wherever opened. In early work it generally appeared that the latite was the line of a main fault, but finding andesite on the hanging wall opposite a lower rock on the foot rather disproves this theory.

The andesite found on the hanging wall of the latite does not appear to be the same andesite within which the ledge occurs. It may be a glassy phase of the biotite andesite, but probably it is a more basic variety.

Angular inclusions of the andesite have been found within the latite

50 ft. from the actual contact. This seems to be characteristic of the latite andesite contacts in this zone.

Microscopic examination showed a resorption by remelting of the phenocrysts within the border of the andesite on the foot of the latite intrusion.

All of the rocks show an intense hydrothermal action. Practically all of the specimens collected were too highly altered for accurate determination. All but the latites show high oxidation. Pyrite in fine cubes is the strongest characteristic of this latite.

The drawback to mining is the high cost of prospecting, but when it is remembered that one United Eastern will pay 6 per cent. on 50 or more well-financed prospects and one Tom Reed will do over twice as well, the gambling chances will not seem so discouraging.

J. B. PLATTS, Hawthorne, Nev. (communication to the Secretary*).—I will confine my remarks to the Oatman district and to two points in which my ideas are in opposition to those of Mr. Schrader.

First, regarding the older andesite (or earlier andesite), I note that Schrader admits that this information has greater importance in the district than he allotted to it in his former report on the district. Schrader was the first to describe the older andesite, and it is for him to say what it shall include, but in my work I have taken it to include everything between the Paleozoic and the base of the green chloritic andesite. Between these limits may be found not only the rock described by Schrader near the Vivian mine, but a number of variations from this type. The rock is always light gray in color and is nearly always in the form of consolidated tuff, or breccia and tuff, but sometimes appears as hard uniform lava flows. I was able to trace this formation on the surface from a point near the Ivanhoe mine to a point several miles south of the Black Range mine, a distance of about 8 miles. In some places, notably near the Gold Range shaft, the beds of tuffaceous andesite are intercalated with thin beds of limestone. The older andesite may often be distinguished from the overlying formations by the difference in dip. They dip from 15° to 20° eastward and the upper lavas dip from 4° to 5° eastward. The upper surface of the older andesite forms a wavy contact with the overlying andesites, indicating a former rolling surface in an advanced stage of erosion. This indicates a long time interval between the eruption of the older andesite and the succeeding eruptions. It would seem that there was first a series of island volcanoes followed by quiet and elevation into continuous land. After a period of erosion and tilting, volcanic activity broke out afresh in the eruption of the green chloritic andesite and after a short period of quiet, the latites. After another short period of quiet the rhyolites were intruded in the form of thick dikes and stocks.

* Received Jan. 2, 1917.

The second point has to do with the genesis of the orebodies. Schrader regards the profitable ore as the result of secondary enrichment, and says, on p. 1956: "The gold was probably precipitated in large part along with the manganese dioxide." Statements similar to this have been made by a number of writers. So far the only tenable hypothesis to explain secondary enrichments in gold deposits is that proposed by W. H. Emmons. His idea is that manganese dioxide acts on chlorides, setting free chlorine which acts on any gold present, forming gold chlorides. These being soluble in cold acid water, are carried away and precipitated on the first reducing agent or alkaline substance encountered. It is evident that the gold deposits will not be found with the manganese dioxide but with the precipitating agent. Miners working in gold veins containing rich bunches in the oxidized zone are familiar with the idea that the gold will be found in the iron-stained quartz and that the black manganese stains indicate barren spots. It is evident from the chemical nature of calcite that gold chlorides cannot form in its presence, and that it can only act as a gangue for secondary gold when it is absent from the greater part of the vein that held the primary gold, both from the vein matter and from the walls.

It is unnecessary to enlarge on the application of the above principles to Oatman veins. There the wall rock is so calcitic that it will effervesce with weak acids, as limestone will. The gangue of the largest veins, including the most important so far found, such as the United Eastern, The Big Jim, and the Aztec, contains rather more calcite than quartz. It is, therefore, evident that any secondary gold deposits must be extremely localized and relatively unimportant.

The rhyolitic intrusions seem to be a more probable source of the gold-bearing quartz than the latites. Gold and silica dissolved with alkaline sulphides could have been driven off from them through cross fissures and the more permeable andesite beds, passing outward and upward until they encountered andesite dikes in the case of the prominent siliceous outcrop veins, and calcite bodies in the case of the three large mines previously mentioned. Incidentally, the finding of these large non-outcropping orebodies does not favor Schrader's idea that the siliceous outcrops are a guide to the finding of profitable ore. Also, a number of cases might be cited where prominent outcrops have proved to be the tops of unprofitable veins.

Problems Connected with the Recovery of Petroleum from Unconsolidated Sands

Discussion of the paper of WILLIAM H. KOBBS, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2253 to 2276.

ARTHUR KNAPP, Ardmore, Pa. (communication to the Secretary*).—On p. 2275 I find a quotation from Mr. Thompson regarding the fountains of the Baku field. I have seen a large number of these gushers or fountains. In every case where unusual amounts of sand were produced the wells caved in and shut off in a few days and were an absolute loss. Very often the formation had caved so far up as to allow the water to enter the oil strata, and not only was the well lost but the ground for many acres around the well was useless for drilling purposes because the disturbed ground had allowed the gas and oil to dissipate and the water to enter.

On p. 2268, Mr. Thompson is again quoted regarding the Balakhany field. If the text is consulted further, it will be found that the oil at Balakhany is heavy only in contradistinction to the Bibi Eibat field. The gravity of the oil at Balakhany is from 28 to 32. Also, it must be noted that the illustration is taken from the conditions in the older part of the field. This field was first drilled 40 years ago, and was a worn-out field 15 years ago. The wells which produce sand are wells in which the casings are practically gone and the formations are full of water. The sand must be bailed from these wells because the clays from the upper stratum have caved into the wells and are obstructing the passage of oil.

On p. 2262 is the statement that the Russian wells cannot be pumped on account of the large amounts of free sand produced. They have never tried pumping in the Baku field to any extent. Wherever tried, it has proven a great success.

If we take the author's theories as correct and wish to put them into practice, it would first be necessary to consider a few figures.

Let us assume that we produce oil and sand together in the proportion in which they exist in the productive strata.

It is not uncommon to have wells in the California field which are not gushers, but which, by pumping, produce 1,000 bbl. per day the first 100 days.

Taking the porosity as one-third, there will be drained during the 100 days, 1,680,000 cu. ft. of sand. If the sand were removed we would have to dispose of:

$$62,400 \text{ cu. yd.} = 168,000,000 \text{ lb.} = 84,000 \text{ tons} = 1,000 \text{ car loads.}$$

* Received Jan. 15, 1917.

To haul this sand away in wagons would require 312 wagon loads per 24 hr.

Further, assuming that the oil was dead and not assisted by any gas, it would require five or six times more power to raise the sand and oil together than the oil alone.

The author, after expounding the theory that the best way to extract oil efficiently is by producing a large quantity of sand with the oil and separating the oil at the surface, then says that the best way to extract the sand is by pumping it along with the oil. Of the total volume produced by pumping, not more than 4 or 5 per cent. can possibly be sand. The oil must have a velocity great enough to carry the sand along with it through the pump and up the tubing. Each particle must be separate or they will soon gather together in such quantities as to overcome the velocity of the oil and will settle and stop the pump. Wells producing 500 to 600 bbl. of oil daily and only a few cubic feet of sand will "sand up" the pump unless great care is taken.

I. N. KNAPP, Ardmore, Pa. (communication to the Secretary*).—The author of this paper makes a most remarkable statement concerning a basic principle of oil recovery, as follows:

"Other things being equal, the maximum recovery of oil from an unconsolidated sand is directly dependent upon the maximum recovery of the sand itself."

Unfortunately, the author fails to say what the other things are that must be equal, and gives no intimation of what percentage of the total oil may be considered a maximum recovery when the work is done according to his basic principles. He proposes to extract unconsolidated oil sands from depths of 3,000 to 4,000 ft. and rely for the effectiveness of such operation on the uncontrolled caving of the unconsolidated oil sands at these great depths.

This proposition is contrary to the fundamental precepts of good mining, and can easily be shown to be impracticable.

The paper mentions oil sands of 280 to 300 ft. in thickness; 100 per cent. extraction of these sands would cover the property about 280 to 300 ft. deep. It is self-evident that an extraction of 30 per cent. would present a costly problem in caring for the sand.

The author says, regarding the disposition of sand brought to the surface: "It was loaded upon wagons and distributed on the many dirt roads traversing the property." This indicates an extraction of only a fraction of 1 per cent. of the total sand. Also, after the sand and oil is brought to the surface for maximum oil recovery the author says, "This sand as it comes from the sumps contains sufficient oil to stain the hands and serves as a binding material for the road surface. It soon

* Received Jan. 30, 1917.

packs into a durable layer of asphaltum-like hardness." It is, therefore, evident that a maximum recovery of oil from that portion of sand actually brought to the surface is far from 100 per cent.

The problems connected with the recovery of petroleum from any kind of sand cannot, in my opinion, be properly discussed without taking into consideration the porosity of oil-bearing sands, and this the paper under discussion fails to do.

The porosity of sands, sandstones, etc., has been very elaborately investigated by Professors Slichter and King, and the results are given in the form of professional papers appearing in the *19th Annual Report of the U. S. Geological Survey*, Part II, 1899. Also *Water Supply and Irrigation Papers* Nos. 67 and 140.

The maximum pore space or reservoir capacity of a theoretical sand was found to be 47.67 per cent. and the minimum 25.95 per cent. with a mean of about 36 per cent.

Experiment proved that the sands that occur in nature come well within these limits, that fine sands contain more pore space than coarse ones; also, that the fine sands yield their liquid contents at a slower rate than the coarser ones.

From the data given, it is fair to assume that the porosity or reservoir capacity of the so-called unconsolidated oil sands is one-third. Therefore, every foot in depth of saturated oil sand of the kind mentioned would yield theoretically 4 in. in depth of oil. This figures out about 2,500 bbl. per acre. If a 10-in. hole a foot deep is drilled in such a sand, 125 sq. in. of pore opening would be uncovered to feed 78½ sq. in. of well opening, and the walls of 10 ft. of well would open a pore space 16 times the area of the 10-in. well.

These figures show conclusively that the extraction of oil alone will soon form in the aggregate a large space, and that it is not necessary to remove any sand with the oil to get a space of low density around the well casing and screen, to which oil may flow. Space produced in this way will in no wise affect the stability of the sand or well, and is in accord with good mining practice. Also, the aggregate of the vast number of small pores feeding through the sand to a well are of sufficient capacity to supply gas and oil for the flow of the greatest gushers and this without any movement in the sand itself if the well is properly screened. For setting well screens and cementing in well casing using the rotary method of drilling, I would refer to my papers¹ treating of these matters in detail.

In a field where the depths of formation are known, a well can be drilled by any suitable method to near the sand. It is then entirely feasible to drill in with the rotary using the mud control and have a full-sized well to the very bottom.

¹ Cementing Oil and Gas Wells, *Trans.*, (1914), 48, 651. The Use of Mud-Laden Water in Drilling Wells, *Trans.*, (1915), 51, 571.

A screen of the full size of the hole may then be set to cover the unconsolidated oil sand, then the oil string of casing may be set above the screen and cemented in. By attaching suitable appliances, the well is ready to be bailed and brought in under absolute control as to the outflow of oil and gas.

If a proper mesh of screen has been employed, the extraction of the oil when properly controlled cannot induce caving. The proper screening of a well reduces pump troubles to a minimum, always assuming that the well may become a pumper.

This entire prevention of caving is in accordance with the fundamentals of good mining practice.

Indiscriminate caving around the well casing must endanger if not ultimately destroy the well, and this destruction may happen before any but a fractional percentage of the sand and oil is produced and it must inevitably happen long before any but a small percentage of the total sand is extracted, and not only destroy the well itself but put a considerable area surrounding out of commission so far as profitable drilling is concerned.

Regarding the caving of sand, the author says: "This movement is frequently of such volume, or occurs so suddenly, that the casing penetrating the reservoir is damaged or completely severed," and again "No type of casing is proof against this force of moving sand."

The necessity of following the fundamental precepts of good mining practice in extracting oil from oil sands below the surface is evident from what the author himself has said, which evidence discredits the statement of the basic principle and law set forth in the paper.

That a cavity or opening of any considerable size can be formed in unconsolidated material by extracting sand with the oil from a well at depth of 3,000 to 4,000 ft., as assumed by the author, is, in my opinion, open to question. The weight of the superincumbent strata must be sufficient to cause much of such material to flow toward any point where the equilibrium has been destroyed by the removal of solids and thus tend to prevent the formation of a cavity. It is well known that many soft clays, slates and shales do flow and soon close up any opening made in them at comparatively shallow depths. It is in some cases impossible to hold even a small tunnel open with any kind of timbering without continual cutting back of the flowing ground until two, three or more times the volume of the tunnel is removed, after which the flow will gradually cease. I have worked in ground of this character and seen it squeeze and flow.

I once had the experience of drilling a well with the rotary to about 1,800 ft. in depth in unconsolidated formations and into a horizon where gas was known to exist in great volume. After setting 40 ft. of fine ($\frac{1}{100}$ in.) wire-wound screen and completing the well, and allowing time

for the cement to set, the mud was bailed down, the well cleaned itself and came in like a good gasser, giving dry clean gas with a little dust of fine sand. After blowing a short time, say 20 to 30 min., thin bits of clay came out, that were evidently squeezed through the screen. The screen gradually closed up until the flow of gas was nearly gone. The inside of the screen pipe was cleared of clay and a 20-lb. shot of 40 per cent. dynamite exploded in it. The explosion was followed by a flow of gas, sand shale and clay. Possibly as much as 5 cu. yd. of solid material came out in 10 or 15 min. The well cleared itself and measured around 2,000,000 cu. ft. open flow of clean gas, and showed when closed over 600 lb. gas pressure. The gas from this well was used to start another hole, but it soon played out and clay was found to have squeezed up the casing for about 40 ft. above the break.

This column of clay in the casing could be reduced to within about 10 ft. of the break, when the clay would begin to squeeze in faster than it could be extracted. The well was then filled with mud and cleaned out past the shot and, on again bailing, it partly cleaned itself and then went to salt water and shut itself off. This shows that it is practically impossible to form cavities in unconsolidated materials at great depths in the manner assumed in the paper.

Unconsolidated sands are known to occupy about 10 per cent. less space than in the original bed when they are broken down into loose sand. This can mean but one thing, that the sand grains have moved closer together and consequently the reservoir and seepage capacity is lessened. From this it is fair to assume that an unconsolidated oil sand held in place in the well by proper screening can be as well or better drained of its liquid contents than caved or loose sands left in the well, or even when brought to the surface.

The Influence of the Movement of Shales on the Area of Oil Production.

Discussion of the paper of RICHARD A. CONKLING, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1917, pp. 1969 to 1972.

D. W. OHERN, Oklahoma, Okla. (communication to the Secretary*).
 —The author states: "A shale layer buried beneath 2,000 or 3,000 ft. of strata, in some instances, will upon folding become thicker in the synclines and thinner on top of the anticlines." In any series of folded strata there is a level of no strain. Provided the strata do not slip on each other, there would be above this level a thickening of strata in the synclines and a thinning on the anticlines, as the author states, while the converse would obtain below this level. In order that the author's

* Received Jan. 2, 1917.

position be tenable, it would, in my judgment, be necessary to show that in the series of folded strata of the area under discussion the level of no strain was below the Bartlesville sand. In order even to approximate this level one should know both the thickness of strata removed by erosion since the time of folding and also the depth to which folding extends. Neither of these factors can be obtained.

There is considerable evidence in favor of the view that there is an unconformity below the Bartlesville sand in this region. Might it not therefore be that the variation in the thickness of the shale interval just below the Bartlesville sand is due to erosion? Personally, I am inclined to think this is a partial explanation in this particular case. On the other hand, the more rapid dip of the deeper strata seems to be the rule on structures in the Mid-Continent field. Mr. Conkling's observations on these sands are valuable, but one must wish for further data to substantiate the proposed explanation of their relations.

The Viscosity of Blast-Furnace Slag

Discussion of the paper of A. L. FEILD, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 307 to 332.

A. W. FAHRENWALD, Socorro, N. M. (communication to the Secretary*).—When asked to discuss Mr. Feild's paper, I felt most highly complimented to have the privilege of commenting on such a worthy and needed piece of apparatus.

In consideration of the importance of the viscosity of blast-furnace slags and the relation existing between the melting point of a slag and its viscosity, the introduction of this apparatus should be a welcome means or assistant in the study of the physical properties of a slag. Up to the present time, it is a fact that this side of the practical working of the blast furnace has been sadly neglected by those investigating slags. It is easily seen that the fluidity of a slag is of as great importance as its melting point. It is not necessarily true that a slag having a low melting point will have a correspondingly low viscosity or be highly fluid. For example, a slag having a melting point of $1,100^{\circ}\text{C}$. may have to be superheated to, say, $1,300^{\circ}\text{C}$., in order to have as high a fluidity as another slag whose melting point is, say, $1,250^{\circ}\text{C}$. The latter in this case would be the more economical. This point has been brought out very nicely by Mr. Feild.

Before discussing this paper further, I think it will be well to outline some of the points, a discussion of which may be in order:

* Received Jan. 30, 1917.

1. Effect of variation in rate of speed of rotating crucible.
2. Effect of eccentricity of crucibles.
3. The depth of slag and relative position of surface must be the same in every case.
4. Effect of slag and high temperature upon crucible and spindle.
5. The effect of slip.
6. The volume of slag that can be maintained at uniform temperature is relatively small.
7. Irregularities in viscosity curve may or may not be due to mechanical or observational errors.
8. Reduction of iron by graphite crucibles which is greatly assisted by rotation of crucible.
9. Furnace difficulties.
10. Practicability of the apparatus.

Some of the points outlined above have been carefully considered in Mr. Feild's paper.

The effect due to variation in speed of the rotating crucible would be very small, owing to the slow rate of rotation of the rotating crucible as compared with the speed of an electric motor, especially if it be of the constant speed type and be supplied with uniform current.

Considerable error would undoubtedly result in case the relative position of surface of slag was not the same in all tests, owing to the difference in specific gravity of slags, depending upon the constituents. The same quantity by weight would not give equal volume, and hence would cause a noticeable variation in viscosity in the case of a series where one substance is replacing another. Would the error thus caused be sufficient to be taken into consideration?

All slags do not have a similar effect upon Acheson graphite. In my work, along the same line, I have noted particularly that highly basic slags have a greater tendency to adhere to the graphite on cooling and in some cases they are removed from the crucible with difficulty. Acid slags do not show this tendency. This characteristic would probably have a direct bearing upon "slip," therefore it would be expected that the effect or error due to slip would be more noticeable in determining the viscosity of highly acid slags than in basic ones, on account of its lesser "wetting" ability. It is probable that if any slip is present it could be entirely eliminated if the viscosity spindle and crucible were finely grooved, longitudinally, and especially if the crucible wall was farther removed, say at least 1 in. all around.

I cannot feel thoroughly convinced that the irregularities in the viscosity curve are due entirely to mechanical and observational errors, as Mr. Feild states, but think that possibly they are due to an actual critical point of some kind. So little is known regarding equilibrium between mineral constituents at high temperatures that I think Mr. Feild is

hardly justified in ascribing fluctuations in his curves to error and not to some shifting of equilibrium.

For example, if a refractory mineral compound is present, it is known that the slag will consist of eutectics, or solid solutions, surrounding these crystals of compounds. On heating, the viscosity would perhaps not change much until the compound was all dissolved, when there would be a sharp brake in the curve. On cooling, when the temperature was reached where this compound commenced to separate out, there would begin a new rate of viscosity increase. There are many possibilities for different viewpoints, and in the absence of experimental facts these fluctuations in the viscosity curve cannot be satisfactorily accounted for. It is too bad that Mr. Feild did not take a few cooling curves on some of the samples described, along with the temperature-fluidity curves, as this would have thrown much light on the problem.

The intimacy of contact between slag and graphite parts, also reducing atmosphere in the furnace, causes much iron to be reduced, which, in the case of highly ferruginous slags, I find is considerable. It is also greater when a slag in mechanical mixture is heated and formed in the crucible than it is for formed slags, as from a blast furnace. The error caused by this separating out of metallic iron is so great that I have found it necessary to first form my slags (those having melting point below $1,300^{\circ}\text{C.}$) in a nickel or steel crucible, then subject them to the viscosimeter. While Acheson graphite is an admirable material, it has this disadvantage, it causes boiling in the crucible due to the liberation of oxygen, an objection that cannot be overcome until a more suitable material presents itself.

I notice that Mr. Feild has experienced the same difficulty with the furnace, in connection with the viscosity apparatus, that I have, and that is the cracking of the inner and outer tubes, between which is the resistor (carbon, crystal, etc.). Of course the cracking of the outer tube is not so serious, but the inner must not be deformed very much because local heating will result. F. A. Fahrenwald,¹ Cleveland, Ohio, tells me that the life of the tube can be increased 1,000 per cent. if plastered with alundum cement, and that when they crack he cements them up with the material.

This cracking is undoubtedly due to the unevenness with which the carbon resistor heats up, as alundum tubes do not crack if wound with platinum wire for a platinum wire resistance furnace. For my work on viscosity, in connection with the apparatus, the furnace is not the granular type but one (the principle of which I am holding a secret yet) which is more rapid and in which any temperature can be attained in less than 1 hr. This furnace gives greater heat area with less bulk to the

¹ Personal communication.

heating unit, no inner tube is required, hence the rotating crucible could be large. This heating unit will last many times longer than the granular resistor.

Mr. Feild's viscosimeter certainly gives complete data and the viscosity of a given slag, but I question the practicability of the apparatus. Could this apparatus be used right at the blast furnace to work up a very fluid slag, or would the complexity of the apparatus and the skill required to operate it prevent its practical application?

I am using in my laboratory an improved form of the apparatus described by W. Mc. A. Johnson.² He uses a gas furnace which is a rather awkward arrangement for heating slag and measuring the temperature at which it drips through an elongated orifice of given diameter. I use an Acheson graphite crucible with an elongated orifice of known diameter and length, in an especially designed electric furnace in which temperature as high as 2,000° C. can be readily obtained. I superheat to say, 1,600° C., remove a graphite plug and allow slag to drip, for about ½ min., then insert plug again, allow slag to cool through 50° C. and take another drip sample for ½ min., and so on until the slag refuses to drip. From the weight of these samples I produce a viscosity curve similar to Mr. Feild's. The details and conditions for this method I hope to describe later.

In concluding, I would say that the apparatus described by Mr. Feild is a clever device and will no doubt find an open and hospitable field.

ROBERT B. SOSMAN, Washington, D. C. (communication to the Secretary*).—The author has made an excellent beginning on the difficult problem of the fluidity of molten silicates. From the standpoint of research on the pure silicates, it is to be regretted that he did not make some attempt to correlate the fluidity with the chemical composition, at least to the extent of recalculating the analyses to a molecular basis so that we can see about what is the relation among the important oxides in each of the slags examined. The next step, now that a satisfactory method and apparatus have been developed, would be to study a series of slags made up of pure oxides, preferably beginning with mixtures of only two or three oxides, so that a strict correlation of composition and fluidity, as well as of temperature and fluidity, can be made. The practical man is usually eager to have the properties of his own complicated mixtures determined at once, but unfortunately is not always able to use the results when obtained, for the simple reason that the variables are too numerous; they must be evaluated *one at a time*.

² A New Apparatus to Determine the Melting Points of Slags, *Electrochemical and Metallurgical Industry*, (1906), 4, 262.

* Received Nov. 30, 1916.

The author has overlooked in his review one important contribution of the character indicated. It is a paper by E. Greiner, on the dependence of the viscosity in silicate melts upon their chemical composition.¹ Greiner's method is not, I believe, as good as that used by Mr. Feild, but the results are of some comparative value. More measurements on the same plan would yield results of considerably more value, in the long run, than several times as many measurements on complex slags.

G. A. RANKIN, Creighton, Pa. (communication to the Secretary*).—In the introduction to this valuable paper, Mr. Feild has said, "The relation of high temperature to the viscosity of blast-furnace slags, aside from being of considerable theoretical interest and value, has an important bearing on the metallurgy of iron." This is true, but it would appear that one is justified in making much broader assertions than are contained in this statement; namely, that this investigation has an important bearing on the metallurgy of iron largely for the reason that it is of theoretical interest and value. In other words, for the reason that it is based on sound theory.

In so many of these so-called practical investigations, not only slag investigations but all investigations involving high temperatures, data are obtained by "hit-and-miss" methods of experiment with but little regard for many of the factors concerned. For that reason it is a real joy to read of Mr. Feild's work on the viscosity of slags, since in this study he has not only devised a very ingenious and practical piece of apparatus for viscosity measurements at high temperatures but has also shown due regard for all theoretical factors involved.

In regard to the viscosity apparatus as a whole, it would appear to one who has had no experience with apparatus of this nature that there is little room left for improvement. But if one considers simply the electric furnace employed, a platinum-wire furnace would undoubtedly be found more satisfactory than the granular carbon furnace. The advantages of the platinum over the granular carbon furnace are: (1) It is possible to obtain more uniform temperatures; (2) the furnace atmosphere can be more readily controlled. The one big drawback to a platinum furnace is, of course, the expense involved.

Concerning the preliminary viscosity measurements which Mr. Feild has made, it may be said that they are excellent as far as they go. It cannot be said, however, that sufficient data have been obtained to make possible any broad general statements showing just what effect each chemical component of the slag will have on the viscosity values at various temperatures. To obtain such data it would appear desirable

¹ *Inaug. Diss.*, Jena, 1907. Abstracted in *Chemical Abstracts* (May 20, 1908), 2, No. 10, 1381; (April 20, 1910), 4, No. 8, 1007.

* Received Dec. 17, 1916.

to make viscosity measurements of all possible mixtures of two or more of the principal chemical slag components. By proceeding in this way to the systematic determination of viscosity curves for all systems involved, it should be possible ultimately to state definitely the extent to which each component affects the viscosity of a slag. This would, of course, require the preparation of a large number of special slags and the accumulation of a large amount of data. It would not appear, however, that it will be a matter of any considerable difficulty to obtain these data, now that a satisfactory method has been developed for viscosity measurements at high temperature.

The data which it is thus possible to obtain with this method should be of great value to the iron and steel industry. Besides the determination of the viscosity of slags, this method is equally applicable to the study of the viscosity of glasses, a matter of extreme importance to glass manufacturers.

J. E. JOHNSON, JR., New York, N. Y. (communication to the Secretary*).—I have read with much interest Mr. Feild's paper describing his method of testing the viscosity of slags. This is a subject which is intimately associated with the economical and successful operation of blast furnaces, and while it has received attention in the past, very few of the investigations made have been of any value whatever because they determined softening points instead of free running temperatures, and as I have more than once pointed out, the interval between these two may vary from 50° or less to 200° or more, and as a variation of 200° in the free running temperature means the difference between economical and wasteful furnace operation, it is obvious that softening points are of little practical value.

Mr. Feild has apparently realized this and he has developed a method for determining the viscosity at any desired temperature, which examination indicates to be entirely practicable. The concordant results obtained confirm this conclusion.

It is not a simple matter to develop an apparatus which is reasonably free from experimental errors and which can be operated at temperatures approaching 3,000° F. for considerable periods of time, and I think that Mr. Feild is entitled to congratulation on the success of his efforts.

When it comes to the practical results obtained, however, I am afraid that the practical man will not obtain the benefit he hopes from such tests until they have covered a much wider range, and particularly until they are referred to an accepted standard.

For instance, if we assume that in Lake ore practice a normal slag is silica 33½ per cent., alumina 13½ per cent. and lime and magnesia 50 per cent., we shall not miss average conditions by any great amount.

* Received Dec. 5, 1916.

Furnace men know how a slag like this will act, what coke consumption it requires with a given percentage of ore, etc. The real use of viscosity determinations is to establish the physical nature of such a slag as this under definite conditions and use it as a datum from which to measure the viscosity of other slags.

For this purpose, I would suggest that slag temperatures be determined at a furnace running on a typical slag of this general order. My personal preference as to the method of taking it would be to use an optical pyrometer of approved type on the slag as it runs over the slag dam. A few days spent in visiting normal working furnaces and making such temperature determinations would serve to establish fairly definite limits within which the temperatures of the slag vary, for they *do* vary within the length of a single cast. Now if one of these actual slags were subjected to test by Mr. Feild's process, within this temperature range, we should know the viscosity under what might be called a standard temperature condition. What we then need to know is *not* the change in viscosity at the *same* temperature which is brought about by a variation in the slag composition, but the variation in *temperature* to obtain the same viscosity. This temperature is what I have called for a good many years the critical temperature of the blast furnace. Upon it the whole thermal operation of the furnace depends.

With variations in the viscosity, on the other hand, we are not much concerned because we do not care to have the cinder thinner than a certain standard for a given purpose, while if it is thicker or stiffer than is required to meet the physical condition of flowing out through the cinder notch, the furnace will not operate and we are not interested in measuring this unusual viscosity but in avoiding it. For the latter purpose the only means available is to increase the temperature, which means increasing the fuel consumption in much greater degree.

There is only a small range of viscosity for each slag through which temperatures need be determined. This will reduce the number of determinations required and facilitate the obtaining of information which would be of great practical value to furnace men. Collateral determinations of the effect of calcium sulphide up to, say, 3 per cent., should be made for a few slags of different composition to see whether this component has any effect on free running temperature.

To boil the whole matter down, therefore, if viscosity determinations are to be useful to the furnace men, they should be:

First, based on the physical characteristics of a definite slag.

Second, plotted with composition and viscosity as the independent variables, and temperature as the dependent one.

The Genesis of Asbestos and Asbestiform Minerals

Discussion of the paper of STEPHEN TABER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1917, pp. 1973 to 1998.

* JOHN C. BRANNER, Stanford University, Cal. (communication to the Secretary*).—Wideawake teachers of geology are constantly on the lookout for good illustrations of veins, especially where the processes of formation are either clearly shown or suggested. And I long ago found out that one who would learn nature's laws must despise nothing.

It thus happened that in 1910, Dr. George J. Peirce of the department of botany at Stanford University called to my attention a cracked and broken yellow earthenware dish, $8\frac{1}{2}$ in. across the top and $2\frac{1}{2}$ in. deep, that, to the ordinary observer, seemed to be of no further use, and quite ready for the waste can. The striking thing about this dish was that the thin glaze over much of the outside surface and over the upper margin of the inside surface had been lifted in small scales, varying in diameter from $\frac{1}{2}$ cm. to 1 cm., and thrust outward and held there by some sort of white fibrous mineral. The process looked very much like one we are familiar with in the Southern States where we often see in the winter pebbles and soil thrust out from a clay bank or lifted from a clay soil and held there supported by what is popularly known as needle-ice.

Upon inquiry I learned the following facts in regard to the history of this particular dish: In the summer of 1910, it had been partly filled with a concentrated salt solution taken from one of the salterns of the Stauffer Chemical Co. at Redwood City, Cal. A qualitative analysis of the original brine showed:

Present: iron, magnesium, sodium, potassium, chlorides, sulphates, nitrates (trace), bromides, carbonates, borates (trace), and considerable amounts of organic matter.

Absent: calcium, lithium and iodides.

The solution in the dish was a saturated one made of the unpurified commercial salt just as it is shoveled out of the salterns at Redwood. This salt was dissolved in distilled water to which a little more than $\frac{1}{2}$ per cent. of agar-agar had been added. Inasmuch as more of the agar-agar had been added than was intended or desired, the solution was left standing undisturbed in the dish on a windowsill in one of the botanical laboratories at Stanford University from the time the preparation was made in July, 1910, until Nov. 20, 1910, when the contents were found to have completely evaporated and the dish to have been shattered and

* Received Jan. 24, 1917.



ASBESTIFORM MINERAL FORMED ON THE OUTSIDE OF A GLAZED EARTHENWARE DISH OF POROUS MATERIAL. THE MINERAL CRYSTALLIZED BENEATH THE GLAZE AND LIFTED IT OFF IN SCALES. THE UPPER PHOTOGRAPH SHOWS, ENLARGED, THE FRAGMENT BROKEN FROM THE LIP OF THE DISH.

chipped off by the crystallization of the salt in process of evaporation, as may be seen in one of the accompanying photographs.

Furthermore, it is evident that the nature of the dish itself had much to do with what happened to it. The dish is salt glazed both inside and out, and the glaze, examined under a magnifying glass of low power, is seen to be cracked in angular mosaic-like blocks about 1 mm., in diameter. The body of the dish, where it is broken, is 5 mm. thick, is of a light cream color, and is *porous*.

The accompanying photographs give some idea of what happened. The solution inside of the dish penetrated the cracks of the inner glaze and the porosity of the body of the dish permitted the solution to pass through it, and to evaporate through and from the cracks of the outer glaze. Evaporation of the water caused the crystallization of the minerals in solution beneath the outer glaze, and the formation of the crystals lifted off the scales, as they grew at their inner ends, just as needle-ice thrusts outward the earth or stones beneath which it starts to form.

The minerals are all needle like and parallel—in other words, they are asbestiform. The length of these asbestiform minerals varies considerably; the longest noted are $1\frac{1}{2}$ mm. in length. Somewhat longer crystals were broken off by the handling of the dish before it came to my attention.

It is worthy of note that chipped pieces were formed on the bottom of the dish, all over the outside of it, and on the inside of it above the level at which the fluid formerly stood. A thick scum was evidently formed over the surface of the solution shortly after it was made, and this seems to have prevented evaporation below the level of that protective covering inside of the dish.

In the photographs some of the crystals seem to have no chips of the glaze on their summits. This is due to their having been broken off in handling. In some cases where the glazed chips do not show I have removed the crystals and have found them springing from shallow bare pits, showing that they too had started beneath the glazed chips.

The upper photograph shown in the accompanying plate is of the piece broken from the nick seen in the photograph of the dish. The cracks about this fragment were filled with the white asbestiform mineral, but it is quite possible that these were incipient cracks that have simply been enlarged by the deposition of mineral matter.

Starting from the left edge of the nick in the dish and 1 in. (25.4 mm.) below its lip, another crack runs nearly halfway round the vessel and finally ends on the lip itself. This crack is now filled with the white asbestiform mineral. I suspect that this is also an incipient crack that has been enlarged by the deposition of the mineral.

The specimen is preserved in the department of geology at Stanford University, California.

The application of the theories of Dr. Taber is regard to the origin of asbestiform mineral veins seems to explain satisfactorily all the phenomena shown in this interesting illustration.

In conclusion, I beg to express my high appreciation of the great value of the contributions of Dr. Taber to the subject of vein formations. Such a paper as his is a great credit to the American Institute of Mining Engineers, and it should remind our young men especially that the solutions of some of our greatest problems in geology lie within their easy reach if only they will put themselves in that scientific and receptive attitude of mind which enables us to sit humbly at the feet of nature.

JOHN A. DRESSER, Montreal, Canada (communication to the Secretary*).—The features of Mr. Taber's paper from which I shall dissent are questions of fact rather than of theory.

Regarding the origin of chrysotile veins, Mr. Taber points out that if they were formed by fibers growing outward from a central fracture the fibers would tend to grow unequally in length and thus give the veins irregular boundaries against the massive serpentine wall rock, which he implies is not the case. In this Mr. Taber's observations of the occurrences of Southern Quebec at least are clearly in error. Microscopic evidence shows abundantly that the characteristic boundary is finely irregular and might be described as a minutely jagged line. It is scarcely, if at all, less irregular than the line of boundary between the serpentine wall rock and the adjacent peridotite. This characteristic, of course, affords strong evidence of the growth of the vein at the expense of the wall rock.

Inclusions of massive serpentine in the central partings of chrysotile veins are not infrequent. I have never observed that they are characteristically angular, or that any occur in other parts of the vein. They might be found in "poor" veins in which the chrysotilization is far from complete.

The question of "mass fiber" chrysotile, which Mr. Taber says does not exist, may be only one of definition. The serpentine of the East Broughton mines is largely fibrous. For instance, in some of the mines 90 per cent. or more of the rock is passed through the concentrating mills. Perhaps 10 to 15 per cent. of this amount is recovered and the balance rejected because the fiber is too short. But the rejected dump material differs from that saved only in the length of fiber. In places chrysotile at the Broughton mines appears as slip fiber, and occasionally veins are found, but the great mass of the product of these mines is derived from rock which on crushing and screening separates into fibrous particles. For such material it is difficult to find a more applicable term than "mass fiber," and there seems to be no obvious reason why the term should be restricted to anthophyllite.

* Received Dec. 2, 1916.

R. P. D. GRAHAM, Montreal, Canada (communication to the Secretary*).—For some time past I have been interested in the mineralogy of the Black Lake-Thetford area in the Province of Quebec, and during the past summer completed a short memoir, now in the press,† on the origin of the serpentine and chrysotile which are so abundant in that locality. I was, therefore, especially interested to read Mr. Taber's paper on the genesis of asbestos and asbestiform minerals, and gladly avail myself of the invitation to discuss one feature of it.

While it is possible that Mr. Taber's interesting experiments in the laboratory production of fibrous crystals may throw some light on the mode of genesis of certain fibrous minerals, I do not consider that he is justified in drawing the conclusions he does from these experiments regarding the manner in which chrysotile veins increase in width during their formation; the conditions in the laboratory experiments and in the field are totally different in at least one very important particular.

Mr. Taber writes that he was successful in producing fibrous crystals only with substances which go into solution with decrease in volume (*i.e.*, crystallize with expansion in volume). If a supersaturated solution of such a substance enters the pores of a vessel (such as the cups used in the experiments, and there crystallizes, it would naturally tend to rupture the vessel and produce cracks; any further crystallization of solution entering such cracks or fissures, either directly or through the pores, must obviously widen them by "pushing apart the inclosing walls." I do not wish to discuss here the possible reasons for the fibrous habit of the crystals or for their transverse attitude to the cracks; these features may be due to fresh supplies of material reaching the crystals only at their extremities, they may be the result of a differential pressure exerted upon the growing crystals owing to the tendency of the cracks to open, or there may be other and more complex causes. So far as the widening of the veins is concerned, however, the important point seems to be that in Mr. Taber's experiment the inclosing walls are not in any way chemically related to the solutions which bathe them, and are not capable of reacting with them. The walls, therefore, are quite inert and they are necessarily permanent; once cracks are formed, there is only one way in which they can possibly widen, and that is by the receding of their walls from one another.

Turning now to the chrysotile veins, Mr. Taber is in agreement with the view that "the alteration of a rock to serpentine begins along the existing fractures which divide the rock mass into blocks of variable size" (p. 1987) and further, that this alteration and the formation of chrysotile veins are contemporaneous processes (p. 1984). It would seem to follow that at the very outset serpentinization should be most complete in the layer of rock immediately adjacent to the fracture; here also the pressure

* Received Dec. 2, 1917.

† *Economic Geology*, (1917) 12, No. 2.

developed (due to the reaction) should be at a maximum, and consequently the serpentine formed most readily soluble. If the pressure were not relieved, the serpentine would remain in solution until it reached the necessary concentration, when it would be precipitated; on the other hand, if the pressure were relieved, the separation would take place from solutions of lower concentration. In either case, the original inclosing wall has been destroyed and the serpentinizing waters proceed to attack and destroy successive new layers or zones of rock further and further removed from the original fracture.

I agree with Mr. Taber that the determining factors responsible for the fibrous habit, parallelism and transverse attitude of the serpentine in the veins, have probably been (1) crystallization under differential pressure (due to the tendency of the fractures to open) and (2) the supply of material at one extremity only of the crystals (that furthest removed from the original fracture), and I further agree with his view that the veins have increased in width during growth by the recession of the inclosing walls. I am of opinion, however, that this growth has taken place at the expense of the walls, which, as serpentinization with its contemporaneous chrysolitization progressed, have receded because they were continually destroyed, and not because they were pushed apart by the growing fibers.

While the above refers more particularly to serpentinization along pre-existing fractures, the same argument would apply to the minor chrysotile veins, which are sometimes regarded as following subsidiary strain fractures produced through the expansion which attended the alteration. It seems reasonable to presume that the waters which were responsible for the serpentinization would find their way to such cracks directly from the main fractures rather than by the necessarily very much slower passage through pores in the massive rock.

Two further points in Mr. Taber's paper may be referred to, although they are not very material to his theory. On p. 1986 he states that the pressure developed as a result of the alteration of peridotite, etc., to serpentine "cannot be explained by attributing it to a chemical reaction taking place with increase of volume, as, for example, when plaster of Paris combines with water and sets." The two reactions, however, are entirely similar, so far as the nature of the volume changes involved are concerned.

Finally, the writer has reason to believe that, at least in the Black Lake-Thetford occurrence, chrysotile and ordinary "massive" serpentine have the same specific gravity, and not the values 2.50 to 2.65 and 2.219 respectively, as usually stated in the textbooks.

GEORGE P. MERRILL, Washington, D. C. (communication to the secretary*).—That the subject of Professor Taber's paper is one of

* Received Jan. 5, 1917.

interest to me has been made apparent by his numerous quotations from my writings of several years ago. Referring to my paper of 1895, which related primarily to amphibolic forms, I will say only that the idea he now expresses, to the effect that the asbestiform structure is usually due to the accentuation of a normal prismatic habit and cleavage through the limitation of crystal growth by physical conditions, is essentially the view put forward by myself in the paper he quotes, and I have as yet seen no good reason for changing this view. What I have to say today, however, relates more particularly to the asbestiform serpentines, and this mainly for the reason that Professor Taber seems to have overlooked my paper of 1905 in the *Bulletin of the Geological Society of America*.¹ Perhaps I should state first of all in this connection that my knowledge of the field relations of this form of the mineral is limited to but a few hours each at the Thetford, Canada, and Montville, N. J., localities, and the views which I then put forward I regarded as more in the nature of suggestions than as theories or final conclusions. It is, therefore, with a feeling of some satisfaction that I have noticed even a tendency toward agreement on the part of the several workers, including Professor Taber, who have since written. My paper had in view, however, the explanation of the formation of vein cavities as well as their filling, and as it is here that our views become divergent, I shall have to ask the privilege of quoting briefly from myself, as follows:

"The writer's own opinion, founded on the facts at present available, is that the crevices are due to shrinkage such as is incidental to the change of a highly hydrated colloidal substance into a less hydrated and more solid form, and perhaps also to a loss of silica."

While this was acknowledged as not wholly satisfactory, it seemed to me, and still seems, as free from objections as any that have been since advanced. Cirkel, the well-known Canadian authority, for instance, in 1910 wrote in his report to the Canadian Department of Mines: "The most rational explanation, and the one that seems to gain most support, is that the formation of cracks was caused by cooling and shrinkage of the rock masses, similar to the formation of cracks caused by shrinkage of a gelatinous mass of iron carbonate in the so-called serpentine nodules of clay ironstone, as suggested by Merrill." It is also, he added, probable that the introduction of the granitic dikes so frequently met with in the serpentine masses has caused or facilitated to a great extent the formation of numerous fissures in the immediate proximity of these intrusions by rapid dehydration due to the agency of heat. The same view was accepted in part by Dr. A. P. Lowe and others. Dr. Dresser, however, as shown by the quotations of Professor Taber, was inclined to differ, and, in part at least, for the reason that he was able to point out veins at least 100 ft. in length, which he could not conceive it possible could have re-

¹ (1905), 16, 131-136.

mained as open fissures while filling was taking place. This latter idea is one that I had not taken the trouble to elaborate, since the fact that veins starting as mere lines may increase in width during the process of filling has become long since a part of the general textbook knowledge on the subject of vein formation. I might add, however, that even a length of 100 ft. of open cavity is not impossible, since so far as has thus been determined the vein-bearing portions of the serpentinous rock do not extend to great depth. Indeed, Dresser states distinctly that in the Broughton area the serpentine containing asbestos occurs only in the upper portions of the sills. Further, no idea is given as to the width of the 100-ft. vein which he mentions. This would certainly have an important bearing upon its ability to withstand pressure and remain an open cavity.

With Professor Taber I shall have to differ very materially as to the origin of the vein cavities, inasmuch as I cannot accept his ideas as to the origin of serpentine, nor, wholly, those put forward by the Canadian geologist whom I have quoted. My own views on this part of the subject are still essentially the same as expressed in an article on the use of the terms *rock weathering*, *serpentinization* and *hydrometamorphism*, which appeared in the *London Geological Magazine* for August, 1899, and was subsequently reprinted in the *American Geologist* for the same year. I there stated the opinion that serpentinization is a deep-seated process due to waters or vapors coming from considerable depth, and possibly, in the case of igneous rocks, even constituents of the magmas at the time of their intrusion. I have yet to learn of a single instance in which the serpentinization of a rock mass has been found to be superficial or connected in any way for a certainty with meteoric waters, the action of which is almost invariably accompanied by oxidation. The views which I have since suggested as to the filling of the vein cavities, and which Professor Taber seems to accept in part, were to the effect that the mineral-bearing solutions permeated through the wall cavities, crystallization beginning at the outer walls, and as the fibers grew at the base, pushing out into the interior, sometimes from one wall and sometimes from both. The fragmental matter often found along an irregular longitudinal line in the interior of the vein, I considered particles of matter pushed off from the wall as crystallization proceeded, in the same manner as the fibrous gypsum in limestone,¹ and as the hoar frost in the moist ground. That as the crystals continued to grow in length there was a possibility of interference, I felt was shown by the crimpings where the fibers seemed to meet from the opposite walls of the vein. That this could not be due to any lateral movement of the walls seemed to me sufficiently evident from the fact that these were mere gash veins sometimes of such slight

¹ See my paper: On Formation of Stalactites and Gypsum Incrustations in Caves, *Proceedings of the U. S. National Museum*, (1894), 17, 77-81.

length that an appreciable movement was simply impossible. A feature which seems to have escaped the attention of Professor Taber and others who have argued that the cavities could not have been open cavities, is this: It is impossible to conceive of so many veins within a given area beginning as mere lines and with walls being gradually forced apart by the growing crystals without considering the compressive results that would be produced on the intervening massive serpentine portions. So far as I have observed, there is no evidence of any such crushing effect as this would call for, and, while I am not wholly committed to the open-fissure idea, it would seem to me that this phase of the subject should not be overlooked.

Not to carry the discussion too far, I will simply add that Professor Taber's conclusions, as given in his paper and the one since published in the *Proceedings of the National Academy of Sciences* (December, 1916), as to the origin of cross-fiber asbestos and the effect of "a normal prismatic habit and cleavage" on the amphibolic forms are with the exceptions mentioned substantially in agreement with those I have expressed in the papers above quoted, and as yet I see no reason for changing my views.

Blast-Furnace Gas

Discussion of the paper of K. HUESSENER, presented at the New York Meeting, February, 1916, and printed in *Bulletin* No. 110, February, 1916, pp. 443 to 474, and in *Transactions*, vol. 53, pp. 402 to 433, and of the paper of LINN BRADLEY, H. D. EGBERT and W. W. STRONG, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 209 to 224.

R. J. WYSON, So. Bethlehem, Pa. (communication to the Secretary*).
—At our plant (Bethlehem Steel Co.), we have had several Bradshaw burners under course of construction and erection for some time, but up to the present we have not had opportunity to put them in service. I regret that I can not take part in the discussion with some actual results from these burners. Mr. Huessener properly emphasizes the importance of obtaining the highest combustion temperature practicable in boiler practice, which demands a near approach to perfect combustion conditions. The ideal burner is the one which, while meeting the various mechanical requirements, fulfills the above condition under a wide range of gas pressure.

The question as to the relative value of hot raw gas and cold washed gas for boiler use is a very interesting one. I think that in the past the importance of low temperature of washed gas sometimes has been over-emphasized. Mr. Huessener has pointed out that so far as temperature is concerned, no advantage accrues in washing gas below 100° F. In

* Received Nov. 6, 1916.

many plants, there is a notable operating economy in using furnace cooling water, which is appreciably warmer than the supply water. However, in a given type of washer, other conditions being constant, the lower the exit gas temperature the lower will be the dust content; this is equivalent to saying that the more washing water is used, other conditions being constant, the lower will be the dust content of the exit gas. For each type of washer there is a certain desired average temperature difference between inlet water and exit gas, concomitant with known average dust and moisture contents, and with economical operation. With the present tendency toward smaller stove checkers, it is generally agreed that gas for stove use should be cleaned to a dust content of 0.20 grain or less.

The method employed and extent of gas cleaning are clearly problems for individual furnace plants. As a general proposition, wet washing is decidedly more attractive for furnaces smelting soft, fine Lake ores than for those reducing chiefly hard, lumpy ores, such as the foreign and domestic magnetites. In the former case the average moisture content of the gas is probably 35 grains per cubic foot, the dust content is relatively high and the top temperature relatively low. In our plant, the ore mixtures consist almost exclusively of hard magnetites. The average dust content of the gas leaving the dust catcher will average only about 2 grains and the moisture about 12 grains per cubic foot, corresponding to saturation at about 83° F. The temperature is fully 100° F. higher than in Western practice. Owing to the large percentage of gas consumed in gas engines and under boilers in our plant, gas economy is of prime importance. Clearly, dry cleaning under such circumstances is an attractive issue.

With the possible exception of the Halberger—Beth system, the only dry cleaning method of any promise for blast-furnace gas is the Cottrell process.

We installed a small experimental Cottrell electric dust precipitator at one of our furnaces over a year ago, but have had opportunity to begin tests only recently. While we are not yet in a position to publish any results, we have obtained some that are promising, and we are hopeful of the outcome. We have in mind the possibility of fine gas cleaning for gas engines, though at present we are working for primary cleaning only. I see no reason why this process, successful elsewhere, cannot be adapted to the peculiarities of blast-furnace conditions. If successful, the chief advantages which might be expected, in our case the maximum, would be as follows: First cost and operating cost should at least be no greater than with the present cheapest wet scrubbers, for equal dust elimination; conservation of about 10 B.t.u. of sensible heat per cubic foot; higher combustion temperature; probably lower total moisture content; conservation of more flue dust for sintering, and elimination of undesirable sludge basins.

Potash as a Byproduct from the Blast Furnace

Discussion of the paper of R. J. WYSOR, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 1 to 32.

CHARLES H. RICH, Conshohocken, Pa. (communication to the Secretary*).—Mr. WYSOR has certainly covered his subject in the most thorough and able manner and his paper will no doubt result in enlarged effort toward the conservation of this important byproduct. Not all furnace burdens are as rich in alkalies as those noted by Mr. WYSOR. In our own experience we have never gone into any extended examination of the materials entering the furnace burdens, but have made a number of determinations of the water-soluble alkalies of our flue and boiler dust and found maximum of about $4\frac{1}{2}$ per cent. K_2O —too small for profitable reclamation.

We have, however, frequently encountered the smoky gas described by Mr. WYSOR. Such gas carries heavy fume, and burns with a fitful flame difficult to maintain. So pronounced was the inhibitory effect of the fume that, in extreme cases, the flame of a lighted match was extinguished by the current of gas and it was found impossible to conduct calorimeter tests without preliminary washing of the gas. When the gas was led through a washer filled with distilled water and pieces of coke, the fume was absorbed and the gas burned without difficulty. Careful analyses conducted on the water solution from the washer showed cyanides in increasing amounts in proportion to the difficulty experienced in burning the gas. We note Mr. WYSOR's statement that he has never found evidence of cyanides in their gas, ascribing the difficulty of burning such gas to the presence of chlorides. Our determinations were conducted in such manner as to preclude possibility of interference by chlorine.

Some results obtained in a test are:

CO_2	12.00	8.50	9.50	9.20	per cent.
CO	26.00	29.00	29.20	29.60	per cent.
CO/CO_2	2.16	3.40	3.18	3.21	
H_2	3.60	4.20	4.00	4.00	per cent.
B.t.u.....	101.70	113.50	109.50	110.80	Anal.
B.t.u.....	100.60	114.10	114.10	111.30	Calorimeter.

It will be noted that the CO/CO_2 ratio of the gases is high. This seems to obtain in such gases to increasing extent in proportion to the difficulty of burning the gas. The gas represented by the three last tests could not be burned without preliminary washing. We calculated from our analyses of the water solution, about 1 grain of cyanides to 1 cu. ft. of gas.

* Received Jan. 29, 1917.

Whether the fume indicates the presence of finely divided chlorides or is due to cyanides, we agree with Mr. Wysor's conclusion that the inhibitory effect is mechanical and is due to the absence of intimate mixture of air and gas molecules. The high CO/CO_2 ratio observed in such gas probably has no bearing upon the observed phenomenon and is mentioned only incidentally.

Notes on the Heat Treatment of High-Speed Steel Tools

Discussion of the paper of A. E. BELLIS and T. W. HARDY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 61 to 68.

ROBERT J. ANDERSON, Cleveland, Ohio (communication to the Secretary*).—The paper by Messrs. Bellis and Hardy was interesting to me and has led me to make a few remarks concerning some of the points brought out.

The question of employing correct hardening temperatures is one that is sadly neglected, particularly in many large plants that forge and treat their machine tools for private use. As is most aptly remarked in this paper, "The custom of using one 'high-speed temperature' for all tools is very poor practice;" still, such practice is more common than otherwise. An instance came to my attention some time ago, in connection with some work with the standardization of high-speed steels for machine tools, where there were no less than six different brands of alloy steels being used for machine tools such as lathe, shaper, planer, boring mill and cutters of like nature. All of these different steels received identical heat and forging treatment—or at least that was the aim of the tool hardener. That the results were highly unsatisfactory goes without saying. That the hardening department of many plants is conducted on a rule-of-thumb fashion is the reason for a great deal of the trouble and disappointment which seems to be the lot of many tool-steel users. Exceedingly crude execution is the rule rather than the exception, and it is no small wonder that the results are as good as they sometimes happen to be.

It would seem that at the present price of alloy steels the subject should receive a closer scrutiny by purchasers than it does. One not familiar with the workings of some plants that disdain to lend an ear to scientific heat treatment would scarcely give credence to some of the methods employed, not only in the treatment of the steel but also in its selection and purchase.

Heating for hardening should be conducted under close pyrometric control only after the correct hardening temperature has been determined.

* Received Jan. 11, 1917.

However patent all this may be to most people, it is an exceedingly difficult matter to impress upon some, in spite of the facts of experimental evidence and those gleaned from the apparently trustworthy literature.

Relative to the matter of the decarbonizing effect of BaCl_2 ; potassium ferrocyanide is supposed to prevent decarbonization of cutting edges while heating. Just what reaction takes place here or how this happens is not known to me.



FIG. 1.— $\times 500$.

The photomicrographs show in clear-cut fashion the existence and position of the critical temperature of hardening and the importance of the same.

With the rapid progress in metallography, it is hoped that every plant either manufacturing or using high-speed steel in quantity will soon find the services of the metallographist invaluable.

F. C. LANGENBURG, Watertown, Mass. (communication to the Secretary*).—The authors of this paper have taken great care to state

* Received Jan. 30, 1917.

precisely the reagents used in etching the samples of steel described in this note. They have also stated definitely the time of etching. Anyone who has had much to do with the microstructure of high-speed steel will appreciate their care in covering this point, as it is one of the most important items in judging of the character of the structure produced by any treatment.

Several specimens have come under my observation which gave well-defined austenitic structure after a few seconds' etching, but after a longer



FIG. 2.— $\times 500$.

attack the specimens resembled very much those shown on Plate "D" where the steel was hardened at $2,150^{\circ}$ F., which is, evidently, not a desirable condition for a properly hardened high-speed tool.

To more clearly illustrate this point and to show also that the structure is a great help in determining the actual cutting efficiency, two photomicrographs are introduced. Both samples were etched 2 min. in a 4 per cent. alcoholic solution of HNO_3 and photographed at 500 diameters.

The specimens which these illustrations represent were taken from twist drills that had been subjected to a competitive test. The drill having the structure shown in Fig. 1 was far more efficient than the drill having the structure shown in Fig. 2. Fig. 1 certainly shows a structure more nearly approaching austenite than Fig. 2.

If these samples had been etched 15 min. in the same reagent as above, very little difference would be shown in their structure and one might be led to believe that microscopic examination was without value



FIG. 3.— $\times 500$.

in the examination of high-speed tool steel. Quantitative etching will, if properly carried out, be of very great assistance in forming an opinion concerning hardening temperatures and other high-speed steel problems.

The statement made by Mr. Bellis and Mr. Hardy, that the custom of using one high-speed temperature for all tools is very poor practice, is not clearly understood. If they refer to the use of one high-speed temperature for all steels it would seem more reasonable, and it is inferred that this is their meaning. Often it is not the custom to use the same

high-speed temperature on the same steel for all tools. Many hardeners are afraid that too high a temperature will cause cracking and checking in complicated cutters with sharp angles, and for that reason they use a lower temperature on such pieces. This practice very often arises from the fact that the management will criticise the tool hardener for cracking tools in the hardening operation, whereas a proper viewpoint would show that a few cracked tools brought about by the use of a higher hardening temperature would in the end result in great economy, as one properly hardened cutter will outlast many that are improperly hardened.

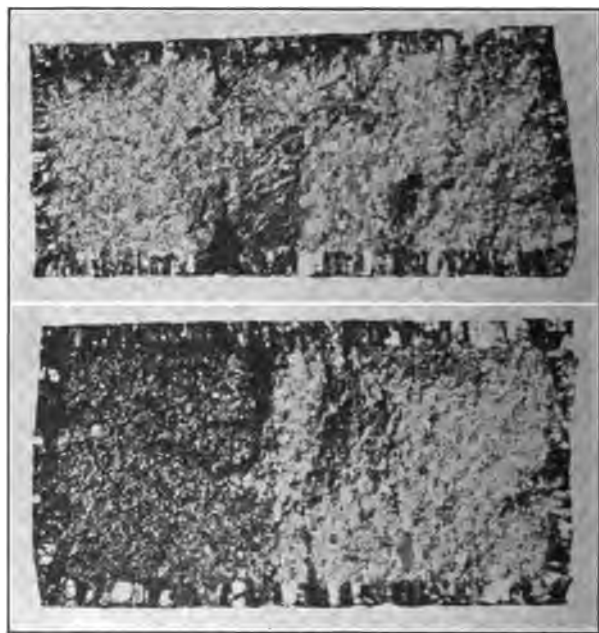


FIG. 4.— $\times 2\frac{1}{2}$.

It is not altogether clear what is meant by the term "burning" in their paper. There is introduced here a photomicrograph at 500 diameters, Fig. 3, which shows the appearance of an intergranular eutectic. This intergranular eutectic becomes evident when the hardening temperature has been too high. It is probable that the formation of this eutectic is brought about by heating above the solidus point, which may be separated by 100°C . or more from the liquidus. It does not seem that the eutectic formation can properly be called "burning," although its effect on the tool may be the same.

If the term "burning" is to be extended to cover all causes of intergranular brittleness, the metallurgist will lose at once a concise word with a definite meaning, and confusion will result. A number of over-

heated steels have come under my observation, but the intergranular brittleness was always traceable to the eutectic shown above.

One of the most important items in tool-hardening, and an end that is fully as difficult of attainment as the proper hardening temperature, is the hardening of a tool so that the surface is not affected in composition to a depth greater than a few thousandths of an inch. If the tool is hardened in a strongly oxidizing atmosphere, it is of course necessary to grind not only to remove the scale but to remove the soft skin which

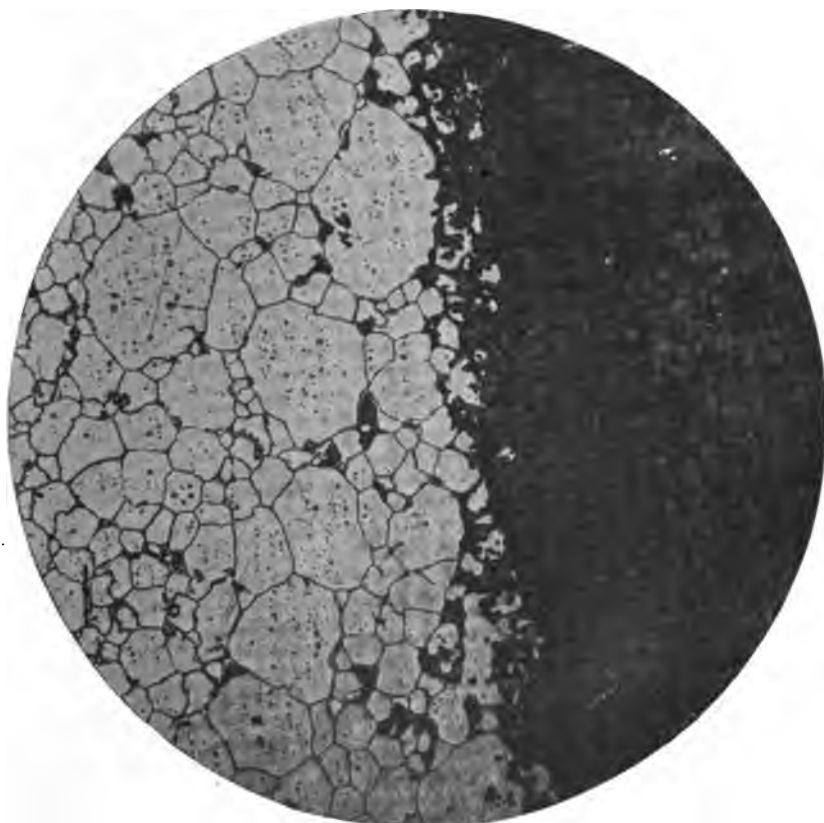


FIG. 5.— $\times 50$.

results from decarburizing. If, to avoid this result, a reducing atmosphere is used, serious troubles will very likely be present, but not so apparent. The tool may come from the furnace perfectly bright and be soft to a depth of $\frac{1}{16}$ in. Fig. 4 shows such a condition. When this tool was removed from the furnace it was perfectly bright and absolutely no trace of oxidation was present. Decarburization had taken place, however, to a very marked degree, as can be seen from an inspection of the edges of the tool.

The micrograph shown in Fig. 5, 50 diameters, was taken at the junction of the decarbonized edge and the inner portion of the tool not affected by reducing gases. Fig. 6, 50 diameters, shows the interior structure of the same tool. The specimen from which these photographs were taken was intentionally overheated to illustrate the coarsening of the grain and the appearance of the intergranular eutectic previously discussed.

This decarburization noted above is easy of explanation. The gases in the furnace were CO and CO₂, which will, if the temperature and

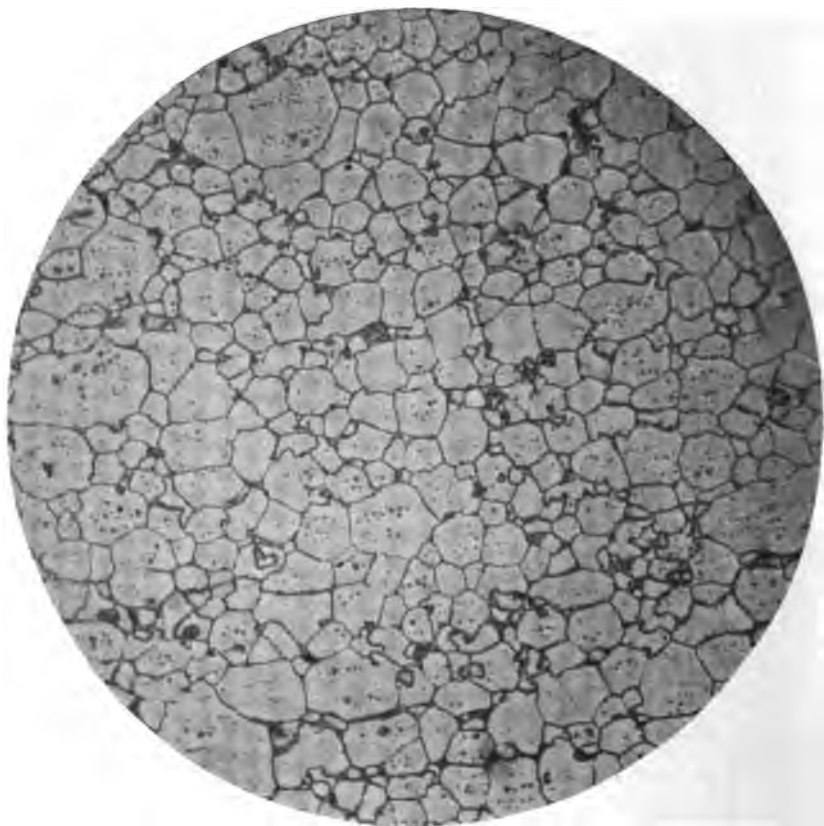


FIG. 6.— $\times 50$.

pressure are fixed, come to an equilibrium condition represented by $\frac{\text{CO}}{\text{CO}_2} = K$.

If a piece of steel is introduced into an atmosphere of CO and CO₂, which is under the equilibrium conditions, $\frac{\text{CO}}{\text{CO}_2} = K$, cementation or decarburization may occur. If the concentration of CO is greater than that for equilibrium, cementation will ensue, whereas if the concentration

is less, the result will be decarburization. At elevated temperatures a very small amount of CO_2 is sufficient to make $\frac{\text{CO}}{\text{CO}_2} < K$, which will limit cementation or cause decarburization.

The authors of the paper under discussion say that their hardening was done in a reducing atmosphere of a carbon resistance electric furnace. The principal advantage to be obtained from such a furnace is that the tool is not scaled, but this does not imply that serious surface softness may not result from this type of apparatus, as the illustration given above will show.

Pack hardening is never very satisfactory, due to the difficulty of adjusting the ratio $\frac{\text{CO}}{\text{CO}_2}$ so that neither decarburization nor carburization will ensue.

It is the opinion of the writer that a satisfactorily constructed oil or gas furnace will give good results if proper attention is given to the regulation of the air and fuel supply.

Two very important points have not been mentioned by the authors; namely, the effect of time at the hardening temperature and the subsequent tempering of hardening high-speed tools. There is a great deal of work to be done on both of these points, but it is evident that the time at the hardening temperature is of vital importance because the coarsening of the grain is a function of the time as is also very probably the solution of the complex carbides.

The question of subsequent tempering of high-speed steels is of relatively great importance in certain classes of work. In heavy cuts where the tool will be tempered by the heat generated in cutting, the necessity of a second treatment is not so important. On twist drills and a great deal of light work where the tool itself will not generate enough heat to cause tempering, the previous tempering operation might be of considerable advantage, as it has been shown conclusively by Edwards and Kikkawa,¹ that a decided increase in hardness results from tempering properly hardened high-speed steel. Whether this increased hardness will result in longer life is the question still open for investigation.

¹ C. A. Edwards and H. Kikkawa: *Journal of the Iron and Steel Institute*, (No. II, (1915), 92.

Erosion of Guns—The Hardening of the Surface

Discussion of the paper of HENRY FAY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2237 to 2251.

LAWRENCE ADDICKS, New York, N. Y., (communication to the Secretary*).—There are two points regarding gun erosion on which I want to say a few words. The first is about the analogy to hardening of wire by drawing and the second the possible rôle played by thermal conductivity of the metal.

The ductile metals do not necessarily become brittle when cold-worked as it is possible to draw copper from $\frac{1}{4}$ in. in diameter soft rod down to the finest hair wire without annealing. This results, as stated in the paper, in forming a hardened core with a very hard skin. The electrical conductivity is depressed some 3 per cent. and the tensile strength nearly tripled. A soft copper wire 0.08 in. in diameter and 6 in. long may be twisted 150 or more turns before the wire will snap under torsion. A hard-drawn wire will break after perhaps a dozen twists, the skin breaking down at one spot. Just wiping the wire with a cloth damp with dilute nitric acid has a remarkable effect in increasing the number of twists without appreciably lowering the tensile strength, showing the skin to be very thin. On the other hand, pickling the rod, from which the wire is to be drawn, in dilute nitric acid also greatly increases the number of twists which can be applied to the resulting wire. This indicates that the copper oxide scale produced when the wire bar is rolled hot is more or less rolled into the rod and may have much to do with the skin effect. We also know that if the conditions of drawing are not correct, the heat of deformation in the die may produce quite an annealing effect.

The application of this analogy to what occurs in a gun to produce the effects noted offers some difficulty.

The second point is that as the temperature of the surface of the gun bore has undoubtedly a great deal to do with the amount of erosion, one natural remedy would be to conduct the heat away fast enough to keep this temperature within desired limits. This brings up the question of the thermal conductivity of the metal of which the gun is made.

We know that the conductivity of pure metals is greatly altered by alloying and that the alloy is usually, though not always, lower in conductivity than the metal before alloying. This is interesting when it is considered that the high-nickel and manganese-steel alloys show greater erosion as stated.

Would it not be instructive to study the question from the point of view of heat conducted away from the eroded surface?

* Received Feb. 3, 1917.

Shot-Firing in Bituminous Mines

Discussion of the paper of M. D. COOPER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 123 to 133.

ARTHUR LAMOTTE, Wilmington, Del. (communication to the Secretary*).—On p. 124, Mr. Cooper makes the following statement: "For the breaking up of an occasional rock fall in an entry, straight 40 per cent. nitroglycerine dynamite was used."

I believe that this practice is one of considerable danger and would suggest that the use of nitroglycerine dynamite should not be allowed in gaseous and dusty mines, inasmuch as we and other manufacturers of



FIG. 1.

explosives make strong, quick-acting "permissible" explosives which are well suited for this purpose.

On p. 125, Mr. Cooper describes a battery used for firing the electric blasting caps, which consisted of two dry cells inclosed in a container, so that accidental contact was avoided. We have made a number of experiments with dry-cell batteries for blasting and have abandoned them all after exhaustive tests, as it is not possible, unless the blaster is equipped with rather expensive and delicate instruments, to determine whether the dry cell will develop enough current to fire the electric blasting caps.

* Received Jan. 26, 1917.

There have been so many instances of misfires due to their use that we have developed a blasting machine of much greater capacity and much smaller size, especially for use in coal mining, which is known as the Du Pont Pocket Blasting Machine (Fig. 1.) This will easily fire three or four electric blasting caps at one time, and is only about as large as the battery described in Mr. Cooper's paper.

Mr. Cooper mentions the method used in making a primer, and also calls attention to the probability of misfires being due to the detonating cap becoming detached from the cartridge. His method of making a primer is described as inserting the cap in the cartridge and wrapping the wires once around the cartridge to prevent the cap from being pulled out. This is not very clear, but it corresponds to the practice known as half-

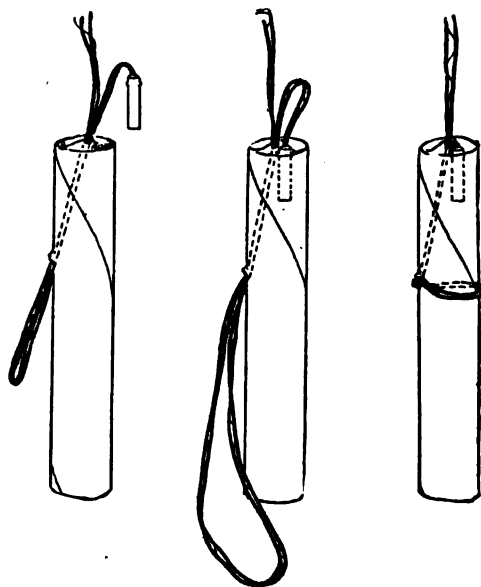


FIG. 2.—PRIMING METHOD FOR ELECTRIC BLASTING CAPS.

hitching, the wires being brought at right angles to each other and a tension applied to the ends forming a hitch and sharp bend in the wires and a possible short-circuit of electric current at that point.

In our catalogs, and in the directions of almost all manufactures of high explosives, it is recommended that wires of electric blasting caps should not be half-hitched around the cartridge on account of the possibility of short-circuiting or breaking the wires, with the attending danger of misfires.

A much better way of making a primer is shown in Fig. 2, which needs no description.

In Mr. Cooper's summary of the causes for misfires, the first one,

defective or exhausted battery, would be obviated by the use of the small-size blasting machine previously described.

The second cause, broken wire or connections on short-circuit due to imperfect insulation, would be largely done away with by correct priming methods.

The third case, defective detonating caps, is one which very rarely occurs in our method of manufacture, as each electric blasting cap is tested twice before leaving the factory.

The fourth, deteriorated explosive, can be guarded against only by using common-sense measures in storing and handling the explosive when received from the factory.

The fifth, detonating cap detached from cartridge, can be entirely eliminated by proper method of priming the dynamite cartridge with the electric blasting cap.

The method that Mr. Cooper mentions for destroying defective electric blasting caps—that of placing them on the haulage road and running a locomotive over them—is a rather dangerous one, as the electric blasting caps are likely to fly for a considerable distance on exploding and embed particles of copper in the flesh of persons standing even 30 or 40 ft. away. A better way to destroy these is to bury them in close contact with a good electric blasting cap, several inches under ground, and fire the good cap. This will detonate the others.

Report of the Secretary of the Committee on Safety and Sanitation

Discussion of the paper of E. MALBY SHIFF, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 87 to 122.

WALLACE MCKEEHAN, Douglas, Ariz. (communication to the Secretary*).—In going over this report, which I have done very carefully, I find that the summary as composed deals with the various problems almost entirely from the employer's standpoint and that practically no consideration is given to the employee's point of view.

An experience of $3\frac{1}{2}$ years as safety inspector, and almost 30 years as workman and official, has convinced me that any plan for the betterment of working conditions which will achieve any degree of success must give the workmen their share of responsibility and allow them full participation. The company may finance and direct the movement, but its ultimate success depends upon the support given by the workmen. The coöperation of the men doing the actual work is so essential that all writers insist that this must be secured, but they do not agree as to how this can best be done.

* Received Feb. 1, 1917.

I am convinced that the employer who, in starting an employees' welfare department, puts the proposition up to his men as being strictly a good business deal for both, will secure results. While he who prates of humanitarian motives, of doing all this for the good of the workmen and their families, at the same time using a paternal tone, will only cause the men to become suspicious. American workmen want no petting. All they ask is a fair deal all around.

Accident prevention and welfare work in general cannot be forced upon English-speaking employees, and the management who tries it will lose out; but a straight-forward, look-you-in-the-eye proposition, with all the cards on the table, will appeal to the men. Once they are convinced that they have an even break, they will sit in the game and raise your bid every time.

Too many welfare organizations are placed almost entirely in the hands of officials and bosses; especially are they in the majority upon the committees. This is not as it should be. In my opinion the workmen should constitute the majority upon all committees. If this is done, rarely will they abuse the privilege. Our experience would go to prove that workmen take more interest in accident prevention matters than the officials or bosses, and are quite quick to censure their fellows for infractions of rules. In proof of this I would state that of the many safety suggestion cards received by this department, only one was sent by a boss.

This department¹ receives many inquiries as to the best method of initiating and conducting a safety and welfare organization. The reply invariably sent is, "Do your part first; make all working conditions as safe, sanitary, and convenient as it is possible for you to do. After you have shown the men that your intentions are of the best, you are in a position where you can ask for their coöperation."

The work that can be done by companies engaged in mining, milling and metallurgical operations is, comparatively speaking, slight as regards the number of accidental injuries prevented, but the work done creates a good impression among the men.

While no fixed method can be advanced for the formation and conduct of an employees' welfare organization, if the coöperation of the workmen has been secured, the other problems may be quite easily worked out. All employees are naturally suspicious of the good intentions of their employers. One of the first things that the manager must do is to issue an order to all those in charge of operations, that safety must be the first consideration in the conduct of all work, and insist that this order be carried out to the letter. Most managers dislike to do this, because they fear it will interfere with the output. Such is not the case, however,

¹ Mr. McKeehan is Safety Inspector with the Copper Queen Consolidated Mining Co.

as safe working conditions make for greater efficiency. All officials and sub-bosses must be made to understand that unsafe conditions, methods and practices will not be permitted. A good way to secure their coöperation is the paying of a bonus for a reduction in accidents; also a bonus for keeping their respective places clean, orderly and sanitary; this last to be cared for by a monthly inspection, preferably by a committee of employees. The Copper Queen Co. pays its mine bosses a bonus of \$30 each. This is paid quarterly and is based on the least number of accidents per thousand shifts worked, ten bosses participating.

One thing that has caused the workmen to view with suspicion the formation of a welfare organization is that some companies have started the work with a great hurrah and then have let the movement quietly die. Although this practice has about died out, it will prove a severe handicap to future operations along welfare lines by these particular companies, to say nothing of its effect upon others.

The present tendency among safety inspectors is to have as few and as simple rules as possible. No rule should be allowed in the book that cannot be enforced. All rules should be couched in the terms and phraseology used by the workmen. For the above reasons, the rules committee should include a number of good practical workmen. There are so many good rule books now in use that the rules committee can select sound practical rules that will cover the operations of practically all departments. These can easily be changed or amended to suit local conditions.

Strict observance of all rules must be insisted upon. To secure this, some form of discipline will be necessary. Just how this matter will be handled should be left to the workmen's safety committees. With our organization, a man who breaks a safety rule is reprimanded for the first offense; given a layoff of from 1 to 15 days for the second, and discharged for the third.

Both men and bosses must have a thorough knowledge of all rules. This is often a difficult thing to obtain, for with a large employing company a personal examination cannot be made of each individual. One way in which the rules can be given wide publicity is to have all the rules covering a certain class of work printed on a card and to have this card placed at or nearby where this operation is being carried out. Rules for carmen and loaders should be tacked up at chutes; for motormen and swampers, at oil houses, sidings, skip pockets, and where these men stop for lunch. Miners on stope work should find the card nailed on timbers. Rules covering machine work, the handling of explosives, in fact all important rules, may be placed at points where the men congregate for lunch. This will be found to be a better plan than to depend entirely upon rule books. The books, however, should always be issued and the employee's receipt taken.

A classified accident record of all injuries, even the slightest, must be kept. This should be so arranged as to cover all operations in every department. This will have greater value if the report sheet used by the bosses is so arranged as to call for full details as to the occurrence of an accident. Information regarding slight, minor and no-loss-of-time accidents rarely reaches the inspector, except from the boss's reports. Injuries causing no loss of time should always be reported. These constitute about 40 per cent. of all injuries, and are the ones most likely to become infected, the men believing that they are not of sufficient importance for medical attention. With our company, all hurts, no matter how slight, must be reported. Failure to report will cause the employee to be discharged.

All reports of accidents should be absolutely impartial. This is very necessary if an inspection and report is made by the inspector or safety committeemen. Particular pains must be taken to fix the responsibility and the blame placed where it belongs. Under no circumstances shall the fact that the company, or one of its officials, was at fault be ignored. If the injured, or any of his fellows, was to blame, he should be censured. In case of a fatal or serious accident, photographs should be taken of the place immediately, if possible posing a fellow-workman to show how the accident occurred. These photographs are of considerable value when used in connection with a bulletin describing the accident, as they serve to fix the occurrence and nature of the accident more firmly in the minds of the workmen.

Live, snappy, up-to-date bulletins, if simply worded and using the familiar trade terms and phrases, illustrated by photographs taken on the ground, will have considerable value in calling the workmen's attention to safety matters. What is more, the men will read and discuss them with their fellows. Bulletins to be effective should be written by one who is acquainted with local conditions. Photographs of local scenes are the only ones that have any value. If workmen who are popular with their fellows are shown in the pictures, it will cause more interest to be taken in the bulletins. The common bulletins sent out by safety organizations to all parts of the country, printed on poor paper and with cheap illustrations, if any, have just about as much effect upon the men as would an advertisement calling for recruits to the National Guard.

Writers upon safety matters are laying considerable stress on the necessity for educating the workmen to protect themselves. This is all very good as far as it goes, but it doesn't go far enough. Bosses and officials need training along these lines as well as the men and should be the first to receive it. Bosses and officials can be given this instruction at weekly and monthly meetings, where lantern slides showing local conditions, methods, practices, etc., may be thrown on the screen, illustrating the talk made by the inspector or some one in authority.

New workmen can best secure this instruction when first they go to work. They should always be placed with an old experienced employee whose business it is to see that they are fully informed as to the dangers of their new work and as to how they should proceed to protect themselves. This may cost a little more at the start but will be found to provide a more efficient workman and will be cheaper in the end than compensation payments. The tendency among up-to-date safety organizations is to secure better training for the new recruit and this method will become universal just as soon as its value is thoroughly understood.

The education of the older employees is a far more complex problem than most people suppose. The workman, quite naturally, thinks he knows enough to care for himself, and being from Missouri, has to be shown. The one who is to do the showing must have had practical experience in the work and be one in whom the men have confidence. They also put a great deal of reliance in the recommendations of their own safety committee.

When it is possible to do so, all instruction should be given to the men when they are at work. They do not like having their leisure time taken for this purpose and will resent it if this is done frequently. If it is necessary to instruct them when they are off duty, it can best be done by giving them some form of entertainment, to which they can bring their families, and using a small portion of the time for an illustrated talk on accident prevention. A well-conducted bulletin board, with safety mottoes, rules, etc., posted near the working places, will do effective work in training the men to watch for unsafe conditions and in protecting themselves.

One thing that should be given more attention than it ordinarily receives, is that of selecting men for a particular kind of work; in other words, fitting the job to the man, not the man to the job. It is the common practice to put a man on a job; if he doesn't make good, discharge him. Yet this same man, if given a chance at something else, perhaps tried at several different kinds of work, will more than likely make good. If a man can be placed at work that he likes, he will be a more efficient, safer and better workman.

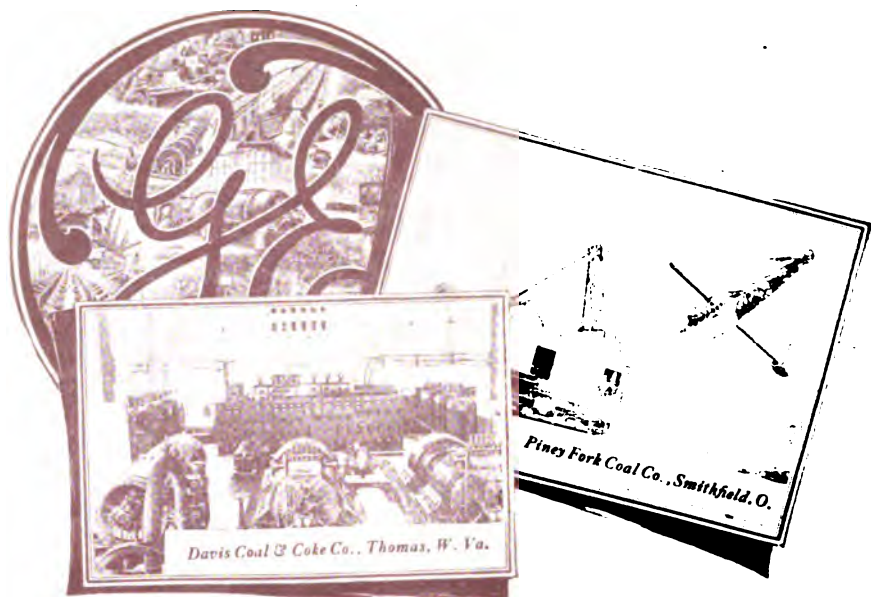
In this connection I would call attention to the indiscriminate discharge of workmen by foremen and bosses. I do not believe that any man in direct charge of workmen should have the right to discharge them, and I say this after many years spent in handling men. The practice is a relic of the dark ages and should be abolished. The trouble with the present system is that the workman holds his place subject to the whims and fancies of his boss, and if discharged, has no recourse. These days the education, training and shaping of the untrained man into an efficient, safe and reliable workman is done at a considerable cost, and the employer should not have this investment jeopardized by the sole judgment

of any one man. The man in charge should have the right to lay off a man, at the same time filing a complaint against him, but his final discharge should only take place after the workman's side of the matter has been heard. The discharge of a workman should be handled through the employment office, and after the case has been passed upon by a committee.

The large corporations, realizing the value of the trained worker, are taking steps to provide a better system of disciplining the men, also to see that the men get a hearing before being discharged. It is certainly disheartening to the leaders in a safety organization to find that men who have been trained in accident prevention, mine rescue, first aid, etc., are being discharged and forced to seek work in other localities.

Accident prevention is a good business for the employee, employer and the community, and if carried to its ultimate success, will pay substantial dividends to all, but the greater returns will be received by the employee, and deservedly.

If the safety campaign is extended to cover the betterment of conditions under which the employees work and live, it will create a better feeling between both parties and will do a great deal to eliminate some of the problems that are confronting capital and labor today.



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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 124

APRIL

1917

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY BROUGHTON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$12 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

VOLUME LIV

Volume LIV has been shipped to all members whose dues were paid on or before Feb. 10, 1917.

The next shipment will be made the latter part of April to all those whose dues were paid on April 1, 1917.

CLOSING DATE FOR MANUSCRIPTS FOR ST. LOUIS MEETING

In accordance with the usual custom, manuscripts to be presented at the next meeting of the Institute must be in the hands of the Secretary before July 1, 1917.

Manuscripts to be properly distributed in advance of the meeting must be in the hands of the Secretary before June 1, 1917.

At the last meeting of the Institute 45 papers were presented, and 42 of them were actively discussed. This experience makes us believe that advance distribution is the best possible means of getting a good discussion of papers and of securing for them the interest of the members of the profession. It will be impossible, however, to give the papers proper distribution if they are not received before June 1, 1917.

ST. LOUIS MEETING*Oct. 8-13, 1917*

The St. Louis Meeting of the Institute will be held Oct. 8-13, 1917. Various committees are being organized to perfect the arrangements for the meeting. A number of excellent technical papers have already been secured and an attractive series of excursions has been outlined.

The following is the outline plan for the meeting. More detailed announcements will be made as the plans mature.

MONDAY, OCT. 8

Morning: Registration and Technical Session.

Afternoon: Boat ride to Herculaneum, Mo., to visit smelter.
and

Evening: Dinner on boat and social evening.

TUESDAY, OCT. 9

Morning: Visits to plants in the vicinity of St. Louis.

Afternoon: Technical session.

Evening: Banquet.

WEDNESDAY, OCT. 10

Special train to Southeast lead district, spending the entire day.

THURSDAY, OCT. 11

Morning: Visits to plants in the vicinity of St. Louis and trip to Illinois coal field.

Afternoon: Technical session.

Evening: Leave on special train for Joplin.

FOR FRIDAY, OCT. 12 AND SATURDAY, OCT. 13

there will be offered a choice of three possibilities.

1. Two days in the Joplin-Miami district, including a visit to the strip-pit coal area at Pittsburg, Kansas, and a technical session on zinc.

2. Two days in the Tulsa area, including visits to oil pools, refineries and gasoline plant, and a technical session on oil.

3. One day in the Joplin district and one day in the Tulsa area.

**PROCEEDINGS OF THE MEETING OF THE BOARD OF
DIRECTORS, FEB. 20, 1917**

Joseph W. Richards was elected Vice-president and Director to fill the vacancy caused by the election of Philip N. Moore to the presidency.

Benjamin B. Thayer was elected Director to fill the vacancy caused by the election of Charles W. Goodale as Vice-President and Director. Sidney J. Jennings was elected First Vice-President of the Institute. George C. Stone was elected Treasurer of the Institute.

Rossiter W. Raymond was appointed Secretary Emeritus of the Institute.

Bradley Stoughton was elected Secretary of the Institute.

Burr A. Robinson was appointed Assistant Secretary of the Institute.

Upon the nomination of the President, the following Finance Committee was appointed:

GEORGE D. BARRON, *Chairman*
CHARLES F. RAND, BENJAMIN B. THAYER

Upon the nomination of the President, the following Executive Committee was appointed:

PHILIP N. MOORE, *Chairman*
GEORGE D. BARRON, EDWIN LUDLOW,
SIDNEY J. JENNINGS, L. D. RICKETTS

The action of the retiring Board of Directors in electing Herbert C. Hoover an Honorary Member of the Institute, was unanimously approved.

A vote of thanks to the Committees of the 114th meeting was passed.

The general proposition of the changes of the By-Laws of the United Engineering Society, intending especially the establishment of the Engineering Council, was approved provided the other Founder Societies approved.

Members, Associates and Junior Members were elected.

Five dropped members were reinstated.

The dues were suspended of two members engaged in active duty at the front.

A unanimous vote of thanks was passed to Dr. L. D. Ricketts for his services as President.

PROCEEDINGS OF THE ONE HUNDRED AND FOURTEENTH MEETING, NEW YORK CITY, FEBRUARY, 1917.

COMMITTEES

Committee on Arrangements

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P. E. BARBOUR
GEORGE D. BARRON
KARL EILERS
LOUIS D. HUNTOON
H. A. MEGRAW

THOMAS T. READ
BURR A. ROBINSON
F. T. RUBIDGE
E. MALTBY SHIPP
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MRS. BRADLEY STOUGHTON, *Secretary*

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LAWRENCE ADDICKS

*Registration Committee*BURR A. ROBINSON, *Chairman**Reunion Committee*H. A. MEGRAW, *Chairman*

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C. A. BURDICK
R. T. DANA
C. V. DREW

L. W. FRANCIS
P. L. GILL
H. W. HARDINGE
P. W. HENRY
L. D. HUNTOON
D. M. LIDDELL
E. P. MATHEWSON
C. P. PERIN

W. L. SAUNDERS

*Banquet Committee*E. B. STURGIS, *Chairman*F. T. RUBIDGE, *Acting Chairman*

ROGER L. WENSLEY

JAMES T. KEMP

*Luncheon Committee*E. MALTBY SHIPP, *Chairman*

C. A. BOHN

ALBERT D. BEERS

P. A. MOSMAN

ENTERTAINMENT OF LADIES

One of the most gratifying and pleasantest features of the meeting was the presence of approximately one hundred ladies on several occasions, such as luncheon and the other entertainments which had been provided for them, and which included luncheon at the Engineering Society's Building on Monday, Tuesday, and Wednesday, the visiting ladies being met as usual by members of the Ladies Committee in the Ladies Reception Room each day shortly before luncheon. On Monday afternoon Henry C. Frick, Esq., very courteously opened his art galleries at No. 2 East 71st Street, New York City, where was seen one of the finest private art collections in America, which was enjoyed not only by the ladies, but also by many of the members of the Institute. The visit to these art galleries was followed by tea and exhibition ice skating on the roof of the Waldorf Astoria Hotel. On Tuesday morning at 10.30 a. m., under the Chairmanship of Mrs. Sidney J. Jennings, an open forum on current events of the day was held. Mrs. Honnold and Mrs. Kellogg, of the Commission for Relief in Belgium, made addresses upon the work of

this Commission, and the ladies present thereupon organized a Women's Auxiliary of the American Institute of Mining Engineers. The aims and purpose of this Auxiliary are outlined in the address of Mrs. Jennings which given elsewhere in this *Bulletin*.

In the afternoon a visit was made by the ladies, guests, and members of the Institute to the art galleries of Senator William A. Clark, who not only opened his house for the purpose but also had an organ recital upon his magnificent pipe-organ and served refreshments to the visitors. Senator and Mrs. Clark received the guests. Wednesday afternoon, following the luncheon, the visiting ladies attended a matinee of the Yellow Jacket at the Harris Theater and thereafter visited a motion picture studio of the Famous Players Film Company and saw some pictures taken.

At the Annual Dinner on Tuesday evening, at the exhibition of motion pictures in colors on Wednesday evening, and at the all-day excursion to West Point on Thursday, the presence of the ladies was especially welcome and lent added attractiveness to these occasions.

SOCIAL FEATURES OF THE MEETING

Reunion Evening: On Monday evening, February 19, the entire fifth floor of the Engineering Society's Building was thrown open to the men members and guests of the Institute. Light refreshments were served, as well as souvenir boxes containing cigars and cigarettes. A pamphlet of songs of a popular nature containing the college songs of practically all of the universities represented at the meeting was kindly published for the occasion, by *Metallurgical and Chemical Engineering*. The meeting opened with the singing of "America." Moving pictures which had been loaned by the French National Committee, showing the interior of munition factories in France, and films showing French artillery in action, were shown. David H. Browne presided and read one of his own humorous and charming stories of Irish child life. Addresses were also made by E. P. Mathewson, Philip N. Moore and Sidney J. Jennings. Mr. Browne then called upon Dr. R. W. Raymond, Secretary Emeritus, who spoke as follows:

"I have been honored by appointment to the pleasant duty of introducing to this company of his professional colleagues Mr. Herbert C. Hoover. Of course, Mr. Hoover needs no introduction. It isn't necessary to look into "Who's Who" in order to find out who's Hoover! If you should consult that admirable volume, you would find an outline of what we well-informed old fellows knew about him a couple of years ago. We remembered him as a promising graduate of the Stanford University, a pupil of the distinguished Prof. J. C. Branner; then as an active and intelligent field-worker on State and Federal geological surveys; then as a mine-manager on the Pacific slope; then as the directing or advising engineer of great corporations, exploring and mining on all the continents, and conducting that "big business" which both requires and breeds big men. And we remembered him also as the author of masterly books on the principles and economics of mining and of a wonderful translation of Agricola's *De Re Metallica*, with footnotes which constitute a cyclopedia in themselves. But all this, that a few of us knew about him, is now swallowed up in the things that everybody knows!

"By special agreement with Mr. Hoover, neither he nor I will utilize this festive occasion by recommending the enterprise of international

philanthropy in which he now stands at the head. It does not seem appropriate to attempt, here and now, to touch either your hearts or your pockets in the name of such a cause, however holy. You have responded elsewhere already to that appeal. Besides all other contributions previously made through other channels, the whimsical corporation of "The Belgian Kiddies," very recently organized, has made a good beginning by receiving from the engineers of this country and forwarding to Mr. Hoover's Commission the sum of \$75,000. But, I repeat, we are not going to consider that subject to-night. We are here to greet a representative American mining engineer, and to acknowledge the glory which in that capacity he has shed upon us all.

"Let me, however, return for a moment to Hoover's translation of Agricola. That book was written in the sixteenth century in Latin—probably the worst Latin of history; for the author coined words of his own for the tools, machines, processes and officials for which classic literature contained no names. The German version, published a few years later, was made by a physician who knew nothing about mining, and is a mass of absurd misunderstandings. So old Agricola, one of the greatest champions of science in the battle against superstition, was deprived of his full fame by being buried in his own bad language! He could be exhumed and exhibited in his true proportions only by the skillful hands of some one who knew all kinds of Latin, medieval German and ancient mining and metallurgy besides. Let me say frankly that the more I study this admirable translation, the more I am inclined to give the chief credit to Mrs. Hoover, whose name, as well as that of her husband, stands upon the title-page!

"But Hoover had studied Agricola to some purpose. On the 93d page of that book, the old expert, writing four centuries before the passage of the eight-hour law, describes the division of mining labor, for purposes of continuous efficiency, into three daily shifts. And when in August, 1914, a hundred thousand American refugees, caught in the sudden catastrophe of war, invaded London as a frantic multitude, thirty American mining engineers, who happened to be residing in that city, organized themselves into a committee with Mr. Hoover as chairman, and, as the first step in systematic operation, constituted three shifts of eight hours each, thus becoming at once a body which could not be tired out, because two-thirds of it was always resting while the other third worked! So while government officials threw up their hands in despair, and well-meaning enthusiasts wore themselves to a frazzle in protracted over-exertion—"pouring themselves out" as a useless libation, instead of containing themselves for steady and tireless effort, this committee in three shifts went steadily on, giving to the world an object-lesson of knowing how and doing well—of the scientific conservation of human energy; of efficiency compared with—well, let us say, *shiftlessness*!

"For these American mining engineers were trained in big business. They knew how to handle both men and materials on the great scale; how to work together; how to keep cool; how to meet emergencies; how to climb over, or tunnel under, or go around obstacles.

"And we of the Institute know what kind of wives American mining engineers are apt to have—companions in enterprise and danger, equally fearless in camp or cabin—oh! you know the kind: you have one yourself!—and they were there too, making the splendid exhibition complete! We always believed we—we pairs, I mean—were the finest double blossoms developed by modern civilization; and now we are sure of it.

"I have promised not to speak of the later and wider work which naturally sought the same hands. But I cannot pass it by in utter silence; for above and through all our shouts and plaudits and self-congratulations over this great achievement of patriotic ardor expressed in scientific method and skill—above it all, and "smiting a silence" through it all, I hear a Voice, saying, "Ye have done it unto the least of these!"

"So I end as I began. You cannot introduce Hoover. You can only precede him—and you must not do that too long. Knowing that Hoover was coming after me, I have felt like the Irishman who was running wildly through a railway-cutting, pursued by a fast train, the engineer of which succeeded in stopping it just as he was about to be run over. When they said, 'Pat, why didn't you get off the track and up on the embankment,' he said 'Faith, I could only just kape ahead on a level; d'ye think I'd try it *up hill*!'

"Well, gentlemen, there is only one way to introduce Hoover. All of you get up, and lift high your glasses, and say with a will:

"Here's—Hoover!"

(Which they did.)

Mr. Hoover, rising to reply, received an ovation from his fellow-members which, for a considerable period, prevented him from speaking. When he could be heard, he modestly disclaimed the praise bestowed upon him personally, declaring that it was due to the splendid team-work of the thirty American engineers who formed the backbone of the Relief Administration and whose possession of the high professional ideals and the co-operative spirit fostered by the American Institute of Mining Engineers under the guidance, for many years, of Dr. Raymond, had made the work possible.

Mr. Hoover described graphically the nature of the Committee's labors, and narrated several incidents, humorous and pathetic, connected therewith.

Finally, in allusion to the later and present work of the Belgian Relief Commission, he said it was planting the American flag, not over conquered territory or public buildings or fortifications, but in ten million grateful hearts.

After Hoover had finished the Chairman announced that any speech which might follow would be in the nature of an anti-climax and the meeting then adjourned after again singing a verse of "America."

Annual Dinner.—The Annual Dinner was held at the Hotel Astor and was preceded by a reception at 7:00 p.m., which was given to President and Mrs. L. D. Ricketts and Mr. and Mrs. Herbert C. Hoover. After dinner short addresses were made by President L. D. Ricketts and President-elect Philip N. Moore. Following these addresses President Ricketts announced that the previous Board of Directors of the Institute, as their last official act, had unanimously elected Herbert C. Hoover an Honorary Member of the Institute. A tremendous ovation was given to Mr. Hoover, who spoke briefly of the work of his associates on the Commission for Relief in Belgium. The dinner then adjourned but was followed by dancing which lasted until well after midnight.

Luncheons.—On Monday, Tuesday, and Wednesday, luncheon was served in buffet fashion on the fifth floor of the Engineering Society's Building. The luncheons were largely attended by both members and ladies. On Wednesday noon several members of the Naval Consulting Board and the Army and Navy Departments of the United States were guests of the Institute at luncheon.

Moving Pictures in Colors and Lecture Thereon.—On Wednesday evening Dr. Herbert T. Kalmus gave a lecture illustrated by experiments and moving pictures in colors, showing the fundamental principles of the process. This was given in the auditorium of the Engineering Society's Building.

Excursion to West Point.—Notwithstanding that snow fell on Wednesday night and was falling on Thursday morning, 259 persons, of whom 81 were ladies, assembled at the ferry of the West Shore Railroad before 9:00 a.m. to take the special train. A Pullman car had been provided for the convenience of the ladies and the sun appeared before the special train emerged from the tunnel onto the western bank of the Hudson, and thereafter the weather was all that could be desired. Upon arrival at West Point the whole party went to Cullum Memorial Hall, the ladies being conveyed in busses. From there the party journeyed to the Riding Academy, where a most interesting exhibition of mounted drill by the cadets was given. This was followed by luncheon at Cullum Memorial Hall. Twenty members of the West Point Military Academy staff, with members of their family, were the guests of the Institute at luncheon and afterward dancing was enjoyed by many of the members and guests. At 2 o'clock all assembled in the auditorium where an informal session was held at which Major Arthur S. Dwight, Chairman of the Institute's committee in charge of this excursion and Chairman of the Committee on Reserve Corps of Civilian Engineers, presided. After a few appropriate complimentary and congratulatory remarks Major Dwight introduced Col. John Biddle, Superintendent of the Military Academy, who welcomed the visitors to West Point, expressing his pleasure in the fortunate change of the weather which permitted them to see the place in the height of its winter beauty. As the most suitable entertainment which he could offer in his address, Col. Biddle proceeded to give a brief but exceedingly interesting summary of the history of the Military Academy, emphasizing the connection of Washington with its establishment and location, and concluding with an instructive statement of its purpose as a school of general preliminary training for officers, who receive further special instruction in other Government institutions, culminating in the War College at Washington.

Mr. Sidney Jennings, Vice President of the Institute, being called upon to reply, introduced, as its spokesman on this occasion, Dr. Rossiter W. Raymond, Secretary Emeritus, whose family had been for many years connected with the Military Academy, especially through his brother, the late Brig. Gen. Charles W. Raymond, of the Corps of Engineers, who spent many years at West Point as instructor; who, as the engineer of the post, erected several of the important buildings, such as the hospital, the cadet barracks, etc.; and who, besides his work for 40 years as a military engineer, was famous as explorer of the Yukon, constructor of the harbor of Philadelphia, designer and builder of the Delaware breakwater, and finally Chairman of the Board of Engineers which planned and directed the great enterprise of the Pennsylvania R. R. Co., with its tunnels, terminals, yards and shops, in New Jersey, Manhattan and Long Island.

Dr. Raymond spoke as follows:

"Mr. Chairman, Fellow-Engineers, Ladies and Gentlemen: I accept with frank pleasure the words just spoken concerning my brother. He was indeed a great civil as well as an accomplished military engineer; and he

illustrated and exemplified, in his long career of forty years of successful achievements without a single failure, that fraternal co-operation between the American engineers in civil life and the Engineer corps of the Army, the recent development of which has inspired us all with patriotic pride. My own distinct recollections of West Point date from the days of the War for the Union, when, as staff officer, I visited the Academy, and took great delight in being respectfully saluted by my cadet brother. In later years, after my brief military career was over, and during his unusually long residence here, I used to visit him, and to maintain a most agreeable friendship with many of his fellow-officers in the Corps of Engineers. I have spent a solemn half-hour to-day, before this session began, in wandering about this splendid memorial hall, and realizing, as I gazed upon these portraits and tablets, that my Army friends and contemporaries have departed, and that "glory guards their graves." But I have been comforted by meeting here, not only those who recall my generation, but also the comrades of the Raymonds of a later date. Out yonder, in the West Point cemetery—the fairest spot on earth—beside my brother's grave, is the grave of his son, Captain Jack Raymond of the 10th cavalry—oh! they all loved Jack!—and there are two of Jack's brothers in active service still. So I may fairly feel that the relations of more than half a century between my family and this historic Academy are still subsisting unimpaired. You will pardon, I am sure, this outbreak of personal feeling, which could not be suppressed in such a presence and at such an hour.

"We mining engineers have been accustomed to claim that mining engineering comprises the operations of all other branches of engineering, together with peculiar processes and difficulties of its own. But perhaps we must now yield that proud comprehensive pre-eminence to the military engineers. It is they who have to know and to employ in modern warfare all arts and sciences. That even the highly abstruse and abstract science of astronomy may be a weapon of conquest, let me prove to you by a little story.

"In 1869, a military expedition was sent by the United States from the mouth of the Yukon river a thousand miles or more, to the British post of Fort Yukon. The expedition was commanded by Captain Charles W. Raymond, of the Corps of Engineers, U. S. A., whose military force consisted of Michael Foley, private, of the 9th U. S. Infantry. The garrison of Fort Yukon, on the other hand, consisted of two men in the service of the Hudson's Bay Company.

"The American force established an encampment (one tent) on the esplanade—that is to say, in the front yard of the fort, where it mounted its heavy artillery—a zenith telescope. Bombarding the sky with this battery, it obtained on the 7th day of August, 1869, during an eclipse of the sun, the data from which the longitude of the post could be determined; and on the next day Capt. Raymond informed the British garrison that Fort Yukon was many miles west of the 141st meridian, which constitutes the eastern boundary of Alaska—in other words, that it was upon American soil—and called upon them to haul down the British flag and substitute the stars and stripes. This they declined to do; but, the two armies being drawn up on opposing lines, the garrison maintained an attitude of innocuous desuetude and watchful waiting, while Private Foley, 9th Infantry, U. S. A., by a bold and skillful sortie, effected the desired change of national colors; after which the British forces shook

hands, and amicably retired, to build a new fort on the other side of the international boundary. This was the first instance in history of the capture of an important fortified position by one man with a telescope! It was a prophetic illustration of the decisive importance of artillery! And what a significant indication did it present of the direction in which the millenium is to be reached in warfare by the progressive reduction of armaments! But we must bear in mind that as quantity is diminished, quality must be improved. The one gun which reduced Fort Yukon, nearly half a century ago, had an effective range of some 90,000,000 miles! (Laughter and applause.)

"Well, science having thus entered the sphere of war in these latter days, it is but natural that war should make its appeal to the men of science. Fifty years ago, there was some controversy between the civilian and the military engineers of the United States. We thought that *they* ought not to have the charge of great public improvements, like those of rivers and harbors and canals. And it was perhaps an aggravation of our dissatisfaction that they performed their work so well. But that controversy has settled itself, like all controversies, religious, political, or other, since the world began—not by the victory of either party, but through the advance of both parties to higher ground, where old differences are forgotten or disregarded or dismissed as trivial in the presence of a new danger, duty and devotion, common to all. Yes, the military engineers have swallowed us, and we do not disagree with them. Indeed, we are proud and happy in our new fellowship. Look at Arthur Dwight, who presides at this moment upon this platform. Do you think he cares for those academic distinctions which, a few months ago, he wore with such dignity and satisfaction? He's a *Major* now! And look at me! When I receive annually from the Recorder of the Military Order of the Loyal Legion of the United States a letter asking me to pay my dues, it thrills me with joy. For it is addressed to me as Captain, without any mention of titles in philosophy or jurisprudence; and it renews my youth and strength to have my alphabetical appendix thus removed!

"Yes, all American engineers are simply patriots now. These are tense and critical hours. Since we listened, a few minutes ago, to the guns which saluted the memory of Washington upon this, his natal day, our ears have been strained to catch the muttered message from a dark horizon, and we know not how soon the storm may break, even upon the beauty and the white-robed peace that surround us here.

"One of the earliest and greatest poets of our country, Drake, chose this region as the scene of an exquisite poetic narrative from which he carefully omitted all human passions or purposes. From its serene opening lines:

'The moon shines down on old Cronest;
She mellows the shades on his shaggy breast,
And seems his huge gray form to throw
In a silver cone on the wave below.'

to the last words—'the fays are gone,' the scenes and images of the poem are those of fairy-land. Yet the same pen wrote in the same environment that immortal lyric,

'When Freedom from her mountain height
Unfurled her standard to the air,
She tore the azure robe of night
And set the stars of glory there!—'

I will not recite it further. You know it. We used to speak it in school—and it is a good piece to speak to-day!

"As you recall it, let it be associated in your minds henceforth with the lovely and the lofty memories of this place and this day. Imagine that upon these heights of Storm King and Cronest was that 'mansion in the sun' whence Freedom 'called her eagle bearer down, and gave into his mighty hand the symbol of her chosen land.'

"And, as we shortly leave this charming place and these its gallant, friendly guardians, our hearts will breathe a friendly benediction upon the scenic beauty, the historic glory and the heroic sons of West Point!"

Major Dwight then introduced Capt. Stuart C. Godfrey, Corps of Engineers, U. S. Army, who, after a graceful acknowledgment of Dr. Raymond's tribute to West Point, and a reference to the mutual courtesies of the occasion, which had made his colleagues and himself both hosts and guests, gave an interesting talk upon the methods and weapons of modern warfare, illustrated with lantern-views from the present European war.

Mr. Philip N. Moore, President-Elect of the Institute, briefly expressed, for himself and other visiting members from the West, a high appreciation of the entertainment which they had received throughout this annual meeting, and the hope that many of the Eastern members would attend the St. Louis meeting, next summer, thus giving opportunity for some return of this courtesy. In closing, he introduced Mrs. Moore, who felicitously extended a similar acknowledgment and invitation to the New York Ladies Committee, and to the Eastern ladies generally.

The excursionists then spent the remaining time most agreeably in inspecting buildings, classrooms and points of interest, including the beautiful cemetery, the new chapel (where an organ concert delighted the music-lovers) the gymnasium (where there was a display of calisthenics and fencing) and Cullum Memorial Hall, with its impressive great auditorium, the walls of which are hung with portraits of the West Point graduates who have won distinction in the wars of the Union—many of them having fallen in battle. At 4.30 p.m. the special train bore the visitors back to New York where it arrived at about 6.15. Thus ended the largest meeting of the Institute which had ever been held, except where some sister Institution had joined forces with it. In size it was matched by interest and social pleasure, ranking in these two particulars as among the best ever held by the Institute. The number of members and guests who registered was 630, but it is known that many were present without registering.

Technical Sessions

Monday Morning, Feb. 19, 1917.—The opening session, like all the technical sessions, was held at the headquarters of the Institute in the Engineering Societies' Building, New York City. It was on Geology and was presided over by William Kelly.

The following papers were presented by their authors or authors' representatives:

Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba, By Max Roesler. (Discussed by William Kelly, J. T. Singewald, Jr., Benj. L. Miller.

John D. Irving, L. C. Graton, C. P. Berkey. Written discussion by W. L. Cummings.)
 The Manganese Ores of the Lafayette District, Minas Geraes, Brazil. By Joseph T. Singewald, Jr., and Benjamin Leroy Miller. (Discussed by L. C. Graton. Written discussion by F. L. Garrison.)

The following papers were presented by title:

Recent Geologic Developments on the Mesabi Iron Range, Minnesota. By J. G. Wolff. (Written discussions by Carl Zapffe, E. J. Collins, J. F. Wolff.)
 The Geology of the Bawdwin Mines, Burma, Asia. By M. H. Loveman.
 Geology and Ore Deposits of Mohave County, Arizona. By Frank C. Schrader. (Written discussions by J. C. Anderson, J. D. Sperr, J. B. Platts.)
 Manganese Ores of Russia, India, Brazil and Chile. By E. C. Harder. (Discussed by J. T. Singewald, Jr. Written discussion by H. K. Scott.)

Monday Morning, Feb. 19, 1917.—Session on Metallography, H. M. Boylston, presiding.

The following papers were presented by their authors or authors' representatives:

Recrystallization after Plastic Deformation. By Henry M. Howe.
 Grain Growth Phenomena in Metals. By Zay Jeffries.
 On Grain Growth. By Henry M. Howe. (The above papers were discussed by W. E. Ruder, F. E. Carter, J. W. Richards, H. M. Boylston. Written discussion by J. A. Mathews.)
 Tungsten and Molybdenum Equilibrium Diagram, and System of Crystallization. By Zay Jeffries. (Written discussion by A. G. Worthing.)
 The System Tungsten-Molybdenum. By Frank Alfred Fahrenwald. (The above papers were discussed by J. W. Richards, H. M. Boylston, Zay Jeffries.)

Monday Afternoon, Feb. 19, 1917.—This session was under the auspices of the Committee on Petroleum and Gas, Mark L. Requa presiding.

The following papers were presented by their authors or authors' representatives:

Reservoir Gas and Oil in the Vicinity of Cleveland, Ohio. By Frank R. Van Horn. (Discussed by David T. Day.)
 Evidence of the Oklahoma Oil Fields on the Anticlinal Theory. By Dorsey Hager. (Discussed by L. L. Hutchison, M. L. Requa, I. N. Knapp, C. Naramore, Dorsey Hager, F. T. Hirschberg, R. H. Johnson, M. M. Thompson. Written discussion by H. A. Wheeler.)
 The Possibility of Deep Sand Oil and Gas in the Appalachian Geo-Syncline of West Virginia.) By David B. Reger. (Discussed by M. M. Thompson, L. L. Hutchison.)
 The Need and Advantages of a National Bureau of Well Log Statistics. By W. G. Matteson. (Discussed by I. N. Knapp, B. L. Miller, Dorsey Hager, David B. Reger, R. H. Johnson, S. A. Taylor, M. L. Requa, C. Naramore, L. L. Hutchison.)

The following papers were presented by title:

Problems Connected with the Recovery of Petroleum from Unconsolidated Sands. By William H. Kobbé. (Discussed by I. N. Knapp, M. L. Requa, C. Naramore, Dorsey Hager. Written discussions by Arthur Knapp, I. N. Knapp.)
 The Influence of the Movement in Shales on the Area of Oil Production. By Richard A. Conkling. (Discussed by Dorsey Hager. Written discussion by D. W. Ohern.)

Monday Afternoon, Feb. 19, 1917.—Session on Milling and Smelting, Sidney J. Jennings presiding.

The following papers were presented by their authors or authors' representatives:

Magnetic Concentration of Low-Grade Magnetic Iron Ore. By S. Norton and S. LeFevre. (Discussed by J. L. W. Birkinbine. Written discussion by G. C. Foote, F. L. Nason.)

An Investigation on Rock Crushing Made at McGill University. By John W. Bell. (Discussed by R. B. T. Kiliani. Written discussion by C. W. Merrill.)
 Countercurrent Decantation. By Luther B. Eames. (Discussed by J. V. N. Dorr.)

The Viscosity of Blast-Furnace Slag. By Alexander L. Feild. (Discussed by R. H. Richards, J. W. Richards, H. A. Guess, A. L. Feild. Written discussion by W. McA. Johnson, Geo. K. Burgess, A. W. Fahrenwald, R. B. Sosman, G. A. Rankin, J. E. Johnson, Jr.)

Matte Granulation at Herculeum, Mo. By S. Paul Lindau and Henry B. Smith.

The following paper was presented by title:

The Function of Alumina in Slags. By Carl Henrich. (Discussed by A. S. Dwight, E. P. Mathewson, W. B. Boggs, J. W. Richards, W. C. Smith.)

Tuesday Morning, Feb. 20, 1917.—Annual Meeting, Professor J. W. Richards presiding.

Sixty-two members were present in person and 2276 voted by letter ballot.

President Ricketts was present and occupied the Chair, but on account of weakness in his voice, at his request, Vice-President Joseph W. Richards presided.

The minutes of the Annual Meeting held on Feb. 15, 1916, were read, and on motion, duly made and seconded, were approved.

Reports of the President, Secretary, and Treasurer were presented in writing.

Report of the Tellers for Election of Officers and Directors was presented in writing and the following Officers and Directors were declared elected:

Director and President,	Philip N. Moore,
Director and Vice-President,	Charles W. Goodale,
Director and Vice-President,	Mark L. Requa,
Director,	Robert M. Raymond,
Director,	Willet G. Miller,
Director,	Allen H. Rogers,
Director,	Howard N. Eavenson,
Director,	J. E. Johnson, Jr.

The report of the Tellers on Simplified Spelling was presented in writing and showed the following:

For	Against
Paragraph 1-919	Paragraph 1-290
Paragraph 2-765	Paragraph 2-429
Paragraph 3-659	Paragraph 3-540

The number of ballots returned unmarked leaving the matter of spelling for discussion by the Board of Directors amounted to 346.

The reports of the following committees were presented:

Library Committee;

Committee on Papers and Publications;

Committee on Membership.

The report of the J. A. Holmes Safety Association was presented in writing.

Tuesday Morning, Feb. 20, 1917.—Session on Non-Metallic Minerals, Heinrich Ries presiding.

The following papers were presented by their authors or authors' representatives:

A Study of the Silica Refractories. By J. Spotts McDowell. (Discussed by J. W. Richards, W. F. Rochow, H. D. Hibbard, J. B. Umpleby.)
The Conservation of Phosphate Rock in the United States. By W. C. Phalen. (Discussed by E. G. Spilsbury.)

The following paper was presented by title:

The Genesis of Asbestos and Asbestiform Minerals. By Stephen Taber. (Written discussion by J. C. Branner, J. A. Dresser, R. P. D. Graham, G. P. Merrill.)

Tuesday Afternoon, Feb. 20, 1917.—Session on Iron Blast Furnace Practice, Joseph W. Richards presiding.

The following papers were presented by their authors or authors' representatives:

Dry-Hot versus Cold-Wet Blast-Furnace Gas. By Linn Bradley, H. D. Egbert and W. W. Strong. (Discussed by J. W. Richards, C. P. Perin. Written discussion by R. J. Wysor, K. Huessener, F. H. Willcox.)

Some Suggestions Regarding Construction of Hot Blast Stoves. By Linn Bradley, H. D. Egbert and W. W. Strong. (Discussed by F. G. Breyer, L. E. Riddle, C. P. Perin.)

Potash as a By-Product from the Blast Furnace. By R. J. Wysor. (Discussed by J. S. Unger, L. E. Riddle, F. G. Breyer, H. A. Huston, W. H. Ross, G. A. Roush. Written discussion by C. H. Rich, J. S. Unger, W. A. Schmidt.)

The following paper was presented by title:

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces. By Henry Phelps Howland. (Discussed by J. W. Richards. Written discussion by A. H. Lee, N. M. Langdon, W. H. Blauvelt, J. W. Richards.)

Tuesday Afternoon, Feb. 20, 1917.—Session on Flotation, Edward P. Mathewson presiding.

The following paper was presented by the author's representative:

Notes on Flotation—1916. By J. M. Callow. (Discussed by E. E. Free, J. W. Bell, H. W. DuBois, G. D. Van Arsdale, J. V. N. Dorr, E. P. Mathewson, H. A. McGraw. Written discussion by Rudolf Gahl, E. R. Ramsey.)

Wednesday Morning, Feb. 21, 1917.—Session on The Manufacture of Iron and Steel, Henry D. Hibbard presiding.

The following papers were presented by their authors or authors' representatives:

The Seasoning of Castings. By Richard Moldenke. (Discussed by H. D. Hibbard, A. E. Outerbridge, Jr., Leonard Waldo, J. S. Unger, M. H. Medwedeff. Written discussion by A. E. Outerbridge, Jr.)

Roll Scale as a Factor in the Bessemer Process. By A. Patton and F. N. Speller. (Discussed by Henry D. Hibbard, C. S. Robinson, J. W. Richards, A. Patton. Written discussion by E. T. McCleary, H. H. Campbell, M. R. Stevenson.)

Temperature Measurements in Bessemer and Open-Hearth Practice. By George K. Burgess. (Discussed by J. W. Richards. Written discussion by R. C. Drinker.)

The Manufacture of Weldless Steel Tires for Locomotive and Car Wheels. By Guillaem Aertsen. (Discussed by Henry D. Hibbard, Bradley Stoughton.)

The following paper was presented by title:

Significance of Manganese in American Steel Metallurgy. By F. H. Willcox. (Discussed by J. W. Richards, Albert Sauveur, Richard Moldenke, D. F. Hewett, A. E. Outerbridge, Jr., Leonard Waldo, H. D. Hibbard. Written discussion by J. S. Unger.)

Wednesday Afternoon, Feb. 21, 1917.—Session on Metallography and Heat-Treatment of Steel, Albert Sauveur presiding.

The following papers were presented by their authors or authors' representatives:

Erosion of Guns—The Hardening of the Surface. By Henry Fay. (Discussed by Hudson Maxim, Bradley Stoughton, Leonard Waldo, H. C. Wilson, J. W. Richards, A. L. Walker. Written discussion by Albert Sauveur, Ralph Earle, Lawrence Addicks.)

Notes on the Heat Treatment of High-Speed Steel Tools. By A. E. Bellis and T. W. Hardy. (Discussed by C. G. Fink, J. A. Mathews, H. M. Boylston, A. Sauveur. Written discussion by M. H. Medwedeff, Robert J. Anderson, F. C. Langenburg.)

Effect of Time in Reheating Hardened Steel Below the Critical Range. By C. R. Hayward and S. S. Raymond. (Discussed by A. Sauveur, J. A. Mathews, Bradley Stoughton, H. M. Boylston, Wm. Campbell, M. H. Medwedeff.)

The Effect of Sulphur on Low-Carbon Steel. By Carle R. Hayward. (Discussed by A. Sauveur, M. H. Medwedeff, J. S. Unger, G. Aertsen, Leonard Waldo. Written discussion by G. F. Comstock.)

A Method for Distinguishing Sulphides from Oxides in the Metallography of Steel. By George F. Comstock. (Discussed by H. M. Boylston, G. F. Comstock.)

Wednesday Afternoon, Feb. 21, 1917.—Session on Mining Methods, etc., R. M. Catlin presiding.

Anthracite Stripping. By J. B. Warriner. (Discussed by S. A. Taylor, R. V. Norris, T. M. Dodson, Geo. S. Rice, Harrison Souder, H. M. Chance, Robert Peele, R. M. Catlin, H. M. Crankshaw, A. O. Ihlseng, W. S. Ayres, D. B. Reger, B. F. Tillson, E. M. Chance.)

Portable Miners' Lamps. By Edwin M. Chance. (Discussed by G. S. Rice. Written discussion by Herbert M. Wilson, M. D. Cooper, Robert P. Burrows, H. H. Clark.)

The following papers were presented by title:

The Pennsylvania Mine Fire, Butte, Mont. By C. E. Nighman and R. S. Foster. Shot-Firing in Bituminous Mines. By M. D. Cooper. (Discussed by B. L. Tillson, Lucien Eaton, T. M. Chance, H. M. Crankshaw, R. M. Catlin, E. T. Lednum. Written discussion by J. J. Rutledge, Geo. S. Rice, E. M. Chance, D. Harrington, A. La Motte.)

Report of the Secretary of the Committee on Safety and Sanitation. (Discussed by B. F. Tillson, E. M. Shipp, H. N. Eavenson, Wallace McKeehan.)

WOMEN'S AUXILIARY OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

As already announced in the notice of the 114th (New York) Meeting of the Institute, the ladies present met and organized the Women's Auxiliary of the Institute. All ladies of the families of members of the Institute are eligible to this Auxiliary. The purpose and objects of the Auxiliary are well stated in the address of the President, Mrs. S. J. Jennings, in opening the organization meeting on the morning of Feb. 20, 1917. The address follows:

"Ladies: Friends from far and near, on behalf of the Women's Entertainment Committee of the A. I. M. E., I extend to you a hearty welcome.

"Most especially are we glad to see our friends from the Pacific Coast and the middle West, whose royal hospitality to the members of the A. I. M. E. and their wives and daughters on their trips to California and to the Copper Country of the Southwest can never be forgotten. We are glad to be able in some small measure to return it.

"This interchange of visits has given to us the idea that it would be pleasant and profitable for the women of the families of the engineers to have some closer bond of union, such as is now enjoyed by the men, and that we might form a Women's Auxiliary to the A. I. M. E., whose members might jointly and individually interest themselves in work for their country or community or for humanity at large.

"That the engineer is the pioneer of civilization is a platitude we are accustomed to hear frequently aired, and we women are rather apt to consider complacently what fine men we have for husbands and what splendid work they have done, and we are not at all blind to the fact that we have been contributors to their success, if not by helping them with mathematical calculations, like one charming wife I know, or like another forceful, delightful woman who put in a pump in her husband's absence, in another way by doing our ordinary woman's work of making home pleasant, providing digestible food, and keeping as far as possible the worries of life from the man who has problems enough in his work to fill two ordinary lives.

"Year by year we grow more keenly alive to the fact that as the art of the engineer draws the world closer together and brings far distant regions within our ken; that as the great ships, the outcome of the mechanical engineer's brain, carry the minerals the mining engineer has wrested from the earth, from continent to continent; that as the skill of the engineers of all kinds is brought into play in the terrible war for freedom and justice, in the throes of which Europe struggles bleeding, so do our responsibilities to humanity increase. We can no longer indulge in the comfortable selfishness of confining our energies to our homes. We have time not fully employed there and humanity calls on woman for help and sympathy in its troubles and woes, for fellow feeling in its joys, and for mutual aid in getting the very best from life. Humanity's cry is insistent and it calls on us to work and to help, to give of our time, our capacity, our thought, our talent, our wealth. Are we to turn a deaf ear and say we have enough to occupy us in our homes? It is a strange anomaly that it is always the busiest people who have the most time. I believe that we are most of us, if not all, giving of ourselves for the world at large, but you remember the old fable that carries an eternal truth: An old man on his death bed called his sons around him and gave them a bundle of sticks tied with a band and told them to break the bundle. The sticks were stout and strong and not one of the sons had strength enough to break the fagot. "Now my sons," said the father, "loosen the band and break the sticks one by one." It was easily done. "See," said the old man, "United you stand strong, singly you break and fall." It is this truth we have in mind when we put before you a scheme to form a Women's Auxiliary to the A. I. M. E., a society in which we can all work together for our country, for our community, or for any section of humanity.

"If it is your pleasure to sanction any such organization, it would seem a good thing that at these yearly meetings we should settle on some particular work that we can do, preferably in collaboration with the A. I. M. E., then that each member returning to her own home should form a center and enlist other women in her district to help. It would probably be wise to have on the governing council six or seven directors from various parts of the country to represent the views of different sections.

"Our work, our help is needed. Can we not work more efficiently together as one band, than as separate units? It may be that our country will need us soon—humanity needs us now. May we make it our proud boast that the women of the A. I. M. E., are ready at the first call?"

Notice will shortly be sent to the members of the Institute asking for information regarding the ladies of their families. The membership dues of the Auxiliary are \$1 per year, which should be sent to the Treasurer. The officers are as follows:

President, Mrs. S. J. Jennings,
First Vice-President, Mrs. Arthur S. Dwight,
Second Vice-President, Mrs. Henry S. Munroe,
Third Vice-President, Mrs. H. W. Hardinge,
Treasurer, Mrs. George D. Barron,
Corresponding Secretary, Mrs. Axel O. Ihlseng,
Recording Secretary, Mrs. Bradley Stoughton.

JOSEPH A. HOLMES SAFETY ASSOCIATION

As a representative of the Institute in the Joseph A. Holmes Safety Association, which comprises some twenty-three different National Societies, it is proper that I make some report of the progress of the Association to the Institute. The societies represented in the organization are as follows:

Association—	Representatives, 1916
American Association for the Advancement of Science.....	L. O. Howard
American Chemical Society.....	Charles Baskerville
American Electro-Chemical Society.....	F. G. Cottrell
American Federation of Labor.....	A. E. Helder
American Forestry Association.....	P. S. Ridsdale
American Institute of Electrical Engineers.....	John H. Finney
American Institute of Mining Engineers.....	Hennen Jennings
American Mining Congress.....	David T. Day
American Red Cross Society.....	Maj. Robert U. Patterson
American Society for Testing Materials.....	A. W. Gibbs
American Society of Mechanical Engineers.....	John A. Brashear
Bureau of Mines.....	Alternate, Gen. W. H. Bixby
Geological Society of America.....	Van H. Manning
International Railway Fuel Association.....	Joseph Hyde Pratt
Mine Inspectors' Institute.....	Eugene McAuliffe
Mining and Metallurgical Society of America.....	J. W. Paul
National Academy of Sciences.....	George S. Rice
National Safety Council.....	David White
Smithsonian Institution.....	H. M. Wilson
Society for the Promotion of Engineering Education.....	Charles D. Walcott
United Mine Workers of America.....	M. E. Wadsworth
U. S. Geological Survey.....	Alternate, O. P. Hood
Western Federation of Miners.....	William Green
	George Otis Smith
	Jos. D. Cannon

A general circular letter outlining the objects of the Joseph A. Holmes Safety Association was sent to our membership some weeks ago; to this and the other National Societies connected with the Joseph A. Holmes Safety Association, some 41,000 announcements have been sent. Accompanying the circular, for the convenience of the members, a subscription blank was inclosed. This initial effort was more useful and instructive

in formulating the object of the Association and giving its composition than in obtaining the money required. The subscriptions were small in number, inasmuch as the circular purposely avoided any definite statement as to the amount of money required for founding this Association. Up to date, 76 subscriptions have been received, varying in amount from \$1 to \$1,000, and there has been received and pledged in this way nearly \$10,000.

The minimum requirements for the right carrying on of the work planned and outlined in the circular will necessitate an endowment fund of at least \$100,000, the interest on which is to defray the necessary expenses of the organization, which will include, besides the cost of medals and honorariums, the necessary office and other expenses to keep up the organization and intelligently investigate the awards.

In addition to a memorial to Joseph A. Holmes, the work of the Association will stimulate the saving of life and limb in both mining and metallurgical industries and will bring leaders of science, labor and capital in touch on humane lines. The organization will have also exceptional facilities, through the 23 different societies connected with it, of obtaining extensive, intelligent and expert advice as to the fitness of its awards for personal heroism and worthy invention.

The American Institute of Mining Engineers took the lead in starting a memorial to Joseph A. Holmes. The nature of the memorial as outlined in the Safety Association is in keeping with his work and should be upheld and fostered by our Institute.

For further details in connection with this work, I would refer to the Secretary of the Safety Association, Dr. David T. Day, Bureau of Mines, Washington, D. C.

HENNEN JENNINGS.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

Members and guests who called at Institute headquarters during the period Feb. 10, 1917 to Mar. 10, 1917:

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|---|--|
| F. H. Armstrong, Vulcan, Mich. | G. Komar, Staten Island, N. Y. |
| Franklin Bache, Philadelphia, Pa. | H. M. LaFollette, LaFollette, Tenn. |
| R. A. Bingham, W. Orange, N. J. | H. Mortimer-Lamb, Montreal, Canada. |
| G. S. Blair, Pittsburgh, Pa. | J. S. Lane, New York, N. Y. |
| Graham Bright, Pittsburgh, Pa. | D. H. McLaughlin, Cambridge, Mass. |
| C. A. Burdick, New York, N. Y. | F. J. McMahon, Wilkes-Barre, Pa. |
| Frederick P. Burrall, Dawson, Y. T. | F. Oskar Martin, San Francisco, Cal. |
| William Campbell, New York, N. Y. | C. Mentzel, New York, N. Y. |
| G. L. Carlisle, Jr., New York, N. Y. | T. C. Merriam, New Haven, Conn. |
| J. Murray Clark, Toronto, Canada. | K. A. Pauly, Schenectady, N. Y. |
| Wayne Darlington, Philadelphia, Pa. | Francis R. Pyne, Elizabeth, N. J. |
| Ralph W. Deacon, Elizabeth, N. J. | C. S. Robinson, Youngstown, O. |
| P. St. John Dixon, So. Porcupine, Canada. | W. A. Scheuch, New York, N. Y. |
| A. E. Drucker, New York, N. Y. | Jesse Scobey, New York, N. Y. |
| Lucien Eaton, Ishpeming, Mich. | Dunbar D. Scott, Rochester, N. Y. |
| W. E. Goldsborough, New York, N. Y. | W. C. Smith, Elizabeth, N. J. |
| C. Marshall Gravatt, Port Royal, Va. | Charles Spearman, Mt. St. Patrick, Can. |
| C. N. Hickok, Cleveland, O. | Harold C. E. Spence, Duncan, Ariz. |
| C. Vey Holman, Rockland, Me. | M. M. Thompson, Morsemere, N. J. |
| W. F. Jahn, Kirkland Lake, Canada. | Benjamin F. Tibby, Salt Lake City, Utah. |
| F. E. Johnesse, Boise, Ida. | J. S. Unger, Pittsburgh, Pa. |
| E. C. Johnson, Chicago, Ill. | |

D. C. Bard has joined the geological staff of the C. M. & St. Paul R. R., with headquarters at Seattle.

Pierce Barker, chief chemist at the Washoe Reduction Works of the Anaconda Copper Mining Co., has been promoted to the position of superintendent of blast furnaces in place of **H. W. Aldrich**, resigned.

Professor R. C. Carpenter reaches the retiring age at the end of the present college year and will sever his active connection with Cornell University at that time.

Respecting his retirement, the Committee on General Administration of the Board of Trustees adopted the following resolution:

"Resolved, that the Trustees in accepting the resignation of Professor Carpenter express their high appreciation of his services to the University for nearly thirty years. As a pioneer in the field of experimental engineering he is held in the highest esteem by all mechanical engineers, and by his writings in this field he has made an assured place for himself in the annals of his profession. As a teacher and investigator he is affectionately remembered by many generations of students and his retirement from the Faculty of Sibley College will be viewed with great regret by all his colleagues."

Professor Carpenter retires in good health and expects to maintain his activities in the fields of engineering investigation and research for several years to come.

B. F. Cresson, Jr., has resigned as chief engineer of the New Jersey State Board of Commerce and Navigation and has opened an office as Consulting Engineer at 30 Church St., New York, N. Y.

Murray M. Duncan, manager of the Cleveland-Cliffs Iron Co., has been appointed vice-president, with headquarters at Marquette, Mich.

C. H. Fry, for many years superintendent of the Yellow Aster mine at Randsburg, Cal., has resigned his position.

Sir Robert Hadfield has been appointed president of the Faraday Society, of London.

W. M. Kelsey, formerly general superintendent of the Mineral Point Zinc Co., Depue, Ill., has been made superintendent of the Palmerton plants of the New Jersey Zinc Co. (of Pa.), Palmerton, Pa.

Edward E. Loomis, until recently vice-president of the Delaware, Lackawanna & Hudson R. R., has been elected president of the Lehigh Valley R. R.

F. L. Manning has been appointed general manager of the Peebles Paving Brick Co., Portsmouth, Ohio.

Bayard S. Morrow has been appointed superintendent of grinding and flotation, Copper Concentrator, Washoe Reduction Works, Anaconda, Mont.

Brent N. Rickard has been transferred to the Murray plant of the American Smelting & Refining Co.

Francis P. Sinn has been made general superintendent of the New Jersey Zinc Co. (of Pa.), Palmerton, Pa.

J. A. Singmaster, formerly general superintendent of the New Jersey Zinc Co. (of Pa.), Palmerton, Pa., is now general manager of the technical department, with offices at 55 Wall St., New York City.

J. D. Sperr has severed all connections with companies operating in the Jerome, Arizona, district, and has changed his address temporarily to Houghton, Mich.

C. M. Thompson has been appointed division engineer of the Union Pacific R. R., Denver, Colo.

W. J. Uren has been appointed manager of the Seneca mine, Michigan.

Elton W. Walker has resigned as superintendent of the Lake Copper Co.

W. A. Williams, Chief Petroleum Engineer, U. S. Bureau of Mines, San Francisco, Cal., has resigned to accept the position of assistant to the general manager of the Empire Fuel & Gas Co., Bartlesville, Okla.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Member, aged 32. Fifteen years' experience in the mining, quarrying, and manufacture of gypsum and gypsum products, as purchasing agent, designer, operating manager and managing executive. Thoroughly familiar with the business in all branches, especially the organization of efficient producing forces and machinery and the value of cost and their application to manufacturing. Desires responsible position with a financially reliable concern in the same or similar business in which an interest may be obtained at a later date. Now doing consulting work, but free for appointments after April 1. New York or Philadelphia. No. 345.

Member, technical graduate. Unmarried. Aged 32. Seven and one half years' experience in mining, timbering, and mine sampling. At present, and for past two and a half years, mine sampler for well known and highly successful mining company. Desire change. Go anywhere. Prefer assisting Mining Engineer. No. 346.

Member, 40 years old. 18 years' experience in examinations and operation in the United States, Mexico and South America. Especially successful with poor class of labor. Open for engagement in any part of the world, in technical, administrative or consulting work. No. 347.

Member, technical graduate. Sixteen years' experience as engineer, superintendent, and manager in metal mines. Exploratory work, developing and equipping new properties. Operation of large mines, and various systems of mining. Low costs. Specialized in iron and copper mining, but have had some experience in zinc, lead and pyrite mining. No. 348.

Member, technical graduate, degrees of B. S. and M. S.; specialized in geology. Aged 26. Experience in Arizona, Mexico, and New England. Desires geological work of any kind. Go anywhere in North America. Speak Spanish. No. 349.

Member, M. S., Met. E., Harvard University. Five years' experience in metallurgy of copper and steel, including metallography, physical testing and research under mill conditions. Exceptional training and broad practical experience. Desire teaching position in Metallurgy or Metallography with chance for research. No. 350.

Member, technical graduate, B. S., aged 43, seeks position superinten-

dent or assistant superintendent, Twenty-five years' general mining experience in New Zealand, Australia, South Africa (Rand) and West Africa (present location). Specialty mine development. Thorough experience all kinds of development, mine sampling and surveying (Rand mine surveyor's certificate). Lately finished two years' strenuous development work large new East Rand company. Knowledge French and German. Robust constitution—go anywhere. Free June 1st. No. 351.

LOCAL SECTION NEWS

ST. LOUIS SECTION

H. A. BUEHLER, *Chairman*,

CHARLES T. ORR, *Vice-Chairman*,

THOMAS T. BREWSTER, *Vice-Chairman*, L. L. HUTCHISON, *Vice-Chairman*,
W. E. McCOURT, *Secretary-Treasurer*, Washington University, St. Louis.

CHARLES J. ADAMI,

EUGENE MCAULIFFE,

A. C. TERRILL,

JAMES A. CASELTON,

JAMES D. ROBERTSON.

The annual meeting of the St. Louis Section was held at the St. Louis Club, Saturday evening, Mar. 10, 1917. Philip N. Moore, President, and Bradley Stoughton, Secretary of the Institute, were the guests of the Section. The meeting was preceded by a dinner at which 70 were present, and was presided over by F. W. DeWolf.

Addresses were made by Mr. Moore and Mr. Stoughton.

The routine reports of the officers were presented and the Executive Committee named above was elected for the current year.

The special business of the meeting was a report of progress concerning the arrangements for the Institute meeting to be held in the St. Louis Section district, Oct. 8 to 13, 1917. The general program for the meeting met with the cordial approval and coöperation of the members of the Section. It was evident that the members are united in an endeavor to arrange a successful meeting.

W. E. McCOURT, *Secretary*.

AFFILIATED STUDENT SOCIETIES

The School of Mines Society of Columbia University held a very successful meeting on Jan. 17, 1917. The Society was fortunate in having three good speakers for the evening. Mr. Charles Piez, President of the Link Belt Co., spoke on the Art of Conveying Materials, illustrating his talk with lantern slides. Mr. J. P. Channing, Vice-President of the Miami Copper Co., gave an interesting and instructive talk on Human Engineering, and Mr. F. H. Rindge, Jr., Secretary of the Y. M. C. A. Industrial Service Department, spoke about the relation of the young college graduate to the workingman.

We expect to have several more meetings of this sort.

RICHARD P. E. HERMSDORF, *Secretary*.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS
AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining

- ABSTRACTS OF CURRENT DECISIONS ON MINES AND MINING, REPORTED FROM MAY TO AUGUST, 1916. (U. S. Bureau of Mines. Bulletin 143.) Washington, 1917.
- GEORGII AGRICOLA. LAE DE RE METALLICA. LIBER PRIMUS. n.p. n.d. (Gift of E. P. Mathewson.)
- PROSPECTING AND MINING OF COPPER ORE AT SANTA RITA, NEW MEXICO. (U. S. Bureau of Mines. Bulletin 107). Washington, 1916.
- QUARRY ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEAR 1915. (U. S. Bureau of Mines. Technical Paper 165.) Washington, 1917.

Metallurgy, Chemistry and Metallography

- THE ANALYTICAL DISTILLATION OF PETROLEUM. (U. S. Bureau of Mines. Bulletin 125.) Washington, 1916.
- BIBLIOGRAPHY OF RECENT LITERATURE ON FLOTATION OF ORES, JANUARY TO JUNE, 1916. (U. S. Bureau of Mines. Technical Paper 135.) Washington, 1917.
- BY-PRODUCT COKE AND GAS OVENS AND SYSTEMS FOR THE RECOVERY OF AMMONIA, TAR AND BENZOLS. Designed and built by H. Koppers Company. Pittsburgh, n.d. (Gift of H. W. Nicolson.)
- COBALT ALLOYS WITH NON-CORROSIVE PROPERTIES. (Canada. Mines Dept. Pub. No. 411.) Ottawa, 1916.
- EXPERIMENTAL RESEARCHES ON THE SKIN EFFECT IN STEEL RAILS. By A. E. Kennelly, F. H. Achard and A. S. Dana. (Reprinted from the Journal of the Franklin Institute, August, 1916.) (Gift of Massachusetts Institute of Technology, Electrical Engineering Dept.)
- INDUSTRIAL RESEARCH IN CANADA. By J. C. McLennan. An address to the members of the Royal Canadian Institute, Nov. 4, 1916. Toronto, 1916. (Gift of Royal Canadian Institute.)

- INSPECTION REPORT ON WASHINGTON PAINT OIL TESTS AND WASHINGTON CEMENT PAINT TESTS. (Bulletin No. 53, Paint Manufacturers' Association of the U. S.) (Gift of Institute of Industrial Research.)
- OPERATING DETAILS OF GAS PRODUCERS. (U. S. Bureau of Mines. Bulletin 109.) Washington, 1916.
- PHYSICAL AND CHEMICAL CONSTANTS, TABLES OF. Ed. 2. By G. W. C. Kaye and T. H. Laby. London-New York, 1916.
- PHYSICAL AND CHEMICAL PROPERTIES OF GASOLINES SOLD THROUGHOUT THE UNITED STATES DURING THE CALENDAR YEAR 1915. (U. S. Bureau of Mines. Technical Paper 163.) Washington, 1916.
- PHYSICO-CHEMICAL PROPERTIES OF STEEL. By C. A. Edwards. London, 1916.
- PRINCIPLES AND PRACTICE OF SAMPLING METALLIC METALLURGICAL MATERIALS. (U. S. Bureau of Mines. Bulletin 122.) Washington, 1916.
- THE WHITE PIGMENT INDUSTRY. By H. A. Gardner. (Bulletin No. 54, Paint Manufacturers' Association of the U. S.) (Gift of Institute of Industrial Research.)

Geology and Mineral Production

- DUTCH EAST INDES. JAARVERSLAG VAN DE WINNING, HET VERVOER EN DEN VERKOOP VAN BANKA-TIN OVER HET EXPLOITATIEJAAR 1914. Batavia, 1916.
- FELDSPAR IN CANADA. (Canada. Mines Department, Pub. No. 401.) Ottawa, 1916.
- GEOLOGY OF THE COUNTRY TO THE SOUTH OF KALGOORLIE, INCLUDING THE MINING CENTERS OF GOLDEN RIDGE AND FEYSVILLE. (Western Australia. Geological Survey, Bulletin No. 66.) Perth, 1916.
- GRAPHITE, 1916. (Gift of Joseph Dixon Crucible Co.)
- INDIANA. DEPARTMENT OF GEOLOGY AND NATURAL RESOURCES. Annual Report 40th. Indianapolis, 1916.
- MARLS AND LIMESTONE OF MISSISSIPPI, PRELIMINARY REPORT ON. (Bulletin No. 13, Mississippi State Geological Survey.) Jackson, 1916.
- MINING AND QUARRY INDUSTRY OF NEW YORK STATE. Report of Operations and Production during 1915. Albany, 1916. (Gift of University of the State of New York.)
- MOLYBDENUM; ITS ORES AND THEIR CONCENTRATION WITH A DISCUSSION OF MARKETS, PRICES AND USES. (U. S. Bureau of Mines. Bulletin 111.) Washington, 1916.
- PENNSYLVANIA. Department of Mines. Report. Part II. Bituminous, 1915. Harrisburg, 1916.
- PILOT BUTTE OIL FIELD. (Wyoming State Geologist. Bulletin No. 13.) Cheyenne, 1916.
- STUDY OF THE MAGMATIC SULPHIDE ORES. By C. F. Tolman and A. F. Rogers. Stanford University, Cal., 1916. (Gift of Leland Stanford Junior University.)

General

- AMERICAN SMELTING & REFINING Co. New York, N. Y. Safety Bulletin. Jan.-Feb., 1917.
- HISTORY OF THE CHEMICAL LABORATORY OF THE UNIVERSITY OF MICHIGAN, 1856-1916. By E. D. Campbell. Ann Arbor, 1916. (Gift of University of Michigan.)
- HYDRO-ELECTRIC POWER. By Lamar Lyndon. 2 vols. Vol. I—Hydraulic Development and Equipment; Vol. II—Electrical Equipment and Transmission. New York, McGraw-Hill Book Co. 1916.

Trade Catalogs

- CEMENT GUN CONSTRUCTION COMPANY. Chicago, Ill. Bulletin No. 5. Gun crete for protection. Illustrations of Gun-crete work.
- DE LAVAL STEAM TURBINE Co. Trenton, N. J. Catalog "B." High Efficiency Centrifugal Pumps.
- LEHIGH CAR, WHEEL & AXLE WORKS. Catasauqua, Pa. Catalogue 50. Fuller Quality Products. Price list covering face hardened sprocket and traction wheels. 1914.
- Catalog 70. Fuller Lehigh Pulverizer Mill. 1915.
- Catalog 71. Pulverized coal equipment. 1916.
- LESCHEN, A. & SONS Co. St. Louis, Mo. Leschen's Hercules. Feb., 1917.
- STEEL MINE TIMBERS. Tables and data on the properties and uses of sections. Ed. 7. Pittsburgh, 1917. (Gift of Carnegie Steel Company.)
- SULLIVAN MACHINERY Co. Chicago, Ill. Bulletin 70 A. Rotator Hammer Drills. Classes "DP33" and "DR33." Ed. 2. Jan., 1917.

MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period Feb. 10, 1917 to Mar. 10, 1917.

- ANDERSON, LAWRENCE W., Min. Engr., Boston Mine, Utah Copper Co.
Bingham, Utah.
- AUSTIN, JOHN W., Assayer, Magma Copper Co., P. O. Box 101, Superior, Ariz.
- BARNARD, C. W., Treas. and Resident Mgr., Arizona Asbestos Assn., Globe, Ariz.
- BIERMAN, A. C., Geol., Producers Oil Co., Box 75, Houston, Tex.
- BOCHERT, W. C., Prof. of Mining, South Dakota State School of Mines,
Rapid City, So. Dak.
- BROOKS, FLOYD R., Shift Boss, Mond Nickel Co., Ltd., Worthington, Ont., Can.
- BROWN, CECIL J., Draftsman & Surveyor, Wallaroo & Moonta Min. & Smelt. Co.,
Wallaroo, South Australia.
- BURRAGE, ALBERT C., Vice-Pres., Chile Copper Co., 85 Amos Bldg., Boston, Mass.
- BURGESS, JOHN ALBERT, Genl. Supt., United Eastern Min. Co., Oatman, Ariz.
- CARLOCK, JOHN BRUCE, Supt., O'Neal Oil Process Co.,
608 Haas Bldg., Los Angeles, Cal.
- CARPENTER, BYRON H., Min. Engr., Willoughby, Ohio.
- CHARLES, PERRY L., Oil Flotation Dept., Anaconda Copper Min. Co.,
Box 842, Anaconda, Mont.
- CHEN, H. T., Engr. in Chief, Yunnan Tin Trading Co., Ketchiu-Chang,
Yunnan Province, China.
- CLARK, JAMES M., Clark & Krebs, 509 Brooks St., Charleston, W. Va.
- COWPERTHWAIT, THOMAS, Chief Safety Inspector, Calumet & Arizona Min. Co.,
Box 794, Warren, Ariz.
- CRANE, EARL B., Mgr., Black Butte Quicksilver Mine, Blackbutte, Ore.
- CROMWELL, CHARLES W., Civil and Structural Engr., A. G. McGregor,
Box 143, Warren, Ariz.
- CURLEY, MICHAEL, Genl. Supt., New Cornelia Copper Co., Ajo, Ariz.
- CUSHMAN, CARLETON W., Mine Supt., Weedon Mining Co., Ltd.,
Weedon, Que., Canada.
- DENNIS, ARTHUR C., Geologist, Gulf Production Co., 915 North Dixon St.,
Gainesville, Tex.
- DEWEY, GEORGE C., Chemist & Assayer, Selby Smelt. & Lead Co., Selby, Cal.
- DONALDSON, KENNETH H., Genl. Foreman, Concentrator No. 4,
Arizona Copper Co., Ltd., Clifton, Ariz.
- DOUGHTY, JOHN H., Mechanical Engr., Lehigh & Wilkes-Barre Coal Co.,
Wilkes-Barre, Pa.
- DRUDING, FRANK D., Asst. Supt. of Mills, Federal Lead Co., Flat River, Mo.
- DULIEUX, PAUL E., Cons. Min. Engr., Head of the Engineering Service,
With French Purchasing Commission in U. S., 10 Bridge St., New York, N. Y.
- DUSCHAK, LIONEL HERMAN, Chemical Engr., U. S. Bureau of Mines,
Mining Bldg., Berkeley, Cal.
- EICHRODT, CHARLES WHARTON, Min. Engr., Shift Boss,
Inspiration Cons. Copper Co., Miami, Ariz.
- ELAYER, CARL S., Mine Foreman, Arizona Commercial Mining Co., Copper Hill, Ariz.
- ENGLEMANN, EDWARD W., Supt. of Flotation, Ray Consolidated Copper Co.,
Hayden, Ariz.
- FAY, CHARLES H., Min. Engr., Empire Zinc Co, Hanover, New Mex.
- FELTON, HERBERT WALTER, Mine Foreman, Moctezuma Copper Co.,
Pilares de Nacozari, Son., Mexico.
- FIELD, ROYCE A., Night Supt., B. H. P. Steel Works, Crebert St., Mayfield,
Newcastle, N. S. W., Australia
- FINNELL, FREDERICK HENRY, Supt., Plant 1, Prime Western Spelter Co., Gas, Kans.
- FLORES, MAXIMO MACAMBYRA MONTE, Min. Engr., Caixa No. 84, Bahia, Brazil,
South America.
- FORBES, DUNCAN DAVIDSON, Draftsman and Estimator, Arizona Copper Co.,
Clifton, Ariz.
- FORRESTER, DAVID L., Testing Engr., Arizona Copper Co., Ltd., Morenci, Ariz.
- FREE, EDWARD E., Chemical Engr., 1105 Madison Ave., Baltimore, Md.

- GARRATT, FRANK Genl. Supt., Latrobe Electric Steel Co., Latrobe, Pa.
GARVIN, JOHN CREIGHTON, Geol. and Min. Engr., 720 Payton Block, Spokane, Wash.
GIESER, HERBERT S., Foreman, Flotation and Regrinding Dept.,
Anaconda Copper Mining Co., Anaconda, Mont.
GILMAN, GEORGE H., Chief Engr., Pneumatic Drill & Channeler Dept.,
Sullivan Machinery Co., Claremont, N. H.
GRAVATT, C. MARSHALL, Min. Geologist Port Royal, Va.
HAMILTON, ALFRED McL., Supt., American Smelt. & Ref. Co., Saco Smelter,
via Red Rock, Ariz.
HAMILTON, ORR R., Min. Engr., Michigan Geological Survey &
Board of State Tax Commissioners, Lansing, Mich.
HIGLEY, ROY C., Supt. of Mines, Caucasus Copper Co., Batoum, South Russia.
HOLMBERG, C. L., Min. Engr., United Verde Extension Min. Co., Box 673,
Jerome, Ariz.
HONDRUM, OLOF, Chief Engr., Cananea Cons. Copper Co., Cananea, Son., Mexico.
HOTCHKISS, WILLIAM O., State Geologist of Wisconsin, Science Hall, Madison, Wis.
HOWE, ROY EDWIN, Asst. Supt., Converter Dept., Anaconda Copper Min. Co.,
500 Pine St., Anaconda, Mont.
HUMES, H. G., Genl. Foreman, Silver King Coalition Mines Co., Park City, Utah.
JOHNSON, GUNNARD E., Chemist, Arizona Copper Co., Box 812, Morenci, Ariz.
JOSLIN, GARNETT ALFRED, Min. Engr. 321 Felt Bldg., Salt Lake City, Utah.
KEMP, DONALD CAMPBELL, Geol., Instructor in Geology,
Missouri School of Mines and Metallurgy, Rolla, Mo.
KING, ROWLAND, Genl. Supt., Highland Valley Min. & Dev. Co., (N. P. L.),
Ashcroft, B. C., Canada.
KLEINHAMMER, PAUL H. . . . Min. Engr., Vinegar Hill Zinc Co., Platteville, Wis.
KOERING, BRUNO R., Pres. & Genl. Mgr., Koering Cyaniding Process Co.,
511 Hammond Bldg., Detroit, Mich.
KROEGER, HERBERT BENJAMIN, Min. Engr., Supt., Cauto Min. Co., San Nicolas,
Oriente Prov., Cuba.
LANG, SIDNEY S., Min. Engr., Supt. of Naumkeag Copper Co., Houghton, Mich.
LEE, OSCAR, Min. Engr., Asst. Testing Engr., Anaconda Copper Min. Co.,
Washoe Smelter, Anaconda, Mont.
LEONARZ, ALBERT, Supt. of Preparation, Lehigh Coal & Navigation Co., Lansford, Pa.
LEVISON, STUART H., Chemist, American Smelt. & Ref. Co., Apartado 101,
Monterrey, N. L., Mex.
LINDSLEY, NORMAN D., Chief Sampler, Leasing Dept., Ariz. Copper Co., Ltd.,
Box 872, Morenci, Ariz.
MACKAY, ALEXANDER NEIL, Min. Engr., Genl. Supt., Frontino & Bolivia Gold Min.
Co., Ltd., Care Messrs. Tracey Bros., Apartado 184, Barranquilla,
Colombia, So. Amer.
MAGUIRE, HUGH J., Supt., Leaching Plant, Anaconda Copper Min. Co.,
Anaconda, Mont.
MARBLE, EARL R. Chemist, American Smelt. & Ref. Co., Hayden, Ariz.
MARIACA, CARLOS ALIAGA, Min. Engr., Casilla de correo 313,
La Paz, Bolivia, So. Amer.
MARTIN, SYLVESTER E., Asst. Met., Wallaroo & Moonta Min. & Smelt. Co., Ltd.,
Sholl St., Wallaroo, South Australia.
MEISSNER, C. E., Mine Shift Boss, Inspiration Cons. Copper Co., Miami, Ariz.
MELITZER, SAMUEL 246 Rivington St., New York, N. Y.
MINNER, A. R., Mill Supt., Vindicator Cons. Gold Min. Co., Independence, Colo.
MOHLER, KARL IRWIN Min. Engr., Calumet & Arizona Min. Co., Warren, Ariz.
MONTGOMERY, HENRY SCHUYLER, Designer, New Cornelia Copper Co.,
Box 911, Warren, Ariz.
NEILL, WILLIAM A., Chief Mech. Engr., The Dorr Co.,
812 Cooper Bldg., Denver, Colo.
NOLAN, PHILIP E., Mine Engr., Nevada Douglas Cons. Copper Co., Ludwig, Nev.
NOYES, HARRY F., Supt., Blast Furnaces, Broken Hill Proprietary Co., Ltd.,
Newcastle, N. S. W., Australia.
OGDEN, J. EDWARD, Manufacturer, J. Edward Ogden Co., Inc.,
147 Cedar St., New York, N. Y.
OLWAGE, MICHAEL FREDERICK 4122 Grand Boulevard, Chicago, Ill.
OSBORNE, LE ROY D. Asst. Supt., Sterling Borax Co., Lang, Cal.
PARKER, HARRY COOPER, Min. and Met. Engr., Nevada Cons. Copper Co.,
McGill, Nev.

- PARK, JAMES CALDWELL, Supt. Construction, Standard Oil Co.,
1309 Barrett Ave., Richmond, Cal.
- PLUMMER, HAROLD CLEVELAND, Mine Foreman, Cananea Cons. Copper Co.,
Cananea, Son., Mexico.
- PRICE, CHARLES JAMES, Min. Engr. 1430 Polk St., Topeka, Kansas.
- PRINCE, GEORGE WALLACE, Smelter Supt., Cananea Cons. Copper Co.,
Cananea, Son., Mexico.
- PROSSER, WARREN C. Silverton, Colo.
- PUTNAM, HENRY R., Min. Engr., United States Smelt., Ref. & Min. Co.,
Room 3260, 120 Broadway, New York, N. Y.
- REA, JAMES C. Mgr., Oliver Iron & Steel Co., Pittsburgh, Pa.
- REED, DONALD D., Met. Asst., Wallaroo & Moonta Min. & Smelt. Co., Ltd.,
Francis Terrace, Kadina, South Australia.
- REEDER, GEORGE KINGSBURY, Mill Supt., Idaho Continental Min. Co.,
Klockmann, Idaho.
- RISTEDT, ERNEST J., Experimental Work, Inspiration Cons. Copper Co.,
Box 100, Miami, Ariz.
- RETTET, HERMAN H., Asst. Mine Supt., Moctezuma Copper Co.,
Pilares de Nacozari, Sonora, Mexico.
- SHRIVER, ELLSWORTH H., Instructor, School of Mines, Univ. of North Dakota,
Box 1032, Grand Forks, No. Dak.
- SMITH, WARREN S., Economic Geologist, General Chemical Co.,
25 Broad St., New York, N. Y.
- SNEDECOR, P. G., Stope Engr., Old Dominion Copper Min. & Smelt. Co.,
Box 2295, Globe, Ariz.
- SOL, PEDRO L., Genl. Mgr. of Oil Fields, Comodoro Rivadaria, Argentine Republic,
South America.
- STEDMAN, L. W. Mine Mgr., Paragon Cons. Min. Co., Paragon, Idaho.
- STEELE, EUGENE W. Plant Engr., American Smelt. & Ref. Co., Maurer, N. J.
- STEGE, PAUL. Asst. Met., Inspiration Cons. Copper Co., Miami, Ariz.
- STEWART, EDGAR McC. Genl. Supt., Sterling Borax Co., Lang, Cal.
- TALBOT, E. L. Genl. Supt., Daly West Mining Co., Park City, Utah.
- TAYLOR, THOMAS. Supt. of Smelter, United Verde Copper Co., Clarkdale, Ariz.
- THOMSON, ARTHUR T., Asst. Supt., Ohio and Colorado Smelt. & Ref. Co., Salida, Colo.
- VISEL, CLIFTON E. Supt., Mile Wide Copper Co., P. O. Box 426, Tucson, Ariz.
- WALLACE, ROBERT. Midvale, Utah.
- WALTERS, CHARLES W., In office of Consulting Mining Engr., Guggenheim Bros.,
120 Broadway, New York, N. Y.
- WEBSTER, WILLIAM H., Asst. to Genl. Mgr., Copper Queen Cons. Min. Co.,
Douglas, Ariz.
- WELCH, HARRY V., Chemist and Met., Western Precipitation Co., 1016 W. 9th St.,
Los Angeles, Cal.
- WILLIAMS, KENNETH, Shift Supt., Copper Queen Smelter, Copper Queen Cons.
Mining Co., Douglas, Ariz.
- WITTENAU, ERNEST. Testing Engr., Arizona Copper Co., Morenci, Ariz.
- YOUNG, OREL E. Research Fellow, Univ. of Utah, Salt Lake City, Utah.

Associate Members

- BOYLE, JOHN, Min. Engr. 765 South Harvard Boulevard, Los Angeles, Cal.
- BRIGHAM, EDMUND D., Mgr. of Freight Traffic, C. & N. W. Ry.
226 W. Jackson St., Chicago, Ill.
- COPELAND, FREDERICK WINSOR, Sales Engr., Sullivan Machinery Co.,
2006 Railway Exchange Bldg., St. Louis, Mo.
- EVANS, T., Purchasing Agent, Cananea Cons. Copper Co., Cananea, Son., Mexico.
- FRASER, GEORGE C., Counselor-at-Law, Fraser & Speir, 20 Exchange Pl.,
New York, N. Y.
- JOURDIN, W. W., Chief Engr., Joint Power Plant, Inspiration Cons. Copper Co.,
International Smelt. Co., Box 1024, Miami, Ariz.
- SHEDDEN, JOHN S., Sales Engr., The Mine and Smelter Supply Co.,
42 Broadway, New York, N. Y.

Junior Members

- BURNHOLZ, HENRY S., Engr. Min. 455 E. Houston St., New York, N. Y.
- DAVIS, THORNTON. Student, School of Mines, Columbia Univ., New York, N. Y.
- DOLMAN, PHILLIPS BROOKS. Student, Missouri School of Mines, Rolla, Mo

DYNAN, ALAN E.	Student, Lehigh Univ., So. Bethlehem, Pa.
ELFRED, FRANK STILLMAN	P. O. Box 418, Rolla, Mo.
FERNANDEZ, ARTURO C.	Student, Missouri School of Mines, Box 219, Rolla, Mo.
GARRETT, STUART G.	Instructor in Geology, University of Virginia, University, Va.
GILBERT, DONALD C.	Student, Ohio State University, Columbus, Ohio
GOLICK, TONY FRANK	Student, Missouri School of Mines, Rolla, Mo.
HATFIELD, PERCY W.	Student, University of Pittsburgh, Pittsburgh, Pa.
HOLMES, JOSEPH AUSTIN, 2d.	Student, Lehigh Univ., So. Bethlehem, Pa.
JONES, WENDELL THEODORE	Student, Univ. of Nevada, Reno, Nev.
LASK, HAROLD A.	Student, Missouri School of Mines, Rolla, Mo.
LEY, HENRY ALFRED, Geol.,	Petroleum Engr., Univ. of Pittsburgh, 306 State Hall, Pittsburgh, Pa.
LIVINGSTON, WILLIAM S.	Student, Columbia University, New York, N. Y.
LUCKY, MAURICE C., JR.	Student, Missouri School of Mines, Rolla, Mo.
RUKEYSER, WALTER A.	Student, Princeton Fellow to the Columbia School of Mines, Fernald Hall, Columbia University, New York, N. Y.
SHAYES, FRED P.	Student, Missouri School of Mines, Rolla, Mo.
SLOMAN, HAROLD JANDORF	Student, Min. Engrg., Lehigh Univ., 462 Chestnut St., So. Bethlehem, Pa.
SOMMERVILLE, WILLIAM B., JR.	Student School of Mines, Columbia University, New York, N. Y.
WANG, HSIFAN F.	Student, Columbia University, Livingston Hall, New York, N. Y.
YOUNG, GUY K.	Engr., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
Total Membership, Mar. 10, 1917	
6,126	

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

Applications Lacking Endorsement

Applications for membership have been received from Mr. Drewitt, Mr. Lehman and Mr. Trench, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

George Edward Drewitt, Mt. Morgan, Queensland, Aust.

Proposed by

Born 1895-1900, Higher Grade School, Gillingham, England. 1903-09, Electrical and Mechanical Engrg., Gillingham Tech. College (evening). 1909-10, Min. Engrg. course, South Kensington, London, England. 1900-09, Engr. and draftsman, Medway Iron Works, Rochester, England. 1910, Engr., Maschinere-patrik, Lubeck, Germany. 1911-13, Asst., West African Mines. 1913, Reorgan-izing cement works and coal mines and reporting on properties, The Victorian Portland Cement Co., Richmond, Vict., Aust. 1912-13, Reporting, Pacific Phosphate Co., Ocean Island, Gilbert Group, Central Pacific. 1913-14, Reporting on mines and mills and Supt. erection of mills.

Present position: Engrg. Dept., Mt. Morgan Gold Mines Co., Ltd.

Andrew William Lehman, El Nihne, Chagres, Chile.

Proposed by Mark R. Lamb.

Born 1878, Paris, France. 1890-95, Dulwich College, London. 1896-1900, Royal School of Mines, London, where I obtained a Royal Exhibition and later became an Associate of the Royal School of Mines (A. R. S. M.). Also a member of the Inst. of Min. and Met., London. 1900, Entered the firm of Bewick, Moreing & Co., and

went out in their employ to Western Australia, East Africa, Sinai Peninsula and N. Brazil. 1904, Went to W. Africa as Surveyor and Assayer for the Himan Concessions. 1905, Went to Peru for the Huinac Copper Mines. 1906, Came to Chile as Min. Engr., Société des Mines des Cuivre de Catemon.

Present position: Genl. Mgr., Société des Mines de Cuivre de Catemon.

Robert Hamilton Trench, Rangoon, Burma, India.

Proposed by A. W. G. Bleeck, John Cadman.

Born 1880, Liverpool. 1902, Univ. of Cambridge, England, 2d class Mechanical Science. 1902-04, British Westinghouse Electric & Mfg. Co., Trafford Park, Manchester. 1904-08, Asst. Mgr. and Mgr., London & Pacific Petroleum Co. 1908-12, general reporting and executive work on properties in Russia, West Indies, East Indies, Saina, United States, Messrs. Thompson and Hunter, London.

Present position—1912 to date: Genl. Mgr., British Burmah Petroleum Co.

The following persons have been proposed during the period Feb. 10, 1917, to Mar. 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

James P. Anderson, New York, N. Y.

Proposed by Louis D. Huntoon, W. A. Johnson, Herbert B. Cox.

Born 1867, West Springfield, Mass. 1890, B. S., Worcester Polytechnic Institute. 1893-1900, Contracting Eng. 1900-05, Engineer, Merton Lawn Co. 1905-07, Manager, Peter Cooper Glue Factory. 1907 to date, Manager of Mines of Ringwood Co.

Present position: Manager of Mines of Ringwood Co.

Thomas Mellor Bains, Jr., East St. Louis, Ill.

Proposed by Chas. H. Fulton, Charles A. Chase, Philip N. Moore.

Born 1877, Philadelphia, Pa. 1895-96, Cornell University, Electrical course. 1907-11, Columbia Univ., E. M. 1911-12, Mexico, Cuba, British Columbia and Western State Mines—practical experience. 1912, Engineer, American Zinc Co. 1913, Engineer, Mariposa Commercial & Mining Co., Cal. 1914, Charge of mining exploration at Whitlock, Cal. 1915-16, Instructor, Case School of Applied Science.

Present position: With C. H. Fulton Experimental Zinc Plant, East St. Louis, Ill.

Walter J. E. Barcus, New York, N. Y.

Proposed by J. Mark Smith, L. D. Albin, Geo. A. Howells.

Born 1885, Cedar Rapids, Iowa. 1908, Mass. Inst. of Technology, Degree of S. B. Mining and Metallurgy. 1908, Miner, Daly West Mine. 1909, Experimental staff, Garfield Smelting Co. 1909, 8 months operator Nevada Cons. Mill. 1910-11, Chief Experimental Staff, Nevada Cons. Mill. 1911-13, Chief chemist and in charge of mine and drill, Sampling Ex. Co., Chuquicamata, Chile. 1916, Engineer in charge exploration work in Cuba for General Chemical Co.,

Present position—1916 to date: Mining Engineer, General Chemical Co.

Henry Clay Beauchamp, Cumpas, Sonora, Mexico.

Proposed by S. A. Gardanier, W. H. Weed, Arthur Houle.

Born 1871, Baltimore, Md. 1881-84, Public Schools, Baltimore, Md. 1884-89, McDonagh School, Baltimore. 1890-99, Baltimore Copper Smelting & Rolling Co., Baltimore, Md. 1900-01, Bolivia & Chile, investigating copper deposits. 1903-06, Secretary, Transvaal Copper Co. 1907-16, Manager Mines & Smelter, Transvaal Copper Mining Co., Cumpas, Sonora, Mexico.

Present position—1916 to date: Manager, Transvaal Copper Mining Co.

Charles Bocking, Butte, Mont.

Proposed by J. L. Bruce, A. D. Beers, W. L. Creden.

Born 1876, Fulton, Mo. 1905-10, Manager, U. S. Smelting Co., Canon City, Colo. 1911, Manager, Cripple Creek Sampling & Ore Co., Cripple Creek, Colo. 1912, Utah Copper Co., Salt Lake City, Utah. Since 1912 with Butte & Superior Mining Co.

Present position: Asst. Mgr., Butte & Superior Mining Co.

Carl W. Brunskog, Humboldt, Ariz.

Proposed by F. K. Brunton, E. S. Smith, G. M. Colvocoresses.

Born 1885, Chicago, Ill. 1906, Univ. of Arkansas, B. C. E. 1906-12, was employed in various capacities as assayer, draftsman, foreman, Vandalia R. R., C. M. & St. P. R. R., Stone & Webster and Pittsburgh Coal Co. 1912-15, Repath & McGregor, Douglas, Ariz., and Los Angeles, Cal. 1915, Anaconda Copper Min. Co., Anaconda, Mont.

Present position—1915 to date: Draftsman, Cons. Arizona Smelt. Co.

Ossian Cassell Burrell, Mascot, Tenn.

Proposed by J. N. Houser, H. B. Henegar, H. A. Coy.

Born 1884, Fredonia, Wis. High School training, one year Univ. Aug., 1908, to present, with American Zinc, Lead and Smelting Co., at Platteville, Wis., as cashier; at St. Louis, Mo., as statistician; at Mascot, Tenn., as auditor; at Mascot, Tenn., as manager.

Present position: Manager, American Zinc Co. of Tennessee.

Clarence E. Calvert, Butte, Mont.

Proposed by C. W. Goodale, Reno H. Sales, W. B. Daly.

Born 1883, Gratiot, Wis. 1902, Whitewater, Wis., High School. 1912, Colo. School of Mines, Golden, Colo., E. M. 1912-14, Britannia Beach Mining & Milling Co., Britannia Beach, B. C., shoveler, miner, engr. 1914 to date, A. C. M. Co., Butte, Mont., as miner, shift boss, and asst. safety engineer.

Present position: Asst. Safety Engineer, Anaconda Copper Mining Co.

Frederic Edward Carter, Newark, N. J.

Proposed by Wm. Campbell, E. J. Hall, Homer L. Carr.

Born Dublin, Ireland, 1886. 1906-09, Leeds Univ., England. 1909, B. Sc. 1909-11, Technische Hochschule, Karlsruhe, Germany. 1911, Dr. Eng. 1911-12, Asst. Versuchsanstalt, Karlsruhe. 1912-13, W. & R. Jacob & Co., Dublin, Ireland. 1913-16, Dept. of Mines, Ottawa, Canada. 1916, Baker & Co., Inc., Newark, N. J.

Present position: In charge of Physico-Metallurgical Dept. of Baker & Co.'s Platinum Works.

George Albert Chapman, New York, N. Y.

Proposed by Frederick Laist, E. P. Mathewson, A. E. Wiggin.

Born 1881, London, England. 1894-99, Polytechnic, Battersea, London, England, Junior and Intermediate County Scholarships. 1899, Employed by Messrs. Sulman & Picard, metallurgists, London, England and 1903, automatically passed into employ of the Cattermole Ore Concentration, Ltd., and then into Minerals Separation Ltd. 1903-12, Employed by Minerals Separation Ltd. investigating, introducing and operating flotation plants in Australia, Finland, Sweden, Russia, etc. 1912-16, Employed by Minerals Separation American Syndicate Ltd. at Inspiration, Anaconda, Braden (Chile), Cuba and other mining centers. 1917, Employed by Minerals Separation North American Corp.

Present position: Metallurgical engineer with same company.

Herbert Naden Clague, Vancouver Isl., B. C., Canada.

Proposed by Thos. A. Wood, Robert Musgrave, Wm. Brewer.

Born 1883, Sheffield, England. 1890-98, Private study. Sheffield Central High School. 1898-1900, Univ. College, Sheffield, Eng., Engineering & Metallurgy. 1900-02, Engineer's Asst., Rothervale Collieries, Ltd., N. Sheffield, Eng. 1903-06, Apprenticeship, Civil Engineering, Messrs. Swann & Brady, Chapel-en-le-Frith & Stockport, England. 1907-08, Draughtsman and Surveyor, Canadian Pacific Ry. Co., Montreal, Canada. 1908-11, Surveying and General Engineering, Duncan, Vancouver Isl. 1911-16, Private practice as civil engineer, and British Columbia Land Surveyor, B. C., Can.

Present position: Mine Engineering Staff, Granby Cons. Mining Co.

Charles Baker Davis, Birmingham, Ala.

Proposed by Robert Hamilton, Erskine Ramsay, H. S. Geismer.

Born 1875, Ann Arbor, Mich. Ann Arbor High School. 1901, Univ. of Michigan, B. S. C. E., Member, Amer. Soc. of Mech. Engrs. 1893-97, Inspector, Dredge work, Sault Ste Marie Canal. 1897-1901, Engr. Course, Univ. of Mich. 1901, summer, Asst. Chief Engr., D. L. & N. R. R. 1901-02, Dredge Inspector and Transitman, U. S. Engrg. Corps. 1902, summer, Asst. Engr., M. C. R. R. 1902, Instructor, Univ. of Michigan. 1903, Supt. Mines, Allegheny Ore & Iron Co. 1903-07, Div. Engr., Tennessee Coal, Iron and R. R. Co., in charge of opening of new mines and erection of electrical, steam and ventilating machinery and boilers, necessary for the operation of these mines. Plants varying in value from \$13,000 to \$450,000. 1907-11, Asst. Chief Engr., Tennessee Coal, Iron and R. R. Co., engaged in the design and construction of plants of the above character and the making of estimates, specifications and purchase of machinery and material for this work; the last and largest of these projects being the development of a mine and its equipment which cost approximately \$900,000. This mine now has an output of 3,000 tons in ten hours.

Present position—1911 to date: Engineer and Machinery Agent, own business.

Charles Herbert Dickie, Vancouver Isl., B. C., Canada.

Proposed by Thos. A. Wood, Robert Musgrave, Wm. M. Brewer.

Born 1859, Beachville, Ont. Public Schools. 1877, Teacher's Certificate. 1883, Business College, Mich. 1896, Substantially interested in Tyce Mine, B. C., in charge first shaft. 1900-07, Pres. and Man. Dir. Richard III Mine. 1898-1901, Member Mining Committee, B. C. Legislature. 1901, Pres. and Man. Dir., Leach River Mining Co., B. C. 1904, Chairman, Copper Queen Syndicate, B. C. 1908-13, Pres. and Man. Dir., Portland Canal Mining Co., B. C. 1915-17, Pres. and Man. Dir., Georgia River Mining Co., B. C. 1917, Managing Director of a company being formed to exploit copper deposits, Nit Nat, B. C.

Present position: Managing Director, Georgia River Mining Co., Portland Canal & other interests.

• **Walter Dobbins**, Hurley, N. Mex.

Proposed by F. C. Martinez, Ira L. Wright, Frank R. Wicks.

Born 1887, Ipava, Ill. 1905, Champaign, Ill., High School. 1910, B. S., Mo. School of Mines, Rolla, Mo. 1906, Morning Hour Mining Co., Joplin, Mo., miner, 2 months. 1907, Copper Queen Mining Co., Bisbee, Ariz., miner, 2 months. 1909, Bunker Hill & Sullivan Mining Co., Wardner, Idaho, miner, 6 months. 1910, Utah Copper Co., Garfield, Utah. 1911, Operating Classifiers & vanners, 15 months. 1911, Chino Copper Co., Hurley, N. Mex. 1911, Mill Foreman, 2 months. Fine Crushing Foreman, 4 years. Concentrating Foreman, 2 months. Classifier Foreman, 1½ years.

Present position: Foreman, Fine Crushing Dept., Chino Copper Co.

Edward English, Lead, S. Dak.

Proposed by A. J. M. Ross, R. M. Mackey, B. C. Yates.

Born 1887, Sheffield, Ia. Graded School. 1908, Special course, Wisconsin State Normal, Platteville, Wis. 1914, Grad., Wisconsin State School of Mines, Platteville, Wis. 1907, St. Rose Min. Co., Platteville, Wis. 1910-11, Vinegar Hill, Lead & Zinc Min. Co., Galena, Ill.

Present position—1914 to date: Sampler, Homestake Mining Co.

• **Robert Wesley Erwin**, Wankon, Iowa.

Proposed by D. M. Barringer, Ledyard Heckscher, Frank C. Roberts.

Born 1872, Tapan, Ohio. 1898-99, Bay City High School. 1900-01, Westminster College. 1893-94, Utah State College, B. S. 1892, Special work chemistry, University Mo. 1893, Utah Experiment Station, Asst. Chemist. 1896-98, Granite City Steel Co., Asst. Chemist, Night Supt. 1898-1900, American Steel Foundry Co., Chief Chemist. 1900-02, Assistant Supt. 1902-06, Sligo Furnace Co., Chemical Engineer & Supt., By-Product Plant. 1906-16, Missouri Iron Co., Supt. 1916, Mississippi Valley Iron Co., Superintendent Mines & Concentrating Plant.

Present position: Mississippi Valley Iron Co., Supt. Mines & Concentrating Plant.

Alex. L. Feild, Pittsburgh, Pa.

Proposed by F. G. Cottrell, Geo. S. Rice, Chas. H. MacDowell.

Born 1890, Oxford, N. C. 1911, A. B., University of N. C., Chapel Hill, N. C.

1914, M. S. in chemistry, N. C. Agricultural & Mechanical College, Raleigh, N. C. 1912 (summer) Research student, Physical chemistry, Univ. of N. C. 1911-12, Science Teacher, Raleigh High School. 1912-13, Science Teacher, Salem College, Winston, Salem, N. C. 1913-14, Asst. Chemist, N. C. Agricultural Experiment Station. 1914-16, Junior physical chemist, U. S. Bureau of Mines, Pittsburgh. Present position—1916 to date: Asst. Metallurgist, U. S. Bureau of Mines.

John Murray Fox, Dover, N. J.

Proposed by H. P. Henderson, R. M. Roche, Edwin O. Holter.

Born 1880, Detroit, Mich. 1903, B. S., in Mining & Metallurgy, Howard University. 1904-06, Mining Engineer with Cleveland-Cliffs Iron Co., Ishpeming, Mich. 1906-10, Asst. Supt., Tonopah Mining Co. of Nevada, Tonopah, Nev. 1910-12, Mgr., Hot Creek Dev. Co., Rattlesnake, Nev. 1912-13, Field Engineer, West End Con. Min. Co., Tonopah, Nev. 1913-16, Field Engineer, Tonopah Belmont Dev. Co., Tonopah, Nev. 1917, Supt., Beach Glen Mine, Beach Glen, N. J.

Present position: Supt. Beach Glen Mine.

Arthur Lawrence Gholz, Crystal Falls, Mich.

Proposed by Philip N. Moore, Percy W. Donovan, R. D. Longyear.

Born 1876, Brown County, Minn. 1901, Graduated Minn. School of Mines, Degree of Mining Engineer. 1901, Millman, Forest Hill Mining Co., Colorado. 1902, Rocky Mt. Smelter, Florence, Colo. 1902-11, With E. J. Longyear on Mesaba Range Mine, as Engineer and Supt. of Explorations. 1911-13, Representing E. J. Longyear Co. of Minneapolis, Minn. at Birmingham, Ala. and Tucson, Ariz. 1913 to date, Representing E. J. Longyear Co. in Michigan—Wisconsin district.

Present position: Representative of E. J. Longyear Co.

Julius H. Gillis, Toronto, Ont.

Proposed by E. P. Mathewson, F. J. Brulé, W. A. Carlyle.

Born 1881, Fostoria, Ohio. Common School 1901-12, four years railroad survey and construction instrument-man. Resident engineer and Asst. Engr., Minneapolis, St. Paul & Sault St. Marie Ry., Great Northern R. R., Can. Northern R. R. Six years structural drafting, detailing, checking, designing & squad foreman, bridges and buildings, Minneapolis Steel and Machinery Co., Toltz Engineering Co., St. Paul, Minn. One year Erection Foreman, Structural Steel, Minneapolis Steel and Machinery Co. 1912-17, Engineering Dept., Anaconda Copper Mining Co.; charge of drafting for copper electrolytic and zinc plant asst., chief draftsman last year there.

Present position: Chief Draftsman, British America Nickel Corp. Ltd.

Cecil Arthur Gore-Langton, Stockton, Cal.

Proposed by Ed. H. Benjamin, C. E. Grunsky, Jr., T. A. Rickard.

Born 1882, York, England. 1891-94, Brighton College Preparatory. 1894-98, Eton College. 1898-99, Oriel College, Oxford University. 1899-1902, University College, London, B. Sc. E. 1902-04, Roads & R. R. on Indo-China frontier, bridges, 4 tunnels (2,800 ft.), water-supply. 1904-06, Chin Hills Coal & Mining Co., building mine works, breaker-plant, aerial tramway (3 miles) 3 ft. gage R. R., 32 miles. Designed and built all head-works. 1906-07, Surveyed and built 180 miles pipe line, pump-stations each 10 miles, Burma Oil Co. 1907, Draftsman, N. Y. C. H. R. R. R., 6 months. 1907-08, Building R. R., Santa Fe Railway. 1908-09, Investigating accidents Joplin, Mo., district for Detra Insurance Co. 1909-11, In Philippines, built R. R. 27 miles, sugar mill, water plant, sewerage system. Tunnel surveys, foreman and drill, boss Los Angeles Aqueduct. 1913-15, Central America, dams, R. R.'s 2 tunnels, rock crusher and quarry plant. El Hormiguers mines, Salvador, rebuilt 25 stamp mill and cyanide plant, then shift boss 1916, Draftsman on structural design, Santa Fe Railway, Amarillo, Tex. 1916 to date, Designing and building extra unit to plant making soda and soda ash, near Owen's Lake, Cal.

Present position: Engineer, erecting plant, Natural Soda Products Co.

Lafayette G. Johnson, Flat River, Mo.

Proposed by H. G. Washburn, Geo. W. Loddewig, H. A. Guess.

Born 1882, Silvertown, Colo. 1888-96, Public Schools, Denver, Colo. 1896-99, Manual Training High School, Denver, Colo. 1900-04, Colorado School of Mines, Golden, Colo., Degree of Mining Engineer. 1904-05, Employed by Portland Gold Mining Co., Victor, Colo. 1905 to date, Federal Lead Co., at Flat River and Fredericktown, Mo.

Present position: Supt. of Mines, Federal Lead Co.

John Johnston, St. Louis, Mo.

Proposed by Wm. H. Shearman, Thomas Read, Geo. C. Stone.

Born 1881, Perth, Scotland. Up to 1897, Perth Academy. 1897-1903, University College, Dundee. 1903, Degree of Bachelor of Science, Univ. of St. Andrews, London. Associate of the Inst. of Chemistry, London. 1903-05, Engaged on Research. 1905-07, University of Breslau, Germany. 1907-08, Research Laboratory, Mass. Inst. Tech., Boston. 1908, Degree of Doctor of Science, University of St. Andrews, Scotland. 1908-16, On staff of Geophysical Laboratory, Carnegie Institute, Wash., engaged in research work the results of which have been published in over 20 papers in various certified journals. 1909-12, Sub-editor of *Chemical Contracts*. 1915, On editorial board of the *Journal* of the American Chemical Society.

Present position: Research Department, American Zinc, Lead & Smelting Co.

Charles Brace Kelly, Youngstown, Ohio.

Proposed by W. D. B. Motter, Jr., David Dows, A. M. Cummings.

Born 1885, Grand Rapids, Mich. 1904-07, B. S., E. M., Michigan College of Mines, Houghton, Mich. 1910-12, Canada Iron Mines Ltd. as Supt. of Concentrator. 1913, Supt. Limited Ore Co., N. Y. 1913-17, District Manager, American Sintering Co., Youngstown, Ohio.

Present position: District Manager, American Sintering Co.

Edwin H. Kennard, Los Angeles, Cal.

Proposed by H. W. Hardinge, W. G. Swart, R. B. T. Kiliani.

Born 1876, Illinois. Denver High School and Special Course in Engineering. 1894-96, Ontario and Daly Mines, Park City, Utah. 1897-99, Ibex Mining Co., Leadville, Colo. 1902, Cananea Copper Co., Sonora, Mexico. 1905-07, Goldfield, Nevada Junito Leasing Co. 1909-11, Arizona Belmont, Silver Lead, Ariz.

Present position: President and Manager, Oatman Rand Gold Mining Co.

Gerald Joseph Kennedy, Tofo, La Higuera, Coquimbo, Chile.

Proposed by C. A. Buck, W. L. Cumings, H. Merryweather.

Born 1881, Easton, Pa. 1897, Graduated Easton High School, Easton, Pa. 1901, Graduated Lafayette College, Easton, Pa., Degree of Engineer of Mines. 1901-03, Transitman Central Railroad of New Jersey. 1903-04, Engineer of construction and underground, Whitebreast Fuel Co., Chicago. 1904-06, Asst. Supt., Seaboard Coal Iron and Railway Co., Talladega, Ala. 1906-09, Engineer, El Oro Mining and Railway company, El Oro, Mexico. 1909-13, Division Engineer and Asst. Engr., Maintenance of Way, National Railways of Mexico. 1913-14, Locating engineer and chief engineer of construction, Ferro carril de Acambaro, a Queretaro, Queretaro, Mexico.

Present position: General Supt., Bethlehem Chile Iron Mines Co.

Harold de Saulles Kennedy, Anaconda, Mont.

Proposed by Selden S. Rodgers, Harry S. Ware, Bayard S. Morrow.

Born 1886, Uniontown, Pa. 1907, A. B., Washington and Jefferson. 1909-10, Two years of post-graduate work at Lehigh Univ. in chemistry, metallurgy, geology and mining. 1907-08, American Mining Co., Monte Cristo, Wash. 1910-11, Tonopah Mining Co., Tonopah, Nev., in all depts. of mill under A. R. Parsons. 1912, Cyanide Supt. of Jumbo extension mill, Bonnie Clare, Nev. 1913, Metallurgist, Tonopah Belmont Dev. Co.

Present position: Operator, Helper, Zinc concentrator, Anaconda Copper Mining Co.

William Walter Kiskaddon, Tulsa, Okla.

Proposed by M. M. Valerius, V. H. McNutt, F. Julius Fohs.

Born 1891, Bellevue, Pa. 1908-13, General education, Grove City College, Grove City, Pa., Ph. B. 1913-15, Missouri School of Mines. Civil and Min. Engrg. work, L. E. Burnside & Co., Grove City, Pa. Petroleum Geol., Kiskaddon Bros. & Priddy, Oil Prod.

Present position—1916 to date: Geol., Kiskaddon Bros. and Priddy.

A. Sonnin Krebs, Newport, Del.

Proposed by John V. N. Dorr, H. N. Spicer, Etienne A. Ritter.

Born 1887, Nykjobing, Denmark. 1885-95, Friends School, Wilmington, Del. 1896, Sibley College. 1900, Cornell University, M. E. Also did extra work in chemistry. Sept., 1900-June, 1901, Was employed in Delaware Chemical Co., Wilmington, Del. July, 1901, to date, The Krebs Pigment & Chemical Co., Newport, Del. (Manufacturers of Lithopone).

Present position: Second Vice President, Secretary & Treasurer of the Krebs Pigment & Chemical Co.

Samuel Ward Laughlin, Ely, Nev.

Proposed by A. G. Marsh, G. C. Riser, Jr., C. E. Pickett.

Born 1874, Tekamah, Nebr. 1885-92, Public School, Rockville, Mo. 1892-95, Business and Normal Course, Chillicothe Normal, Chillicothe, Mo. 1906-10, Colorado School of Mines, Golden, Colo., degree of E. M. 1908, Shift boss, Liberty Bell Gold Mining Co., Telluride, Colo. 1910, Shift boss and amalgamator, Gold Pioneer Mining Co., Telluride, Colo. 1910-13, Mining engineer, Boston & Montana Dept. of Anaconda Copper Mining Co., Butte, Mont. 1913-14, Shift boss, Leonard Mine, Anaconda Mining Co. 1914-15, Private practice-Surveying, reporting, etc. 1915-16, Engineer, Eagle Mining & Milling Co., Gilman, Colo.

Present position: Flotation Dept., Nevada Consolidated Copper Co.

H. H. Lavery, Fort William, Ontario, Can.

Proposed by Julius M. Cohen, Eugene L. Steindler, Samuel W. Cohen.

Born 1887, Hamilton, Ont., Canada. 1902, Central St. School, Evanston, Ill. 1902-06, Northwestern Academy, Evanston, Ill. 1906-09, Michigan College of Mines, Houghton, Mich., B. Sc. and E. M. 1910, Engr., Keane Wonder Min. Co., Death Valley, Cal. 1910, Asst. Engr., Mass. Cons. Copper Co. 1911, Supt., Monarch Mine, Field, B. C. 1912, Engr., B. C. Copper Co., Princeton, B. C. 1913, Shift Boss, Inspiration Cons. Copper Co. 1914, Engr., Peru Mines, El Callao, Venezuela, S. Amer. 1914-16, Engr. to Shaft Foreman, Dome Mines, Ltd.

Present position: Supt., Thunder Min. Co., Ltd.

Horace M. McFarland, Birmingham, Ala.

Proposed by Robert Hamilton, G. B. McCormack, Erskine Ramsay.

Born 1880, Sabina, Ohio. 1902, Graduated Ohio State University, Columbus, Ohio, as Engineer of Mines. 1913, Received Certificate for First Class Mine Foreman, Alabama. 1902, C. S. & H. and Pennsylvania R. R., Assistant Engineer, Columbus, Ohio. 1903, Dayton Coal & Iron Co., Asst. Engr., Dayton, Tenn. 1903, Tenn. Coal, Iron & R. R. Co., Assistant Engineer Mines, Birmingham, Ala. 1904, Jeffrey Mfg. Co., Draftsman, Columbus, Ohio. 1905, Tenn. Coal, Iron & R. R. Co. Asst. Engr. of Mines, Johns, Ala. 1906-09, Tenn. Coal, Iron & R. R. Co., Asst. Div. Engr. of Mines, Ensley, Ala. 1910-11-12, Tenn. Coal, Iron & R. R. Co. Asst. Supt., Ensley, Ala. 1913, Asst. to Consulting Engr., Birmingham, Ala. 1914-15, Jeffrey Mfg. Co., Asst. Mgr., Birmingham office, Birmingham, Ala. 1915-16, Montwalla Mining Co., Supt., Aldrich, Ala. 1916, Opened office, Birmingham, as Mining Engr.

Present position—1916 to date: Mining Engineer; Consulting, Mining and Civil Work.

Dentaro Matsuzawa, Nishiyama, Echigo, Japan.

Proposed by Hisahiro Naito, Wataru Watanabe, Benzo Katsura.

Born 1887, Shinano, Japan. 1912, Graduated Mining Dept., Tokyo Imperial University. 1912-16, Mining Engineer in Main Office of Nippon Oil Co. 1917 to date, Supt. of Nishiyama Oil-field of Nippon Oil Co., at Echigo in Japan.

Present position: Supt. of Nishiyama Oil-field of Nippon Oil Co.

John G. Menke, Rico, Colo.

Proposed by L. V. Emanuel, H. H. Utley, R. B. Earling.

Born 1893, Neola, Iowa. 1913, Graduated Neola High School. July, 1913, Entered Colo. School of Mines (Summer school). Sept., 1913, Entered Colo. School of Mines. 1913-14-15-16, Attended Colo. School of Mines. Lack of finances prevented my return to complete the course. May, 1911-Oct., 1911, With "Great Matchless Coal Co." Louisville, Colo. Employed in succession as: mucker, miner, machine-man and cager. May, 1914-Sept., 1914, With Portland Gold Mining Co., Victor, Colo., as mucker and trammer. May, 1916, to Sept., 1916, With Arizona Copper Co., Metcalf, Ariz., as chemist and assayer. Sept., 1916, to present, With Cadiz Mining Co., office at Denver, as Supt.

Present position: Supt., The Cadiz Mining Co.

Nelson Newton Nichols, Scranton, Pa.

Proposed by H. G. Davis, H. M. Warren, William W. Inglis.

Born 1876, Clarks Summit, Pa. 1882-92, Public Schools. 1892-96, Keystone Academy and Wyoming Seminary. 1896-1900, Correspondence School courses.

1900 to date, home study and practical mining work. 1902 to date, With D. L. & W. R. R. Coal Dept. mining engineers, beginning as chairman, advancing to transitman, Dist. Engineer and Chief Mining Engineer of D. L. & W. R. R. Coal Dept.
Present position: Chief Mining Engineer, D. L. & W. R. R. Co., Coal Dept.

Harry Hackett Nowlan, Tulsa, Okla.

Proposed by M. M. Valerius, E. Barth, F. Julius Fohs.

Born 1888, Cheyenne, Wyo. 1913, Missouri School of Mines, B. S. in M. E. 1907-09, Wyoming Land and Irrigation Co., Basin, Wyo. 1913-14, Miami Copper Co.

Present position—1915 to date: Geol., Valerius, McNutt & Hughes.

Randall Holden Ormsbee, Christmas, Ariz.

Proposed by S. H. Sherman, Kuno Doerr, H. F. Easter.

Born 1893, Tracy City, Tenn. 1900-06, Public Schools, El Paso, Tex. 1906-11, Kingsley School, Essex Fells, N. J. 1911-15, Columbia Univ., School of Mines. N. Y. C. Sept., 1915, to date, Mine engineer, Christmas Mine, American Smelting & Refining Co., Christmas, Ariz.

Present position: Mine Engineer, Christmas Mine, American Smelting & Refining Co.

Robert Newton Palmer, Worthington, Ontario, Canada.

Proposed by C. V. Corless, O. Hall, J. R. Thoenen.

Born 1882, Norwich, Ontario. 1892-97, Public School Education, North Norwich Township. 1897-1902, High School Education, Norwich Continuation Class School, Brantford & Owen Sound Collegiate Institute. 1903, Teacher's Professional Training at Woodstock Model School. 1907-10, Technical training, mining, at School of Mining, Kingston. 1911, Technical training, mining, Mich. College of Mines. 1909, Mining and mechanical work, Wittlauffer, Lorraine Silver Mines, Temiscaming Mining Co., Canadian Copper Co., Cream Hill Mine. 1910-13, Lake Superior Iron & Steel, Sault St. Marie, Ontario, Helen & Magpie mines. Summer 1910-June, 1911-12, Prospector Magpie Mine & Engr. Helen Mine. 1912-13, Shift Boss, Helen mine.

Present position—May, 1913, to date: Supt. Worthington Mine, The Mond Nickel Co., Ltd.

Dorsey Julian Parker, Pittsburgh, Pa.

Proposed by George S. Rice, O. P. Hood, F. G. Cottrell.

Born 1877, Roberts, Ala. 1896, Completed High School. 1901, B. S. in C. E., Alabama Polytechnic Inst. 1907, C. E., Alabama Polytechnic Institute. 1911-12, Special work, Harvard University. 1901-03, Asst. Division Engr. (mining) Tenn. Coal, Iron & R. R. Co. 1903-07, Chief mining Engr., Galloway Coal Co. 1907-10, Assistant Gen. Supt., Galloway Coal Co. 1910-11, Supt. Corona Coal & Iron Co. (3 mines). 1912-14, Gen. Supt., American Coal Corporation. 1914-16, Mining Engineer, United States Bureau of Mines. 1916 to date, Mine Safety Engineer, U. S. Bureau of Mines

Present position: Mine Safety Engineer.

Clarence Thomas Patterson, Chrome, N. J.

Proposed by Fred Crabtree, George S. Blair, Chas. R. Fettke.

Born 1890, Saline County, Mo. 1895-1905, Common School. 1905-07, Normal School. 1909, Naval Electrical School. 1911-15, Carnegie Institute of Technology, B. S. in Met. Eng. 1915-16, employed by W. S. Kuhn & Liberty Powder Co., direction of Dr. Chas. E. Munroe, Research. 1916, Michigan Steel Casting Co., electric furnace operator. 1916-17, American Smokeless Powder Co., construction engineer and acting superintendent. 1917, U. S. Metals Refining Co., chemist.

Present position: U. S. Metals Refining Co.

William Julius Plumbhof, Frisco, Utah.

Proposed by E. R. Lidvall, Ernest Gayford, Jas. A. Block.

Born 1878, Eureka, Nev. 1902-04, Bingham Smelter, Midvale, Utah. 1904-05, Copperton Mill, Bingham, Utah, charge of Laboratory. 1905-08, Had charge of laboratory, bookkeeper and timekeeper, Boston Consolidated Mine, Bingham, Utah. 1909-10, Had charge of the sampling mill, Steptoe Smelter, McGill, Nev. 1913-14, engineer and chemist at the Ophir Hill Mine, Ophir, Utah. Feb.-June, 1914, engineer in charge of the construction of the United States Fire Clay Silica Brick Plant.

Present position: Caldo Mill, Frisco, Utah, charge of laboratory.

Frederick Byron Plummer, Tulsa, Okla.

Proposed by R. A. Conkling, Dorsey Hager, I. Perrine.

Born 1887, Hanover, N. H. 1905-09, Dartmouth College, B. S. 1909-11, University of Chicago—Chemistry. 1913-15, University of Chicago—Geology. 1912, New Hampshire State Geologist. 1913, Geological Survey, South Western Colorado. 1914 (Summer), Chief of Party, Wisconsin State Geological Survey. 1915, Geologist, Roxana Petroleum Co., working in Kansas, Oklahoma, and Texas.

Present position: Geologist, Roxana Petroleum Co.

John Humphreys Ramage, Bloomfield, N. J.

Proposed by Frank R. Van Horn, J. Burns Read, Zay Jeffries.

Born 1891, Cleveland, Ohio. 1898-1906, Cleveland Public Schools. 1906-10, Glenville High School. 1911-15, Case School of Applied Science, Cleveland, B. S. in Metallurgy. 1910-11, National Carbon Co., Lakewood, Ohio, Laboratory Asst. 1915-16, National Transit Co., Oil City, Pa., Foundry Chemist. 1917, Westinghouse Lamp Co., Bloomfield, N. J., Metallurgist.

Present position: Metallurgist, Westinghouse Lamp Co.

Elliott Eugene Reyer, El Paso, Tex.

Proposed by S. F. Shaw, W. M. Drury, R. F. Manahan.

Born 1884, New York City. 1892-98, Colorado Springs Public Schools. 1898-1902, Colo. Springs High School. 1902-06, Colo. College, Colo. Springs. Colo., Graduated with degree B. S. C. E. July-Oct, 1906, Taylor & Brunton Sampling Co., Cripple Creek, Colo., office work. Oct.-Nov., 1906, Colo. Midland Ry., Maintenance work. 1906-07, Central Colo. Power Co., Kremling, Colo., general surveying, tunnels, dam sites, etc. 1907 to date, American Smelting & Refining Co. and allied companies. During the period July, 1907, to date, various branches of mining in Mexico at various properties of the A. S. & R. Co., such as smelter representative, assayer, engineer at mine, cashier, asst. supt. and supt.

Present position: Supt. Matehuala Unit., American Smelting & Refining Co.

Joseph Murray Riddell, Iron River, Mich.

Proposed by Andre Formis, Chas. E. Lawrence, Felix A. Vogel.

Born 1888, Pasadena, Cal. 1904-09, Lewis Institute, Chicago, Ill., preparatory and two years college. 1909-11, Michigan College of Mines, Houghton, Mich., degrees B. S. and E. M. 1911, Asst. Instructor, Mining & Civil Engineering Dept., Mich. College of Mines. 1911-12, Engineer, Mass Consolidated Mining Co., Mass, Mich. 1912-13, Construction Engineer, Florence Iron Co., Florence, Wis. 1913-16, Mining Engineer, Bates Iron Co., Iron River, Mich.

Present position: Chief Engineer for Florence and Bates Iron Companies.

Wm. F. Rochow, Pittsburgh, Pa.

Proposed by J. S. Unger, Kenneth Seaver, E. G. Spilsbury.

Born 1889, Columbo, Pa. 1912, Chemist, Pennsylvania State College. 1912-17, Research Dept., Harbison-Walker Ref. Co.

Present position: Research Dept., Harbison-Walker Ref. Co.

John Christian Rodder, Stockton, Cal.

Proposed by Tasker L. Oddie, G. McM. Ross, Ernest A. Hersam.

Born 1882, Germany. 1903, Grad. College of Mining, Univ. of Cal., B. S. 1903-05, Underground work and mill work in Royal Cons. mine, Calaveras Co., Bunker Hill Amador Co., and Confidence mine, Confidence, Cal. 1905, Asst. foreman Ariz-Pyramid Mg. Co. (8 months), Mohave Co., Ariz. 1906-07, Supt. Stockton Ridge Gravel Mine at Mokelumne Hill, Cal. 1908-09, Independent examination work. 1909-15, Supt., Grace Darling Mg. Co., Hawthorne, Nev. 1915-16, Independent examination work. 1916, Eng. Atkins Kroll Co., Sodaville, Nev.

Present position: Am at present engaged in examination work and not directly in the employ of any mining corporation.

Harold Hastings Sanderson, Seattle, Wash.

Proposed by M. C. Butler, W. H. Linney, L. K. Armstrong.

Born 1881, Philippston, Mass. 1896-99, Colorado College. 1899-1901, Division Engineer, El Paso & North-Eastern Ry. 1901-07, Division and Chief Engineer, Northern Coal & Coke Co. Denver, Colo. 1907-13, Consulting Engineer, Trinidad, Colo. 1914, Mine Supt., Consolidated Fuel Co., Hiawatha, Utah.

Present position: Mine Safety Engineer.

Walter M. Small, Tulsa, Okla.

Proposed by M. M. Valerius, E. Barth, G. H. Cox.

Born 1887, Cooperstown, Pa. 1911, Allegheny College, Meadville, Pa. Degree of A. B. 1911-12, Science Teacher in Miles City, Mont. 1913, Northern Pacific Land Survey. 1914, Consulting Geologist. 1915, Geological Staff of Producers Oil Co., Houston, Tex. 1916, Consulting Geologist.

Present position is that of Consulting Geologist, though I represent the Mid-Continental Development Co. of 30 East 42d Street, New York City.

Frank Griswold Smith, Waterbury, Conn.

Proposed by Wm. H. Bassett, A. Clayton Clark, Frank L. Antisell.

Born 1885, Torrington, Conn. 1911, S. B. Mining Eng., Mass. Inst. of Tech. 1904 to date, With the Technical Dept. of the American Brass Co., metallography and physical metallurgy of non-ferrous alloys.

Present position: Technical Dept. of the American Brass Co.

Stuart St. Clair, Rolla, Mo.

Proposed by Philip N. Moore, H. A. Buehler, F. W. DeWolf.

Born 1884, Gainesville, Fla. 1901, Preparatory School. 1907-12, Pennsylvania State College, degrees of B. S. and E. M. 1912-13, Univ. of Iowa, degree of M. S. in Geol. June-Nov., 1912, U. S. Geological Survey. 1912-13, Graduate student and asst. Dept. Geology, Univ. of Iowa. 1913-17, Alternately with U. S. Geological Survey, Missouri Bureau of Geology and Mines and Illinois State Geological Survey on general geological, oil, and mining work.

Present position: Geologist, Missouri Bureau of Geology and Mines; Asst. Geologist, U. S. Geological Survey.

Rudolph John Stengl, Joplin, Mo.

Proposed by S. H. Davis, G. B. Corless, L. C. Church.

Born 1890, Antigo, Wis. 1904, Antigo Grammar School. 1908, Antigo High School. 1913, Univ. of Wis., Mining Engineering Course, B. S. Degree. 1911, Laborer. Various occupations at Old Jordan Mine, Bingham Canyon, Utah. 1912, Machine-man, B. & C. Mine, Shullsburg, Wis. Employed by Vinegar Hill Zinc Co. 1913, Machine-man—Some bookkeeping—Ellsworth Mine, Livingston, Wis. 1913-14-15, Vinegar Hill Zinc Co.: Foreman Century Mine, Neck City, Mo.; Supt. Vinegar Hill Mine.

Present position: Supt. and Field-man, Century Zinc Co.

Frank M. Stone, Park City, Utah.

Proposed by G. S. Krueger, O. W. Friendly, James Humes.

Born 1892, Sonora, Cal. 1908, Unfinished course in Electrical Engineering with Heald's of San Francisco. 1910, Course in complete mathematics with American Correspondence Schools. 5 years with Calavera Copper Co., Copperopolis, Cal., in mechanical and electrical departments. 3 years with Silver King Coalition Mines Co., Park City, in charge of mechanical and electrical departments and with the above company at present.

Present position: Master Mechanic with Silver King Coalition Mines Co.

Galen L. Stone, Boston, Mass.

Proposed by D. C. Jackling, E. E. Olcott, L. D. Ricketts.

Born 1862, Leominster, Mass. 1878, English High School, Boston, Mass. Director of the following companies: American Agricultural Chemical Co., Atlantic, Gulf & West Indies S. S. Lines, Clyde Steamship Co., Eastern Steamship Co., Mallory Steamship Co., N. Y. & Cuba Mail S. S. Co., N. Y. & Porto Rico S. S. Co.

Present position: With Hayden, Stone & Co.

Forest John Sur, San Francisco, Cal.

Proposed by Thor Warner, M. H. Hotchkin, W. W. Case.

Born 1880, Wheatland, Cal. 1895-96, Tool boy, Penn Mine, Yuba Co., Cal.; Miner, Penn Mine, Yuba Co., Cal. 1897-98, Miners' Assay Office and School, S. F. 1899, Van der Naillens Mining School. 1900, Assayer, Gold Bug Mine, Ore. 1901, Nome, Alaska—prospecting. 1902-06, Mine Supt., Anglo-Chilean Exp. Co., Mina Portazuela, Chile, Mohawk Leasing Syndicate, Nev. 1907-08, Mining In-

spections. 1909, Pyramid Oil Co., Taft, Cal. 1910, Cons. Willon Min. Co., Cal. 1911, H. A. Rispin County Oil Co., Cal. 1912, Traveled, studying N. A. oil fields. 1913-14, Western Pacific Oil Co., Calgary, and general inspections. 1915, Inspections Louisiana, Cal., Can. Have general petroleum engineering practice. 1916, Wyoming Montana Dev. Co. T. S. F. Cal. (a producing company).

Present position: Con. Engr., Wyoming Montana Dev. Co.

Henry Traphagen, Weehawken, N. J.

Proposed by E. M. Johnson, P. deP. Ricketts, Bradley Stoughton.

Born 1887, New York. 1913, Cooper Institute, B. S. 1907-14, I. G. Johnson & Co., New York City. 1914-15, Reading Bayonne Steel Casting Co., Reading, Pa., Chief Chemist. 1915-16, Nova Scotia Steel & Coal Co., New Glasgow, N. S., Metallurgist. Hewitt Steel Corporation, Newark. Installed laboratory and standardized procedure.

Present position: Chief Chemist and Metallurgist, High Speed Tools Co., Toledo, Ohio.

Ralph Angelo Watson, Humboldt, Ariz.

Proposed by F. K. Brunton, G. M. Colvocoresses, E. S. Smith.

Born 1885, Champaign, Ill. 1899, High School, Calumet, Mich. Training School, Calumet, Mich. 1903, University of Ill., Urbana, Ill. 1906, Stanford University, Palo Alto, Cal. 1907, Armour Institute of Technology, Chicago, Ill. 1897, Guggenheim Smelting Co., Aguascalientes, Mexico (2 years). 1904, American Smelting & Refining Co. succ. to above (1 year). 1905, Mexican R. R. Co. Aguascalientes, Mexico (1 year). 1907, The Tezuitlan Copper Co. Tezuitlan, Pue, Mexico, 2 years. 1909, Mexican Light & Power Co., Mexico City, Mexico, (1 year). 1910, Tezuitlan Copper Co. (2 years). 1912, Arizona Copper Co. 1912, Repath & McGregor. 1913, Consolidated Arizona Smelting Co., Humboldt, Ariz. (3 years).

Present position: Chief Engineer, Consolidated Arizona Smelting Co.

Associate Members

Alfred W. Dodd, New York, N. Y.

Proposed by Wm. H. Shearman, Thos. Read, Geo. C. Stone.

Born 1864, Hartford, Conn. Hartford High School. 1897-1915, New Jersey Zinc Co.

Present position: Manager of Sales, American Zinc Lead & Smelting Co.

Walter Charles Schmidt, Humboldt, Ariz.

Proposed by F. K. Brunton, G. M. Colvocoresses, E. S. Smith.

Born 1891, New Britain, Conn. To 1910, Public Schools, New Britain. 1910-13, 1914-15, Sheffield Scientific School, Yale Univ., Ph. B. 1913-14, Pratt Business College, New York City. 1915, Civic Secretary to E. A. Filene, Boston, Mass. 1916, Estimating Engr., John Kunz Co. Contractors, New Britain, Conn.

Present position: Townsite Engr., Cons. Arizona Copper Co.

Ynau Tze Tsai, Shanghai, China.

Proposed by Robert Peele, Wm. Campbell, Thomas Read.

Born 1887, Huchow, China. 1906, Tien Tsin Univ., Tien Tsin, China, B. A. 1910, M. I. I., Boston, Mass., S. B. 1912, Columbia Univ. N. Y. C., M. A. 1913, Wisconsin Univ., Madison, Wis., Commerical course. 1913-15, Mining Engineer, Penn. Coal Co., Dunmore, Pa. 1915-16, Valuation Engineer with Mr. W. J. Wilgus, N. Y. C., & Mr. W. A. Cochran, Pottsville, Pa.

Present position: Will give definite information after reaching China.

Junior Members

Charles Albi, Golden, Colo.

Proposed by Harry Wolf, J. C. Roberts, Geo. J. Young.

Born 1893, Grimaldi, Italy. 1907-10, Denver Grammar Schools. 1910-14, East Denver High School, Denver, Colo. 1914, Colorado School of Mines,

Present position: A junior at the Colorado School of Mines.

Clarence Ward Andrews, Wellsville, Ohio.

Proposed by Fred Crabtree, Chas. R. Fetteke, Joseph Jensen.

Born 1891, Welsford, Kans. 1909, Graduated from Wellsville, Ohio, High School. 1913-17, Carnegie Institute of Technology, Metallurgical Engineering. 1909, Bookkeeper, Vulcan Clay & Brick Co., Wellsville, Ohio. 1910-11, Supervisor's Clerk, Penna. Lines, Wellsville, Ohio. 1912-13, Bookkeeper and Cashier, The Riggs Co., East Liverpool, Ohio. Summers of 1914 and 1915, Bookkeeper, The Riggs Co., East Liverpool, Ohio. Summer of 1916, Section laborer and Supervisor's Clerk, Penna. Lines, Wellsville, Ohio.

Present position: Senior Student in Metallurgy, Carnegie Institute of Technology.

John Raymond Bryan, University of Nevada.

Proposed by Chas. F. Williams, Francis C. Lincoln, J. C. Jones.

Born 1893, Quincy, Ill. 1913-15, Colorado School of Mines. 1915, University of Nevada, Mackay School of Mines. Practical Mining: Dec., 1916, Anderson Mine, Luning, Nev. May-June, 1916, Crown Point-Belcher Mines, Gold Hill, Nev. I had only worked a short time at Gold Hill when the fire in this mine put an end to my work.

Present position: Student.

Albert Buckingham, Univ. of Nev., Reno, Nev.

Proposed by J. E. Jones, Walter S. Palmer, Francis C. Lincoln.

Born 1893, Philadelphia, Pa. 1909-12, Central Manual Training School, Philadelphia, Pa. 1916, with J. H. Causten, Antimony Mines, near Lovelock, Nev.

Present position: Student, Mackay School of Mines.

James Lysle Chase, Leavenworth, Kan.

Proposed by Arthur C. Terrill, Erasmus Haworth, Richard L. Grider.

Born 1890, St. Paul, Minn. 1900-04, University Military School, Mobile, Ala. 1904-06, Barton Academy, Mobile, Ala. 1906-07, Leavenworth High School, Leavenworth, Kan. 1912-13, Spalding's Com'l. College, Kansas City, Mo. 1913-15, International Corres. Schools, Scranton, Pa. 1915-17, Kansas University, Lawrence, Kan. 1907-11, United States Navy. 1913, White & Davis, Pueblo, Colo. 1913, Colorado Fuel & Iron Co., Pueblo, Colo. 1913, Denver & Rio Grande R. R., Pueblo, Colo. and Thistle, Utah. 1913-14, Oregon Short Line R. R., Pocatello, Idaho. 1914-15, Kansas City, Mexico and Orient R. R., Wichita, Kans. 1915-17, Kansas University.

Present position: Student, Kansas Univ. Special—Geology & Mining.

Roscoe H. Clarke, Golden, Colo.

Proposed by H. C. Parmelee, J. C. Roberts, Harry Wolf.

Born 1892, Lawrence, Kans. Two years Univ. of Okla. Two years Colorado School of Mines.

Present position: Student, Colorado School of Mines.

Delos I. Dobson, Houghton, Mich.

Proposed by M. F. Houle, F. W. McNair, W. E. Hopper.

Born 1895, Detroit, Mich. 1908-12, Detroit Central High School. 1912-13, Highland Park H. S. (Mich.) 1913-15, M. C. M., Houghton, Mich. 1916-17, M. C. M., Houghton, Mich. 1915-16, Chemist, General Motors Co., Detroit, Mich.

Present position: Student, Michigan College of Mines.

Lewis Geo. Eisele, Golden, Colo.

Proposed by J. C. Roberts, H. C. Parmelee, Harry Wolf.

Born 1893, Iron Mt., Mich. 1907-11, Iron Mt. High School. 1911-12-13, Univ. of Wisconsin. 1914-15-16-17, Student at the Colo. School of Mines. Summer. 1912, Machinist at Iron Mt., Mich., Oliver Iron Mining Co. Summer, 1913, Rodman, Oliver Iron Mining Co. In charge of erection of an electrical transmission line for the Oliver Iron Mining Co. 1914, Rodman, Tenn. Coal, Iron and R. R. Co., Bessemer, Ala.; Asst. City Engineer, Iron Mt., Mich. Summer, 1915, Miner, Oliver Iron Mining Co., Iron Mt., Mich. Summer, 1916, City Engr., Iron Mt., Mich.

Present position: Student, Colo. School of Mines.

John M. Harrington, Winona Mine, Mich.

Proposed by M. F. Houle, W. E. Hopper, F. W. McNair.

Born 1892, Ironwood, Mich. 1909, Adams Township High School, Painesdale, Mich. 1914, Michigan College of Mines. 1909-11, Bell Ringer, Baltic Mining Co., Baltic, Mich. 1911-12, Clerk, So. Range Mine Co., Baltic, Mich. Clerk, H. W. Eave & Co., South Range, Mich. 1912-13, Engineer's Assistant, Baltic Mining Co. 1913-14, Trammer, miner, pumpman and mine guard, Baltic Mining Co. 1916-17, Two weeks, mucker, Winona Mining Co., Winona, Mich.

Present position: Student, Mich. College of Mines.

Robert Higgins, Golden, Colo.

Proposed by Harry Wolf, H. C. Parmelee, Geo. Young.

Born 1895, Pueblo, Colo.

Present position: Student, Colo. School of Mines.

Albert Bleile Hill, Clarksburg, W. Va.

Proposed by Fred Crabtree, Chas. R. Fettke, Joseph Jensen.

Born 1893, Shirley, W. Va. 1912, Graduate of Clarksburg High School. 1913-17, Attended Carnegie Tech. Summer, 1912, 1914 and 1916, Worked for oil well contractor, Samuel Bowman, and in drilling department of Hope Natural Gas Co., both of Clarksburg, W. Va. Summer, 1915, timberman Tenn. Copper Co., Ducktown, Tenn.

Present position: Senior student, Carnegie Institute of Technology. (Min. Eng.)

Lawrence A. Holmes, Menominee, Mich.

Proposed by M. F. Houle, W. E. Hopper, F. W. McNair.

Born 1896, Menominee, Mich. 1914, Menominee High School. 1914, Mich. College of Mines. 1911, Crawford Lumber Co., Piler, Prescott Foundry, asst. machinist. 1912, Frank Wecke, brick-layer. 1913, Mich. Refining & Preserving Co. 1914, Menominee Light and Traction Co., Laborer.

Present position: Student at the Mich. College of Mines.

Harry Hotchkin, Chicago, Ill.

Proposed by W. E. Hopper, M. F. Houle, F. W. McNair.

Born 1894, Chicago, Ill. 1912, Graduate Lane Technical High School, Chicago, Ill. 1912-15, attended Northwestern Univ. 1915-17, Mich. College of Mines. 1914, Summer, surveying, sampling and machine work at Tough Oakes Gold Mine, Ontario. 1915, Summer, work in the cyanide mill, Tough Oakes Mine, Kirkland Lake, Ontario.

Present position: Student, Michigan College of Mines.

Kwei Lun Hsueh, Golden, Colo.

Proposed by Harry Wolf, G. J. Young, H. C. Parmelee.

Born 1893, Wusih, Kiangsu, China.

Present position: Student, Colorado School of Mines.

Jerre Earl Huber, Mt. Carmel, Pa.

Proposed by Fred Crabtree, Chas. R. Fettke, Joseph Jensen.

Born 1893, Mt. Carmel, Pa. 1912, Graduated from Mt. Carmel High School. 1913-17, Attended Carnegie Institute of Technology, Mining Department. 1915, At Copper Cliff, Ontario, Canada, employed by Canadian Copper Co. 1916, At Alliance, Ohio, employed by Morgan Engineering Co.

Present position: Senior Student in Mining Engineering, Carnegie Institute of Technology.

Khoren L. Hussissian, Madison, Wis.

Proposed by Richard S. McCaffery, F. A. Kennedy, E. A. Barnard.

Born 1894, Armenia. Ripon College, two years, U. W. one year. 1916, Wisconsin Zinc Co., Winskell Mine.

Present position: Student.

Arthur Kendall, Hancock, Mich.

Proposed by M. F. Houle, W. E. Hopper, F. W. McNair.

Born 1893, Quincey, Mich. 1910, Hancock Central High School. 1913-14, Michigan College of Mines. 1915, Michigan College of Mines. 1910-12, Machinist, Quincey Mining Co. 1912-13, Miner, Quincey Mining Co. 1914-15, Miner, Quincey Mining Co. 1914-15, Surveyor, Quincey Mining Co.

Present position: Student, Michigan College of Mines.

Arthur Carruthers Kinsale, Golden, Colo.

Proposed by Harry Wolf, W. G. Haldane, J. C. Roberts.

Born 1890, Colorado Springs, Colo. 1911-13, New Mexico Military Institute, Roswell, N. M. 1914-17, Colo. School of Mines, Golden, Colo. Summers, 1913-14-15, Elkton Cons. Mining & Milling Co., Elkton, Colo., General Mining.

Present position: Student, Colorado School of Mines.

Stuart Condit Lawson, Madison, Wis.

Proposed by Richard S. McCaffery, F. A. Kennedy, E. A. Barnard.

Born 1894, East Orange, N. J. 1899-1911, E. Orange, N. J., Grammar and High School. 1913-17, Univ. of Wisconsin.

Present position: Senior in Mining Eng. Course, Univ. of Wis.

Marcus William Link, Madison, Wis.

Proposed by Richard S. McCaffery, F. A. Kennedy, E. A. Barnard.

Born 1896, Madison, Wis. 1916, Summer, employed by the Vinegar Hill Zinc Co.

Present position: Student, Univ. of Wis.

Sinclair H. Lorain, Houghton, Mich.

Proposed by M. F. Houle, F. W. McNair, W. E. Hopper.

Born 1895, Philipsburg, Pa. 1907-10, Bethel Military Academy. 1910-13, Philipsburg High School. 1913-14, Army and Navy Preparatory School. 1914, Michigan College of Mines. 1914, 4 mos., Mine surveying and mapping for Lorain Engineering Co.

Present position: Student.

Herbert Macmillan, New York, N. Y.

Proposed by Robert M. Raymond, Robert Peele, R. V. Norris.

Born 1892, Jersey City, N. J. 1911, New York University. 1913-17, Columbia University.

Present position: Student.

Richard Mead, Cambridge, Mass.

Proposed by Geo. S. Raymer, Albert Sauveur, H. L. Smyth.

Born 1893, Weston, Mass. 1915, Harvard, A. B. 1915, summer, at Mammoth Mine, Shasta County, Cal., general mining, mucking, surveying, etc.; U. S. Smelting, Mining & Refining Co. 1916-17, Summer and part of winter at Eustis Mill and Mine. Eustis, P. Q. milling, engineering, sampling and general work.

Present position: Student, Harvard Mining School.

Sydney A. Mewhirter, Golden, Colo.

Proposed by J. C. Roberts, H. C. Parmelee, Harry Wolf.

Born 1894, Thomas Co., Kansas. 1908-11-12-13, Manual Training High School, Denver, Colo. 1913, Colo. School of Mines. 1911-12, Donovan & Isaacs, Leasers, Cripple Creek. 1912, Summer, Strong Gold Mining Co., Victor, Colo. 1913, Summer, Portland Mining & Milling Co., Victor, Colo. 1914-15, Summer, Henri de Birse C. E., Paris, France.

Present position: Student, Colo. School of Mines.

Maynard Mizel, South Bethlehem, Pa.

Proposed by Howard Eckfeldt, Benj. L. Miller, Jos. W. Richards.

Born 1896, Sandusky, Ohio. 1910, Graduated Grammar School, New York City. 1914, Manual Training High School, Bklyn. N. Y. 1914, Entered Lehigh Univ. 1914, Summer, Acme Burlap Bag Co., New York, Salesman.

Present position: Student Lehigh University.

Harold Ellsworth Munn, Golden, Colo.

Proposed by Harry Wolf, J. C. Roberts, H. C. Parmelee.

Born 1888, Boston, Mass. 1902-06, Polytechnic High School, San Francisco, Cal. 1909-10, Associated with Chas. Renolds, M. E., in hydro-electric investigation for Candelaria Mines, San Dimas, Mex. 1910-11, Assayer, Surveyor, Mine Foreman, Mill Supt., Assistant Supt., LaPuerta Mining Co., Sinaloa, Mex. 1911-13, Assistant Supt. for Estaca Mining Co., Sinaloa, Mex. 1913-17, Student, Colo. School of Mines.

Present position: Student, Colo. School of Mines.

Paul T. Norton, Jr., Madison, Wis.

Proposed by Richard S. McCaffery, F. A. Kennedy, E. A. Barnard.

Born 1889, Elizabeth, N. J.

Present position: Senior in College of Engineering, Univ. of Wis.

Arthur Frederick Peterson, Madison, Wis.

Proposed by Richard S. McCaffery, F. A. Kennedy, E. A. Barnard.

Born, 1893, Ironwood, Mich. 1914-15-16, Univ. of Wis. 1908-09, Newport Mining Co. 1913, Newport Mining Co. 1914, Newport Mining Co. 1915, Newport Mining Co.; about 4½ years total; various underground jobs.

Present position: Student.

Virgil T. Price, Rapid City, South Dakota.

Proposed by Welton J. Crook, Albert S. Halley, Chas. Bentley.

Born 1894, Rapid City, South Dakota. 1909-11, Rapid City High School. 1911-12, Omaha High School. 1912-17, South Dakota State School of Mines.

Present position: Student, So. Dak. School of Mines.

Edward W. Robinson, Golden, Colo.

Proposed by J. C. Roberts, H. C. Parmelee, Geo. J. Young.

Born 1895, Denver, Colo. 1909-13, East Denver High School, Denver, Colo. 1913 to date, Colo. School of Mines, Golden, Colo. 1915-16, Wyoming Mines & Development Co., Rawlins, Wyo. 1917, Paramount Reduction Co., St. Elmo, Colo.

Present position: Student at Colo. School of Mines.

Louis Sandler, Pittsburgh, Pa.

Proposed by Fred Crabtree, Chas. R. Fettke, Joseph Jensen.

Born 1893, Pittsburgh, Pa. 1913, Pittsburgh Central High School. 1913-17, Carnegie Institute of Technology, Metallurgical Engineering. 1913-14, Clerk, Strassburger & Joseph, Pittsburgh. 1915, Clerk, Penn. Railroad. 1916, Chief of Party, Valuation of West Penn. Traction Co., Pittsburgh, Pa.

Present position: Senior student, Metal. Eng., Carnegie Institute of Technology.

Fred C. Sealey, Colo. School of Mines.

Proposed by J. C. Roberts, H. C. Parmelee, Harry Wolf.

Born 1895, Denver, Colo.

Present position: Student, Colorado School of Mines.

C. Kenneth Smullen, Baltimore, Md.

Proposed by Fred Crabtree, Chas. R. Fettke, Joseph Jensen.

Born 1895, Baltimore, Md. 1909-13, Baltimore City College. 1913-17, Carnegie Institute of Technology, Mining Engineering. On completion of course will receive a degree of B. S. in Mining Engineering. 1914, Pittsburgh, Silver Peak Gold Mining Co., Blair, Nev., Mucker and machine helper. 1915, Maryland Steel Co., Sparrow's Point, Md., Fitter on steel construction. 1916, Baltimore Copper Rolling & Smelting Co., Canton, Md., assistant to chief of construction on building of tank house for electrolytic copper precipitation.

Present position: Senior Student (Mining Engineering), Carnegie Institute of Technology.

Robert A. Thurstin, Golden, Colo.

Proposed by J. C. Roberts, Harry Wolf, H. C. Parmelee.

Born 1888, Bowling Green, Ohio. 1910, Kenyon College, Gambier, Ohio, Degree of B. S. 1906, Graduate of High School, Bowling Green, Ohio. 1910-11, 1913-14, 1915-16, Colorado School of Mines, Golden, Colo. 1906-07-08, The Ohio & Western Lime Co., Huntington, Ind. 1911, Pride of the West Leasing Co., Silverton, Colo.

1912, American Rare Metals Co., Dolores, Colo. 1913, Primos Chemical Co., Newmire, Colo. 1915-16, Elkton Cons. Mining and Milling Co., Elkton, Colo; Golden Cycle Mining Co., Victor, Colo.; Cresson Consolidated Mining Co., Elkton, Colo.
Present position: Student, Colorado School of Mines.

Emory M. Tiffany, Golden, Colo.

Proposed by J. C. Roberts, David Carroll, Geo. J. Young.

Born 1895, Durango, Colo. 1910-14, Durango High School. 1914, Colorado School of Mines. 1915, Summer, American Flotation Co. 1916, American Smelting & Refining Co., Silverton, Colo.

Present position: Student at Colorado School of Mines.

Ralph Oliver Williams, Uniontown, Pa.

Proposed by Fred Crabtree, Chas R. Fettke, Joseph Jensen.

Born 1892, Uniontown, Pa. 1911, Uniontown High School. 1913-17, Carnegie Institute of Technology, Metallurgical Engineering. 1912-13, Baker Engineering Co., Uniontown, Pa. 1915, Summer, E. E. Porter, Boro. Eng., Uniontown, Pa. 1916, Summer, The Barrett Co., Pittsburgh, Pa.

Present position: Senior Student, Metallurgical Eng., Carnegie Institute of Technology.

Frank K. Ziegler, Bee Ridge, Fla.

Proposed by Fred Crabtree, Chas. R. Fettke, Joseph Jensen.

Born 1893, Georgetown, Ohio. 1907-11, Parkersburg, W. Va., High School. 1911-12, Marietta, Ohio, College. 1913-17, Carnegie Institute of Technology (Metallurgical Engineering). 1912-13, Parkersburg Iron & Steel Co. (Furnace Helper), Parkersburg, W. Va. 1915 (Summer), Canadian Copper Co., Copper Cliff, Ontario, Canada, Converter and Blast Furnace work. 1916 (Summer), Pennsylvania Smelting Co., Carnegie, Pa., Helper on Blast Furnace (Lead), Helper on Parke's Process (Kettles) for refining base bullion.

Present position: Student in Carnegie Institute of Technology (Senior Year).

CHANGE OF STATUS

Associate to Member

John Boyle, Los Angeles, Cal.

Proposed by A. B. W. Hodges, Seeley W. Mudd, Robert E. McConnell.

Born 1866, Illinois. 1880-83, Manual Training School, St. Louis, Mo. 1883-87, Washington University, St. Louis, Course in Mining Engr. 1886, Degree of Bachelor of Engineering, Washington Univ. 1887, Degree of Engineer of Mines, Washington University. 1887-92, Manager, Mountain Key Mining Co., Pinos Altos, N. M. 1896-97, Manager, Trust Ruby Mining Co., Ouray, Colo. 1898-1901, Assayer-in-charge, U. S. Assay office, St. Louis, Mo. 1902-07, Consulting Mining Engineer, St. Louis, Mo. 1908-15, President and Manager, Union Basin Mining Co., owning Golconda Mine, Kingman, Ariz. While student at Washington Univ. was put in as associate member.

Present position: Mining Engineer.

CHANGES OF ADDRESS

The following corrections for the Year Book have been received at the Secretary's office during the period Feb. 10 to Mar. 10, which taken together with the lists of new members and changes of address published in the Bulletins Nos. 121 to 123 therefore supplement the annual Year Book and bring it up to date of Mar. 10, 1917.

ADAMS, HENRY, Cons. Min. Engr. 25 Fremont St., New London, Conn.

AERTSEN, GUILLIAEM. Midvale Steel Co., Box 1323, Philadelphia, Pa.

ARMSTRONG, J. A. B., Central Chili Copper Co., Ltd., Panulcillo Alto,
Coquimbo, Chile, S. Amer.

BACHE, FRANKLIN. . . . Min. Engr., Pres., Bache-Denman Coal Co., Fort Smith, Ark.

- BALLEMBERG, ADOLF G. 1517 Conway Bldg., Chicago, Ill.
 BANKS, HAROLD P., Min. Engr. and Met. 61 Broadway, New York, N. Y.
 BARNARD, ENOCH A. 911 University Ave., Madison, Wis.
 BARNES, WALTER A. Clift Hotel, San Francisco, Cal.
 BARRINGER, DANIEL M. 1242 Real Estate Trust Bldg., Philadelphia, Pa.
 BATCHELLER, JAMES H., Mgr., Virginia Lead & Zinc Corporation, Mineral P. O., Va.
 BEESON, JOSEPH JOSIAH, Geol. and Mine Mgr., Emma Cons. Mines Co.,
 Alta, Utah.
 BEDDALL, THOMAS H. P. O. Box 4, Pottsville, Pa.
 BENT, QUINCY. Bethlehem Steel Co., Steelton, Pa.
 BRANDES, JUAN FELIX. Cons. Min. Engr., Pres., Advance Mfg. Co.,
 P. O. Box 242, Santa Barbara, Cal.
 BROWNE, KENNETH C. Creighton Mine, Ont., Canada.
 BRYANT, ALBERT D. P. O. Box 145, Cerrillos, New Mex.
 BUGBEE, EDWARD E., Asst. Prof. of Min. Engrg. and Met.,
 Mass. Inst. of Technology, Cambridge, Mass.
 BURNS, WILLIAM. 827 McIntyre Bldg., Salt Lake City, Utah.
 BUTLER, MEL C. Waldorf Hotel, Seattle, Wash.
 BUTTERFIELD, GEORGE B. 524 Federal St. N. S., Pittsburgh, Pa.
 CAIRNS, JOHN M., Care Messrs. Henry S. King & Co.,
 65 Cornhill, London, E. C., England.
 CALLEN, ARTHUR S. 3510 N. 16th St., Philadelphia, Pa.
 CALVERT, W. R. 1209 Grand Ave. Temple, Kansas City, Mo.
 CAMPBELL, ARTHUR R. 413 Republic Bldg., Kansas City, Mo.
 CARROLL, FRANK. Ingersoll-Rand Co., 11 Broadway, New York, N. Y.
 CARROLL, GEORGE A. C. P. Broad Co., Ryan, Cal.
 CARUTHERS, ANTHONY W., Coal Dept., Delaware, Lackawanna & Western R. R. Co.,
 Scranton, Pa.
 CLEMENT, HARRISON E., Mine Manager, Delta Copper Co., Ltd.,
 Tramville, B. C., Canada.
 COCHRAN, RALF S. Struthers, Ohio.
 CONOVER, M. J. Hotel Bellevue, San Francisco, Cal.
 CREMER, FELIX, Andes Copper Min. Co., Care Jorge Larrien, Casilla 230,
 Antofagasta, Chile, S. Amer.
 CROOK, WELTON J. Box 985, Stanford University, Cal.
 CURRIE, DAVID. Norfolk House, 7 Lawrence Hill, London, E. C., England.
 DAGGETT, ELLSWORTH. Hotel Biltmore, San Francisco, Cal.
 DEANE, WILLARD A. Waxhaw, N. C.
 DELANE, LEWIS A. St. Joseph Lead Co., Bonne Terre, Mo.
 DICKINSON, ALBERT W. Wyoming, Iowa.
 DICKINSON, EDMUND S. Granby, Mo.
 DOWS, DAVID, Pres. and Treas., American Grondal Co.,
 120 Broadway, New York, N. Y.
 DUNSFORD, ENSOR R. Quarry St. Anns, C. B., Nova Scotia.
 EDELEN, ALEXANDER WALTER, Supt. Anganguero Unit, American Smelt. & Ref. Co.,
 Socorro, New Mexico.
 FARGO, LIVINGSTON WELLS. Box 596, Chicago, Ill.
 FELLENCER, CHARLES A. Eden Min. Co., Bluefields, Nicaragua, C. A.
 FERRAZ, JORGE B. DE ARAUJO, Asst. of Geology & Petrography, Servicio Geologico e
 Mineralogico, Ministerio de Agricultura, Rio de Janeiro, Brazil, So. America.
 FERRIS, ALBERT L. Yaeger Canyon Mine, Dewey, Ariz.
 FRASER, WILLIAM L. 1070 E. Ocean Ave., Long Beach, Cal.
 FRENCH, C. L. Y. M. C. A., Portsmouth, Va.
 FRENCH, R. W. Care Chas. Butters & Co., Oakland, Cal.
 FRY, LOUIS D., Smelter Supt., The Mazapil Copper Co., Ltd.,
 Saltillo, Coahuila, Mexico.
 FUKETA, FUSAJIRO, Yoshioka Mine, Fukiya, Kawakamigun, Okayamaken, Japan.
 FULLAWAY, RICHARD M. 102 So. Central Ave., Eagle Rock City, Cal.
 GALLAGHER, FRANCIS JAMES. Box 97, Bisbee, Ariz.
 GARNER, CARROLL A. Division Engr., Lehigh Valley Coal Co., Hazelton, Pa.
 GLOBE, ALEXANDER R., Hollinger Con. Gold Mines, Ltd., Timmins, Ont., Canada.
 GORDY, SHEPPARD B. Casilla 49 D, Santiago, Chile, So. America.
 HALL, MORTIMER L. Kennet, Cal.
 HART, VERNE A. Walker Mine, Portola, Cal.
 HAYDEN, WALLACE H. 520 Jeannette Street, Wilkinsburg, Pa.

- HEISE, HENRY C. Zinc Co., Ltd., Montaubon, P. Q., Canada.
HILBY, GEORGE R. Salinas, Cal.
HILL, JOSEPH H. General Delivery, Ray, Ariz.
HINCKLEY, ELMER R. Swansea, Ariz.
HINMAN, B. C. Mannsville, Jefferson Co., N. Y.
HOBART, EDMUND NORRIS. Linden Hotel, El Paso, Tex.
HODGKINSON, HAROLD H. Box 1536, Jerome, Ariz.
HOFMAN, H. O., Prof. of Met., Mass. Inst. of Technology, Cambridge, Mass.
HOFFMANN, JOHN STONE, Rodman, Burro Mountain Copper Co., Tyrone, New Mexico.
HOLDEN, EDWIN C. Davison Sulphur & Phosphate Co., Cienfuegos, Cuba.
HOOK, JOSEPH S. Marland Oil Co., Ponca City, Okla.
HUMPHREY, GEORGE S. Clayville, Oneida County, N. Y.
HUTCHINS, JOHN P. American International Corp., Dvortsovaya Nabereshnaya,
No. 8, Petrograd, Russia.
ISHIKAWA, ICHIRO. 375 Nishigahara Takinogawa, Tokyo, Japan.
JACKSON, G. T. Asst. Mgr., Alaska Gastineau Min. Co., Juneau, Alaska.
JOHNSON, AMOS D., JR., Min. & Met. Engr.,
Rm. 1409, 71 Broadway, New York, N. Y.
JOHNSTON, ALBERT WHEELER. Rm. 2011, 111 Broadway, New York, N. Y.
JOHNSTON, IRVING H., Pittsburgh Testing Laboratory,
304 New Telegraph Bldg., Detroit, Mich.
KAEDING, C. D. The Dome Mines Co., Ltd., 43 Exchange Place, New York, N. Y.
KAMP, WILLIAM HENRY. Tooele, Utah.
KAWAMURA, TAKESHI, Kenjiho Iron Works, Mitsubishi Goshi Kaisha,
Kenjiho, Koshugun, Kokai-do, Korea.
KELLEY, ARTHUR L. Tombstone, Ariz.
KELLY, FRANK J. Care Y. M. C. A., Sacramento, Cal.
KELSEY, W. M., Supt., Palmerton Plants, New Jersey Zinc Co. (of Pa.),
Palmerton, Pa.
KEMP, JAMES TAYLOR. 51 Hyatt St., Staten Island, N. Y.
KOBBE, WILLIAM H. Pierce Fordyce Oil Assn., Dallas, Tex.
KRAUSE, WALTER B. American Steel Foundries, Chester, Pa.
KWARI, ARTHUR M. Hotel West Court, Denver, Colo.
LEE, HUGH B. Engr., Porcupine Crown Mines, Ltd., Timmins, Ont., Canada.
LEE, HUNYET. Care Toko Lay Lau Sim, Passar Barol, Batavia, Java, D. E. I.
LEOPOLD, FOREMAN N. 1517 Conway Bldg., Chicago, Ill.
LINDBERG, CARL O., Min. Engr. 1216 Hollingsworth Bldg., Los Angeles, Cal.
LINDSAY, WILLIAM R. 902 Hobart Bldg., San Francisco, Cal.
LIVINGSTON, ADDISON S. The Detroit Copper Mining Co., Morenci, Ariz.
LOCKE, CHARLES E., Asst. Prof. of Min. Engrg. and Met., Mass. Inst. of Technology,
Cambridge, Mass.
LOVEJOY, JOHN M. P. O. Box 748, Electra, Tex.
LYON, THOMAS T. JR. 526 Hennessey Bldg., Butte, Mont.
LYON, DORSEY A., Care U. S. Bureau of Mines, Univ. of Washington, Seattle, Wash.
MCALLISTER, J. E. 804 Traders Bank Bldg., Toronto.
MC CREERY, JAMES H., Sec. Tres., Ophuls, Hill & McCreery, Inc.,
112-114 W. 42d St., New York, N. Y.
MCKEE, WALTER S., Vice-Pres., American Manganese Steel Co.,
1850 McCormick Bldg., Chicago, Ill.
MCKINLAY, JAMES. Avenida Uruguay 78, Mexico City, Mex.
MCNUTT, VACHEL H. 328 Mayo Bldg., Tulsa, Okla.
MACKALLOR, D. C. Care A. E. Bendelari, Picher, Okla.
MACKENZIE, K. G. The Texas Co., 17 Battery Place, New York, N. Y.
MARSH, HARRY W. Tamarack & Custer Cons. Min. Co., Wallace, Idaho.
MEANS, ALAN H. Eden Mining Co., Bluefields, Nicaragua, C. A.
MENTZEL, CHARLES, Min. Engr., Lewisohn Bros., 11 Broadway, New York, N. Y.
MILLER, D. IRVING. 224 Brown-Marx Bldg., Birmingham, Ala.
MILLER, HARRY H. 226 N. Vendome St., Los Angeles, Cal.
MOORE, EDWARD W. El Cajon, Cal.
MOORE, STANLEY R., Min. Engr. Clayton, Idaho.
MOTHERWELL, HARRY A. B. General Delivery, Trail, B. C., Canada.
MORROW, BAYARD S., Supt., Grinding and Flotation, Copper Concentrator,
Washoe Reduction Wks., Anaconda, Mont.
MOULE, JOHN W., Genl. Manager, Corella Copper Co., Rosebud Mine,
via Ballara, Cloncurry, North Queensland, Australia.

- MOULTON, JOHN CARROLL, Pittsburg Silver Peak Gold Min. Co., Sonora, Cal.
 NAGEL, FRANK J. Box 777, El Paso, Tex.
 NELSON, C. W. Strand Hotel, San Francisco, Cal.
 NEUSTADTER, HAROLD ARTHUR. Desloge, Mo.
 OF, CHARLES. Palisade, N. J.
 OHNSORG, N. L. The Phosphate Mining Co., Nichols, Polk Co., Fla.
 O'NEIL, FREDERICK W., Asst. Genl. Sales Mgr., Ingersoll Rand Co.,
 11 Broadway, New York, N. Y.
 OTAGAWA, TATSURO, Care Messrs. Z. Horikoshi & Co.,
 115 E. 23d St., New York, N. Y.
 OTTO, HENRY H. Lehigh Valley Coal Co., Wilkes-Barre, Pa.
 OWEN, TOM M., Genl. Supt. of Mills, Federal Mining & Smelting Co., Wallace, Idaho.
 PARKER, MORRIS B. 1540 Curson Ave., Hollywood, Cal.
 PARSONS, ARTHUR B. 402 1st. Ave., Salt Lake City, Utah.
 PATCHELL, FREDERICK J. Instructed to hold all mail.
 PHELPS, WILLIAM B. Tom Reed Gold Min. Co., Oatman, Ariz.
 PHILLIPS, WILLIAM B. 507 West 33d St., Austin, Tex.
 POAGE, J. G. Central Hotel, Port Pirie, So. Aust.
 PORTER, FRED S. Engr., Yukon Copper Co., Ltd., White Horse, Y. T., Canada.
 PORTER, JESSE C. C. L. Constant Co., 43 New Street, New York, N. Y.
 PROMMEL, HAROLD W. C. 1503 Ford St., Golden, Colo.
 RANDALL, JOHN, Met. Engr. 1705 Arapahoe Ave., Boulder, Colo.
 RAU, H. L. Bisbee, Ariz.
 RICHARDS, R. H., Prof. of Min. Engr. and Met., Mass. Inst. of Technology,
 Cambridge, Mass.
 RISQUE, JAMES B. Care Edward H. Clark, 15 Broad St., New York, N. Y.
 ROBERTS, EDWARD J., Gen. Mgr., Nevada Pyramid Mining Co., Riverside Hotel,
 Reno, Nevada.
 ROBERTS, DUDLEY E. El Paso Smelter, El Paso, Tex.
 RODEGERDTS, C. A. Atolia Min. Co., Atolia, San Bernardino Co., Cal.
 ROGERS, OSBORN PEABODY. Robt. Munro, Alaska Bldg., Seattle, Wash.
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 SANDOVAL, P. P. O. Box 535, Calexico, Cal.
 SCHUETTENHELM, JOHN B. American Smelt. & Ref. Co., Murray, Utah.
 SELBIE, CHARLES C., Min. Engr., 62 N. Meredith Ave., Pasadena, Cal.
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 Hawaii), 1106 Merchants National Bank Bldg., 625 Market St., San Francisco, Cal.
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 55 Wall St., New York, N. Y.
 SINN, FRANCIS P. Gen. Supt., New Jersey Zinc Co. (of Pa.) Palmerton, Pa.
 SMITH, HENRY B. St. Joseph Lead Co., Rivermines, Mo.
 SMITH, LYON. Asst. Supt., River Smelt. & Ref. Co., Florence, Colo.
 SNOW, FREDERICK W. Room 1227, 42 Broadway, New York, N. Y.
 SOPER, RALPH H. Apartado No. 106, Tampico, Tamps., Mexico.
 SPERR, J. DANA, Min. Engr. Houghton, Mich.
 STAY, THERON D. Aluminum Castings Co., Cleveland, Ohio.
 STEELE, CHESTER H. 644 W. Quartz St., Butte, Mont.
 STEWART, HOWARD R. Mine La Motte, Mo.
 STEWART, JOHN B. Instructed to hold all mail.
 STILLMAN, CHARLES A. Room 1361-332 So. Michigan Ave., Chicago, Ill.
 STOCKWELL, RUPERT K., Mgr., Robins Conveying Bolt Co.,
 Newhouse Bldg., Salt Lake City, Utah.
 SUVERKROP, EDWARD A. Chile Exploration Co., Chuquicamata, Chile, S. Amer.
 SWEETSER, RALPH H., Works Mgr., The Columbus Iron & Steel Co., Columbus, O.
 TARR, RUSSELL S. 429 W. 117th St., New York, N. Y.
 TOWNSEND, ROBERT H. Care Melchers Sucs., Mazatlan, Sinaloa, Mexico.
 VERNON, JOSEPH A. 7 Cottage St., Newport, R. I.
 WEBSTER, ERNEST B. 2422 Montague St., Regina, Sask., Canada.
 WELLS, ARTHUR EDWARD. U. S. Bureau of Mines, Salt Lake City, Utah.
 WENDEL, EDMUND. 6013 Franklin Ave., Cleveland, O.
 WESTON, CHARLES V. 934 Lincoln Pkway, Chicago, Ill.
 WHEELER, ARCHER E. Room 1227, 42 Broadway, New York, N. Y.
 WHERRY, HENRY P., Rosendale Reddaway Belting & Hose Co.,
 Euclid Ave., Newark, N. J.

WIEGAND, AUGUST JOHN, Hudson Coal Co., Union National Bank Bldg., Scranton, Pa.
 WIEBELT, FRANK J. Dorothy Block, Butte, Mont.
 WILLIAMS, W. A., Asst. to General Manager, Empire Gas & Fuel Co.,
 Drawer F, Bartlesville, Oklahoma.

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Name	Last address of record from which mail has been returned.
AHIER, PHILIPPE D.,	Anglo-Greek Magnesite Co., Ltd., Afrati, Chalois, Euboea, Greece.
BYERS, WHEATON B.	50 Hampden Hall, Cambridge, Mass.
CHURCH, JOHN L.	P. O. Box 1123, Los Angeles, Cal.
DALEY, STEPHEN HOWARD JR.,	Cananea Cons. Copper Co., Cananea, Sonora, Mexico.
DOUGHETT, GEORGE M.	Crescent Metals Co., Lock Box 5, Glenwillard, Pa.
HOHAGEN, GEORGE.	Potosi, Bolivia, So. America.
JAMES, CYRIL H.	Kingman, Ariz.
JEFFREY, ROBERT H.	Haileybury, Ont., Canada.
NANCE, W. FRED.	Belnap Coal Co., Newcastle, Pa.
PARKER, JOSEPH EDMUNDSON,	Rhodesia Exchange & Development Co., Box 213, Bulawayo, Rhodesia, So. Africa.
SIBBALD, ALEXANDER.	P. O. Box 1091, Globe, Ariz.
STEVENS, HUGH M.	Canadian Mining & Finance Co., Timmins, Ont., Canada.
WINNE, ROLLA D.	17 College Ave., Adrian, Mich.
ZACH, LOUIS M.	U. S. Smelt., Ref. & Min. Co., Salt Lake City, Utah.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Feb. 10 to Mar. 10, 1917.

Date of Election.	Name.	Date of Death.
1899	Hazard, Frederick R.	Feb. 27, 1917.
1892	Burrell, Alexander.	Feb. 13, 1917.
1872	Church, John A.	Feb. 12, 1917.
1881	Keller, Hermann A.	Feb. 16, 1917.
1895	Hiller, Edward.	Feb. 3, 1917.
1916	Saladin, Edouard E.	Feb. 12, 1917.

BIOGRAPHICAL NOTICES

ALEXANDER BURRELL

Alexander Burrell, general manager of the Furnace Creek Oxide Copper Mining Co., and a member of the Institute for 20 years, died suddenly at the Argo mine on Feb. 13, 1917. Mr. Burrell was one of the best known mining men in Montana. He was born at Edinburgh, Scotland, Jan. 14, 1851, and came to Chicago with his parents in 1856. His father, who was a coppersmith, died soon after their arrival, and the son had few opportunities to gain an education in school, but was a natural student and was possessed of a liberal education acquired through self-teaching and through his constant reading and studying, particularly along scientific lines. It was this technical knowledge both of mining and business, as well as his experience in the practical side of mining and his ability as an executive, that won for him a high place in mining circles in Montana. As a very young man he worked as a miner in Illinois and later became an operator of a coal property. Subsequently he engaged

with his brother in a manufacturing business in Chicago, but, disposing of his interest in 1888, he moved to Marysville, Mont., where he entered the employ of the Montana Co., Ltd., an English concern which owned and developed the Drumlummon Mine. Through steady promotion he finally became general manager of this company and operated it for many years, living during this time at Helena, Mont., where he possessed an attractive home and delighted in entertaining his friends. He was a Republican and was always interested in politics. In 1892 he was elected to the legislature from Lewis and Clark county and was the author of some very important mining legislation through his membership on the Committee on Mines and Mining. He had not been engaged in active business for several years, and had moved from Helena to Choteau, Teton county, when he was recently made general manager of the company in whose service he was when he died.

On Apr. 8, 1879, Mr. Burrell married Miss Abbey Kiersted of Morris, Ill., and is survived by the following children: George L., Alexander A., Grace, Sidney H., John A., Roger A., Howard and Nan K. Another son, William D., died in California in 1906.

AUGUST RAHT*

August Raht, whose death occurred at San Francisco, Dec. 25, 1916, was born Feb. 25, 1843, at Dillenburg, Duchy of Nassau, where his father was president of the Court of Appeals. After passing through the gymnasium, he studied mining and metallurgy, graduating from the Polytechnic Institute at Hesse-Cassel in 1863, and continued his studies at Freiberg.

He came to the United States about 1867 and engaged in professional work with the Ducktown Smelting and Reduction Co., at Ducktown, Tenn. He was connected with metallurgical work in the United States more or less continuously except for some professional trips to Italy and to Africa, being absent about two years. On his return to the United States, he went West and became connected with various metallurgical operations at Pueblo, Colo., New Mexico and Utah, at first with the Mingo plant and later, in 1883, building the Horn Silver plant. About 1887 he went to Wickes, Mont., and later to Helena, Mont., building the plant of the Helena & Livingston Reduction Co., at East Helena, which afterward was consolidated with the Great Falls plant, built by Anton Eilers, forming the United Smelting and Refining Co., of which Mr. Raht was manager.

About 1889 Meyer Guggenheim & Sons built and put into operation the Philadelphia Smelting Works at Pueblo, Colo., their chief purpose being to smelt more advantageously than the terms offered by other smelters, the zinky lead sulphide ore of the A. Y. & Minnie mine at Leadville, in which the Guggenheims were interested. The Philadelphia plant was put in operation with metallurgical and commercial results that looked disastrous and would indeed probably have wrecked a concern less strong financially. After a series of vicissitudes the Guggenheims made up their minds that what they needed was the best metallurgical advice and direction that possibly could be obtained, and seeking

* From the *Engineering and Mining Journal* of March 10, 1917.

such assistance, they engaged Mr. Raht, who took up the work at Pueblo, about the latter part of 1891 or early in 1892. He soon succeeded in putting the enterprise upon its feet and thereby convinced the Guggenheims of the correctness of the principle that the best technical knowledge—no matter what salary or fee may be required to get it—is the cheapest thing in the operation of such a business.

Mr. Raht continued with the Guggenheims in an operating and consulting capacity for a number of years while they were enlarging and developing their metallurgical plants, acting for them in Colorado, Mexico, Perth Amboy, N. J., and other places. He visited Australia for the London Exploration Co. about 1899 and, on returning to the United States, was again connected with the Guggenheim interests, especially the American Smelting and Refining Co., until his retirement from active work about 1910.

August Raht was actively and prominently connected with the development of the smelting industry in the United States during its most important period, both in copper and lead, although more prominently in the latter. He originated many improvements in details of operation and was one of the first of the Western metallurgists to recognize the importance of being able to treat complex sulphide ores, his work of this kind at the Philadelphia smelter at Pueblo, Colo., being very largely the cause of its success. This plant was one of the pioneers in providing itself with equipment for treating lead sulphide ores on a large scale. Mr. Raht was one of the first to recognize the possibility and benefit of low fuel in the blast-furnace operations connected with lead smelting. In the early stage of lead smelting, and before the present practice of converting mattes became common, he worked up such copper byproducts at the Philadelphia plant into blister copper. During the greater part of his later technical career he successfully devoted himself to solving the new problems which were arising with the evolution of lead smelting.

In addition to being a man of unusual technical ability, he had a fund of general knowledge, particularly with regard to all branches of natural history, which was a constant source of wonderment and admiration to those who had the privilege of knowing him intimately. This knowledge seems to have come from his early education under the thorough German system, coupled with a deep personal love for everything in nature. He was a sportsman of the highest type, both as regards hunting and fishing, indulging in this purely for the sport rather than for the sake of getting the game; and in the latter part of his life, especially after his retirement from active work, his principal relaxation was, while living on the California coast, to indulge in fishing of the various kinds which the locality afforded. With all his varied knowledge, he was so exceedingly modest in using it that it took a rather intimate acquaintance to find it out.

To the younger metallurgists whom he met during the development of this science in Colorado and elsewhere, Mr. Raht was of the greatest help. They all soon learned that they could freely ask his advice and assistance with the assurance of receiving it in the most unreserved and helpful way.

Mr. Raht leaves two daughters, Mrs. L. G. Eakins, wife of general manager L. G. Eakins, of the American Smelting and Refining Co. in Denver, and Mrs. Stewart A. Marsh, of Grand Rapids, Mich.

JOHN A. CHURCH

By the death of Prof. Church, the Institute loses one of its earliest active members. The following account of his life is taken in part from the *Engineering and Mining Journal*.

John Adams Church was born April 5, 1843, at Rochester, N. Y. He graduated from the School of Mines, Columbia University, in 1867, as a member of the first class to receive the degree of the school. Shortly afterward he went to Europe and spent several years in travel, the professional side of which was revealed in his "Notes on a Metallurgical Journey," published in 1873. During this period he took a practical course at Clausthal.

On his return to this country, he took over Prof. Egleston's courses in mineralogy at the School of Mines, during the latter's absence on leave. From 1872 to 1874, he was assistant editor of the *Engineering and Mining Journal*. In 1877, as a member of the U. S. Geological Survey west of the 100th Meridian ("Wheeler's Survey"), he made the studies which resulted in a monograph on the Comstock Lode, published in 1880. In this treatise he advanced the novel and ingenious theory that the heat of the rocks and waters encountered in depth on the Comstock lode is due to the kaolinization of the rocks—a theory which elicited much discussion, and was not generally accepted by chemical geologists as a complete and satisfactory explanation. In the light of subsequent publications in the science of economic geology and the formation of ore-deposits, this discussion has become an academic one—the heat of the earth's interior being recognized as both the product and the cause of various thermic processes, among which those of metamorphism have their secondary place. Prof. Church's paper on the subject in the *Transactions* of the American Institute of Mining Engineers (Vol. vii, pp. 45-76) is very interesting and able. It was controverted with equal ability by George F. Becker, of King's survey, in Vol. viii of the same *Transactions*.

From 1878 to 1881, Mr. Church was professor of mining and metallurgy in the State University of Ohio. Besides the paper just mentioned, he made at this period many other contributions to professional literature.

In 1880, he became superintendent of the Tombstone Milling and Mining Co., in Arizona. One of his problems here was to find a basic flux for the lead-silver concentrates from his mill; and this he solved by replacing iron with manganese. For two years he ran his blast-furnaces on slags containing about 43 per cent. MnO , characterized by great fluidity and low metal losses. These slags have become the classic examples of their types.

In 1886, the Chinese Viceroy, Li Hung Chang, engaged him to introduce American mining in China. Some old silver mines in Mongolia were selected for modern development, and Mr. Church took his family 150 miles north of the Great Chinese Wall, among a people who had never before seen a European. Here he spent nearly three years, reopening the mines, modernizing the practice and erecting a smelter. Again the problem of basic flux came up, and this time it was solved with crushed magnetite, the only available material. A Chinese belief in dragons forced him to send an extra day's journey for coal; the mines were threatened by a band of robbers, which was finally broken up by cavalry; the labor was utterly ignorant and at times difficult to handle. Never-

theless, at the expiration of his contract, Mr. Church left a producing property, although he foresaw the continued difficulties of modern industry in a land without railroads.

Upon his return to America in 1890, he established a consulting practice, which he maintained until his death.

Prof. Church became a member of the Institute in 1872, a few months after its formation. He was a Manager from 1879 to 1891, and from 1894 to 1896, and a Vice-President in 1907 and 1908. His contributions to the *Transactions* were as follows:

Title	Vol.	Page	Year
Economical Results in the Treatment of Gold- and Silver-Ores by Fusion.....	I	242	1872
Coking Under Pressure.....	I	322	1872
Velocity of Blast-Furnace Gases.....	IV	119	1875
Blast-Furnace Statistics.....	IV	221	1876
The Mode of Combustion in the Blast-Furnace Hearth....	VII	33	1878
Heat of the Comstock Mines.....	VII	45	1878
Accidents in the Comstock Mines, and Their Relation to Deep Mining.....	VIII	84	1879
Heat of the Comstock Lode.....	VIII	324	1880
Concentration and Smelting at Tombstone, Ariz.....	XV	601	1887
The Cause of Faulting.....	XXI	782	1893
Remarks on Prof. Posepny's Paper on the Genesis of Ore-Deposits.....	XXIII	593	1893
The Manganese Slags of Tombstone, Ariz.....	XXIV	559	1894
Discussion of the Chemistry of Ore-Deposition.....	XXXIII	1,065	1902

Prof. Church was also a member of the Century Club in New York; and a frequent contributor to technical literature. In 1879, he received the degree of Ph. D. and very recently that of M. Sc., both from his Alma Mater. In 1884, he married Miss Jessie A. Peel, who, with a son, still survive him.

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¹ Until Feb., 1918.² Until Feb., 1919.³ Until Feb., 1920.⁴ Until Feb., 1921.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Lead Mining and Smelting at Galetta, Ont.

BY WILLIAM E. NEWNAM, * B. S., COLLINSVILLE, ILL.

(St. Louis Meeting, October, 1917)

LEAD mining has been carried on in several localities of the Province of Ontario in a desultory fashion for the past 60 years, but up to 1916 the results have not been of much commercial importance. The most ambitious attempt was made by an English company which built a lead smelter at Kingston, Ont., in 1880 to treat the concentrates from the Frontenac mines. After 2 years' operation, the mines and smelter were abandoned. Up to the present there has not been a sufficient tonnage of concentrates to support an efficient blast-furnace plant and as most of the small properties could not stand the heavy freight and duty charges into the United States, development work has not been carried on in a systematic manner. As a result the industry has languished.

At Galetta, in southeastern Ontario, considerable prospecting and development work has been carried on in the last few years and very promising deposits of galena have been found in the Chats Island group by the James Robertson Co., Ltd., of Montreal. The chief deposit is that of the Galetta or Kingdon lead mine, located on Chats Island in the Ottawa River about 5 miles east of the town of Arnprior.

The rocks¹ in the vicinity of the mines consist of an interbedded series of crystalline limestone and biotite gneisses of pre-Cambrian age. This part of the Ottawa River basin has been severely faulted and the overlying Paleozoic limestones may be seen in normal fault contact with the pre-Cambrian. The vein filling is of a very well-marked fault fissure type and although the amount of displacement is not determined at this point, similar faults to the east show a displacement of from 1,500 to 1,800 ft. which would indicate mineralized fault fissures of considerable depth.

The ore occurs in a highly crystalline calcite with some barite and fluorite, and the galena occurs more or less richly disseminated in clusters and crystal aggregates. In some cases these masses weigh several hundred pounds. Small amounts of sphalerite are occasionally encountered.

* Superintendent, St. Louis Smelting & Refining Co.

¹ Report of the Ontario Bureau of Mines.

The workings at the east and west ends of the vein are about 1,000 ft. apart and the vein width frequently reaches 10 ft. Although workings have not been carried to any great depth, sufficient development work has been done to disclose a large orebody.

Early in 1916 a small concentrating plant was placed in operation. Owing to the coarseness of the galena grains and the highly crystalline character of the calcite, a high-grade product is obtained with a low percentage of lead in the tailings.

The following is an average analysis of the concentrates which consists of about 50 per cent. jig product and 50 per cent. table concentrates:

Ag	Pb	Insoluble	Fe	CaO	Zn	S
Ounces	Per Cent.					
1.14	79.0	0.60	1.20	0.80	2.00	14.40

On account of the absence of lead smelting facilities in Eastern Canada, it was decided, early in 1916, to build a smelter on the property as quickly



FIG. 1.

as possible as the concentrates were piling up and the demand for lead was great. The writer was called upon to furnish plans for the work and decided that a mechanical hearth plant would be the only solution both from the standpoint of first cost of installation and operating expense. The concentrates being high-grade and rather coarse they are an ideal product for hearth treatment.

Active construction was begun in July and although considerably hampered by lack of building material, the first hearth was placed in operation on Oct. 9, 1916. Since that date the hearth has produced about 15 tons of lead daily. From 2 to 3 per cent. of fine crushed limestone is mixed with the galena as fed to the hearth and about 3 per cent. of coke breeze has been found sufficient to maintain the correct smelting temperature.



Fig. 2.

Figs. 1 and 2 show the general arrangement of this simple but effective plant. The main hearth building contains the latest type of 8-ft. hearth, drossing and molding kettle and ore-, coke- and coal-storage bins. In order to secure a quick drop in flue temperatures and separation of the coarse particles of dust, four goose-necks in parallel are placed immediately back of the hearth. This sudden lowering of the temperature makes possible a short flue, 300 ft. long, balloon-shaped with clean-out hoppers every 4 ft. This flue is housed to equalize the temperature in summer and winter and in connection with a No. 10 Sirocco exhaust fan delivers about 8,000 cu. ft. of dust-laden gas per minute to the bag house. Below the thimble floor the bag house is divided into three compartments each connected to the damper flue by gas-tight dampers, thus any compartment can be cleaned without interrupting the operations. Above the thimble floor are 99 cotton bags, 18 in. in diameter and 30 ft. long. These bags are shaken from the outside by the efficient Murray bumping system. The temperature of the gases in the bag house does not exceed 180° F.

An industrial track runs under the goose-neck hoppers and flue system and by means of a closed dust buggy the dust may be conveyed to the pugging room without mechanical loss or endangering the health of the workmen. This dust is pugged with a small quantity of burnt lime and together with the burnt bag-house product is bedded with the concentrates.

A total of about 15 hp. is used in the plant and as cheap electric current is obtainable in this district a power plant was not required.

The hearth is worked by two men each on the three 8-hr. shifts, each shift producing about 100 pigs of 100 lb. each. The work is paid for according to the weight of pig lead made, 6 c. per 100 lb. being paid the charger and 5 c. to the helper. This is a direct labor cost of \$2.20 per ton of lead produced. A total of 14 men per 24 hr., including the man in charge, is required to handle all the smelter operations and when the low labor, fuel and power items are considered it can readily be seen that the total cost per ton of lead produced is low and could not be approached by any other process.

As the smelting is paid for according to the metallic lead produced, the only cost in connection with the fume and dust is that of transporting it back to the hearth bins, which is a small item.

The lead pigs from the hearth are melted down in a kettle, skimmed clean of dross and molded into trade bars. The dross is returned to the hearth.

The following figures show the average metallurgical outcome per 24 hr.:

	Pounds
Ore charged (dry weight).....	51,100
Lead contents.....	40,369
Pig lead made.....	29,873
Gray slag made.....	11,815
Dust and fume (73 per cent. lead).....	7,742
	Per Cent.
Coke breeze used.....	3

The percentage of the total lead in products is approximately as follows:

	Per Cent.
Pig lead.....	74
Gray slag.....	12
Dust and fume.....	14

Analysis of Gray Slag

Pb	Insoluble	FeO	CaO	S
Per Cent.				
41.0	11.2	13.0	11.6	1.7

Analysis of Lead

Ag	Pb	Cu
Ounces	Per Cent.	
5.06	99.95	0.03

As the amount of gray slag produced is small, it is saved up and periodically shipped to a distant blast-furnace plant, the low sulphur contents and its self-fluxing qualities making it a desirable blast-furnace material.

With the installation of larger concentrating facilities, in the near future, additional hearths will be erected, and at that time provision will be made to smelt the gray slag at the plant.

The lead loss in this plant will not exceed $1\frac{1}{2}$ per cent. and will be principally confined to the blast-furnace loss in smelting the gray slag. The sanitary conditions of the furnace room are perfect and no plumbism is likely to occur in that department.

Several other mines in this district are developing well, and no doubt in time southeastern Ontario will occupy a creditable position among lead producers.

As Canada produces only about 60 per cent. of her lead requirements, the government is encouraging the production of refined metal from Canadian ores by placing a bounty thereon and this will no doubt stimulate the exploration and development of this field as will also the close proximity of a lead smelter which will eliminate the heavy freight charges of the past.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Mine Models

BY H. H. STOEK,* E. M., URBANA, ILL.

(St. Louis Meeting, October, 1917)

MINE models have three distinct uses:

1. As exhibits in expositions and museums.
 2. As exhibits in law suits.
 3. As illustrations in teaching mining engineering.
- All three uses are in a sense educational. The third, or distinctively teaching function, has not been extensively developed in America as it has been abroad, chiefly because of the cost of the models and the scarcity of model makers.

In connection with expositions, such as the Columbian, the Louisiana Purchase, and the Panama-Pacific, a number of very good mine and metallurgical models were prepared with funds furnished by State commissions, by mining organizations or mining companies. At the close of these expositions, some of these models have been deposited in National or State museums or in educational institutions, but too many of them have been lost, or have been stored away in some mining company's office or in the basement of some State building, where they have been lost sight of and have gradually fallen to pieces.

A visitor to a German mining school is impressed by two facts:

1. The mining laboratories are in general inferior to those in America.
 2. The mining museums are infinitely superior to ours, particularly in models exhibited.
- One may reply to this comparison by saying that a museum is necessarily a work of time. From the historical standpoint this is, of course, true, but it is not necessarily true in connection with a working museum intended to illustrate current practice. For instance, the oldest mining school in Germany, that at Freiberg, has much excellent historical material, but very little on modern practice, while Berlin, one of the youngest German mining schools, has a most extensive and valuable collection of modern appliances and models. This is true in spite of the fact that one of the best-known model shops is at Freiberg, from which many of the models to be seen in America have come. Because many of the models in American mining schools have come from

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this shop, they mainly illustrate ore-mining rather than coal-mining practice. Also, the excellent models put out by the Engineering Model Works of Butte, Mont., represent almost entirely geological problems or ore-mining practice.

Because it has been impossible to obtain foreign models and on account of the prohibitive charges of model makers in the Middle West, who confine their attention mainly to mechanical subjects, the Mining Department of the University of Illinois is attempting to develop in its own shop a series of models to illustrate the typical methods of mining. Thus far those built have referred to coal-mining practice and have been constructed mainly of wood. This material offers certain difficulties not met with in connection with plaster models, and as some of these difficulties seem to have been overcome, the following description is offered in the hope that it may possibly assist others who are attempting to solve the educational model problem and also that others may give their experiences along similar lines.

Baseboard.—It was foreseen that in models built of wood contraction and cracking would be serious items, as was illustrated by a wooden model bought in Freiberg some years ago, and which, after being at Illinois for 25 years, still contracted. An effort was, therefore, made to build up a base for the model upon the principle of a well-constructed drafting board. The foundation for the relief portion of the models consists of a baseboard of kiln-dried soft pine, 5 ft. square, made up of 6 by 1½-in. stock held together by four 1¾ by 3½-in. oak strips, evenly spaced. Considerable trouble was experienced at first by the board shrinking and cracking, and in an effort to obviate this, slots were provided in the oak strips through which the screws pass to the baseboards proper, washers being placed under the screw heads. This construction permits a movement or sliding of the board components upon the strips, and on several of the models some shrinkage has evidently taken place without cracking. On one of those first constructed, however, a few slight cracks have appeared, but these can be easily filled with putty, and repainted without injuring the model. The climate in central Illinois is very hard on any construction of this nature.

Upon the top of this large baseboard was placed a blue print of the layout of the mine workings, each pillar being numbered. Then by placing carbon paper underneath the blue print and tracing over it, a reproduction of the mine plan was left on the board, all parts being numbered as on the blue print.

The "coal" pillars and coal in place were built up of kiln-dried white pine, the thickness of the board depending upon the thickness of the seam and the scale of the model. It is usually necessary to exaggerate the vertical scale. The blue print plan was pasted on the pine board and the numbered small blocks representing the pillars and solid coal

cut out with a jig saw. The large board was first cut up into smaller sections for easy handling.

The separate pieces were then given a coat of hot cabinet glue on the bottom, placed in position on the baseboard according to number, and at least two $\frac{5}{8}$ -in. brads driven through each block and countersunk. The larger pieces required more nailing to hold them down. With a pocket knife the rooms and entries were trimmed, so as to make them as realistic as possible. After this trimming, the whole surface was



FIG. 1.—PILLAR-DRAWING MODEL AND MODEL FRAME.

sandpapered down to a uniform level. A large block covered with sandpaper was used for the first models, but later a revolving circular sandpaper-covered block propelled by an electric motor and flexible shaft reduced the labor and time by at least two-thirds.

The entire surface, including the underside of the baseboard, was then given a coat of drab paint, followed when dry by a coat of black on the upper surface. Nail holes were then puttied and smoothed.

The appearance of bituminous coal was secured by first applying a coat of black paint on top of the relief portions only (but not to the sides and bottom of the rooms and entries), then sprinkling upon the fresh

paint a layer of coal dust. Anthracite dust proved the best to give a permanent luster and uniform color. When the paint had dried sufficiently any superfluous coal dust was dumped from the board, leaving a smooth layer firmly fastened to the surface of the relief and giving it a very satisfactory coally appearance.

Overcasts were made of sheet iron, formed into a rectangular hollow section by soldering covers on troughs, cutting to required lengths, and filing to the exact shape desired. These overcasts were glued on wooden abutments and the whole painted drab to resemble concrete.

Worked-out sections, where roof had caved, were filled nearly to the level of the top of the coal seam with broken shale firmly glued together.



FIG. 1a.—NEARER VIEW OF PILLAR-DRAWING MODEL SHOWN IN FIG. 1.

The light portions in Figs. 1 and 1a, where the pillars have been drawn, show the contrast between the broken rock and the coal.

In the longwall model (Fig. 2) the pack walls were built up for half their vertical thickness with wood strips, which were then veneered with thin flat pieces of shale of fairly uniform size and thickness, that were glued to the wood strips. At first it was very difficult to make the glue stick permanently to the wood, for with the shrinkage of the board, sections of glue and shale would loosen and pop off. This was overcome by cutting holes in the wood strips by means of a double tracing wheel with blunted points, thus giving the glue an additional anchorage. A number of schemes for the rapid laying of these pack walls were tried, but the only effective way to produce a natural-looking wall was to lay the pieces one at a time with tweezers, suitable pieces being first selected

from the mass of crushed stone and placed in a shallow pan. This was a slow process but results have justified the method. Gob between the pack walls was filled in by pouring large and small fragments into a layer of glue.

Mine track is represented by tacking along the entries and in the rooms copper wire screen cloth cut into strips one square wide. As screen wire cloth is not soldered, it will fly to pieces when cut in narrow strips. This was avoided by dipping wider strips into molten solder before cutting. Tracks were placed before the coal dust was applied. A neat method of fastening down tracks, which was used on the last two models, was to drive nails with solder-dipped heads flush with the bot-

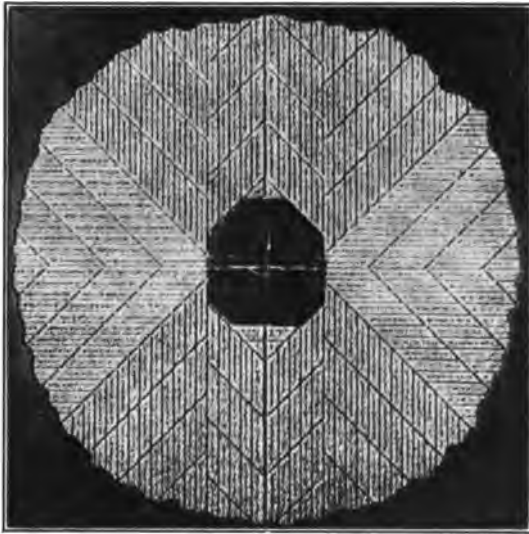


FIG. 2.—LONGWALL MODEL.

toms of entries and about 4 in. apart. A hot soldering iron was then used to unite the track with the nail heads. Curves were bent to required radius before setting.

Arrows placed in the entries show the direction of the air currents, white arrows representing fresh air courses and red arrows return courses. Doors and regulators are made of thin wood, painted. By attaching pins to the overcasts, doors, regulators and ventilation arrows, they can be made temporary and movable and different methods of carrying the air can be illustrated, so that the model can be used instead of a map for working out with a class problems in ventilation.

The frames are of oak picture molding, about $4\frac{1}{2}$ in. wide, supplemented by oak pieces placed at right angles to the molding so as to hide the baseboard. These sides are mitered and secured by brass screws.

A title is placed at the top of each frame.

The supporting stands, Fig. 1, are built of 1-in. pipe, have double-wheeled ball-bearing castors, and are equipped with a quadrant and

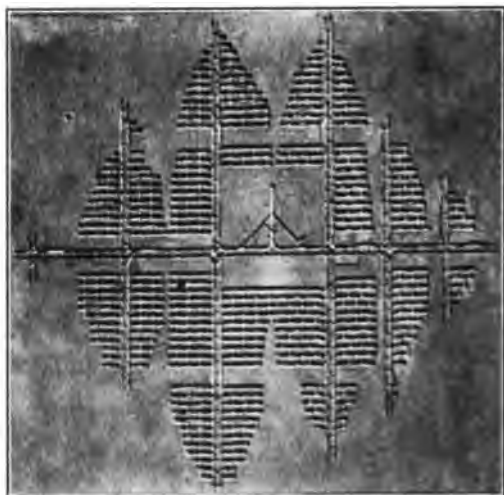


FIG. 3.—ROOM AND PILLAR MODEL.

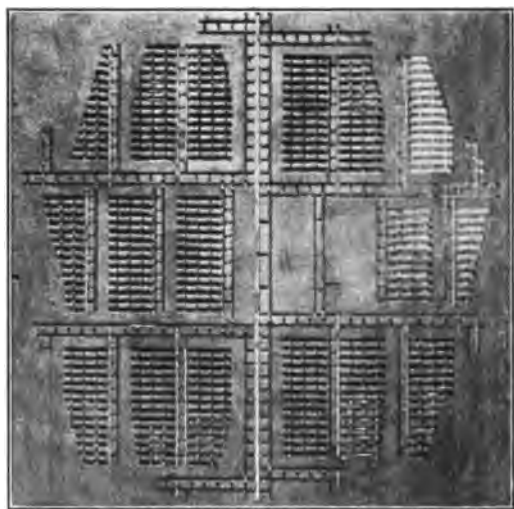


FIG. 4.—PANEL ROOM AND PILLAR MODEL.

hand screw, as shown, so that the model may be inclined at any desired angle—from horizontal to nearly vertical. When inclined, the width over all is such that the model can be taken through any door in the mining building.

In the development of these models great credit is due L. S. Baldwin, at present instructor in engineering drawing at the University of Illinois, who as an undergraduate student at the University did all of the detailed work on the models and developed most of the methods of construction, and also to H. J. Vanderbeek, mechanician of the Mining Department, University of Illinois, who designed and built the supporting stand for the models.

Thus far four models have been built to illustrate the three typical methods of coal mining, namely, simple room and pillar, Fig. 3; panel room and pillar, Fig. 4; longwall, Fig. 2; and the method of drawing pillars in the panel system where narrow rooms and wide pillars are used, Fig. 1.



Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

Discussion of the paper of MAX ROESLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1789 to 1839.

THE CHAIRMAN (WILLIAM KELLY, Vulcan, Mich.).—Some 8 or 9 years ago I was at Daiquiri, only a few miles east of Firmeza, where there are deposits of iron ore of the same general character. There is a great conical hill perhaps 800 or 1,000 ft. in height and the ore deposit appears to be somewhat in a canoe shape on one side of the hill. The mining has been done by quarrying the ore in benches. These are called levels and to go to the 10th level it was necessary to take mules and ride to the top of the hill, which was to me an upside down arrangement of numbering levels and a novel method of entering and viewing a mine.

A little copper ore can be seen in places, and not being a geologist I thought the deposit might possibly be a gossan, but none of the geologists who have written on the subject approve of this hypothesis and the great question seems to be what part, if any, the limestones have played in the origin of ore; whether these deposits are replacements of limestone and whether they are contact deposits between the limestone and the other rocks. Examinations have been made by many eminent geologists and the problem has been discussed vigorously on both sides.

The deposits are exceedingly interesting from a commercial standpoint and although they have been studied for a number of years the origin of the ore is still a difficult and somewhat controversial question.

J. T. SINGEWALD, JR., Baltimore, Md.—I have had the privilege of visiting these deposits and have been very much interested in this paper of Dr. Roesler's; having been on the ground, I can easily appreciate the difficulties under which he labored in attempting to work out the geology of that region, and he certainly deserves considerable credit for the way in which he has deciphered the geological relations of the Firmeza district.

There are several points in his paper I would like to discuss; first, in regard to the geological interpretations. His differs from previous interpretations in the relative age of the diorite and the granite, but you will note that Prof. Lindgren suggests the probability that those two rocks are very closely of the same age, and Prof. Kemp actually suggests, although he does not accept the idea, that the granite is probably later than the diorite, so that there seems to be no particular reason

to doubt the evidence on this point and therefore to hesitate to accept Dr. Roesler's conclusion in that regard.

One of the most important features of his paper is the interpretation that he gives to the aplitic dikes. Prof. Lindgren and Prof. Kemp in their papers mention the occurrence of these dikes, but do not lay any particular stress on them, and in my own notes on the mines I have mentioned their occurrence, but neither Dr. Miller nor I at the time of our visit to the district attached any particular significance to these dikes, so in this respect Dr. Roesler's paper is a decided change in the geological interpretation.

I agree with Dr. Roesler in his interpretation of the age of the relations of the magnetite and the hematite; that is, that the hematite and the magnetite are both due to the primary mineralization, except in so far as some little hematite has developed as the result of later alteration.

The point of greatest interest, of course, in connection with this paper is the interpretation of the genesis of the deposits. Those of you who were present last year when these deposits were under discussion will remember the amusement that was created by the remark that everyone who had visited these deposits had come to a different conclusion as to their genesis, and now, as Dr. Roesler predicted at that time, he is bringing forward a still different interpretation. Although he has presented his evidence in a very able manner and though undoubtedly there is much to be said for his interpretation, I still feel inclined to adhere to the interpretation that Dr. Miller and I gave last year to these deposits.

The broad geological features of these deposits have been very succinctly stated by Prof. Kemp, in a single sentence:¹

"They furnish a variety of deposits ranging from those of small size in streaks in limestone associated intimately and microscopically with quartz, garnet, and epidote; through larger and more extensive developments of the same with the practical extinction of the limestone; to the extreme of ore developed in great tabular masses in diorite, but still associated with the same quartz, garnet, and epidote, both in the large and in the microscopic way."

Looking at the deposits from that viewpoint, one hardly hesitates to ascribe to them a contact-metamorphic origin, and the burden of proof would seem to lie upon him who would apply a special explanation to those of the deposits in which evidence of the original presence of limestone is lacking. Lindgren and Ross did not hesitate to ascribe such an origin to all of the deposits, but I think they made the mistake of not realizing to what extent the igneous rock itself had also undergone transformation. Prof. Kemp, on the other hand, found considerable difficulty in explaining such an ore mass as the large Lola Hill deposits at Daiquiri as due to the alteration of an included block of limestone, and conse-

¹ *Trans.* (1916), 53, 36.

quently ascribed the whole mineralization in the case of that deposit and in the case of one or two others, to the alteration of the diorite. But, as we pointed out, there is a great deal of garnet and calcite at the southern end of that deposit, and, just beyond the mine workings, actually a limestone out-crop. The fact that this deposit is terminating at a comparatively shallow depth would indicate that originally some sort of rock was present where it is now localized, which was far more susceptible of replacement than the diorite underlying it, through which the mineralizers had to come.

These three theories, those of Lindgren and Kemp and the one advanced by Dr. Miller and myself, differ only in details and are very much alike otherwise. Several objections that Dr. Roesler advances against these theories are as follows:

"The conception of the orebodies as metamorphosed, engulfed limestone masses is difficult. The shape and position of the ore masses demand that the entombed blocks should have been comparatively thin and that they should have come to rest in every conceivable position." This is actually the shape of many of the limestone blocks, as is admirably shown in the Ocania mine, where there are plainly exposed several narrow slabs of limestone standing at a high angle within the diorite, and between two of these slabs and with the same shape and position is the Ocania orebody. The suggestion at once is that this orebody represents simply an alteration of one of those limestone blocks. The fact that this theory would require that the limestone masses should have "come to rest in every conceivable position" is just what one would expect if they are actually engulfed blocks of limestone in the diorite. Another objection he advances is that "this theory completely fails to account for the gradual transition from ore to diorite." He completely overlooks the endomorphic transformation that the diorite underwent at the same time the limestone was undergoing its alteration. A further objection is to placing "the burden of producing the orebodies upon the diorite" and that "Singewald and Miller regard the diorite as the source and as the cause of the mineralization." I think herein he slightly misinterprets what we really had in mind. The old theory that the mineralizers producing contact-metamorphic deposits were derived from the igneous rock immediately in contact with the orebody has long since been abandoned and it is generally agreed among geologists who have studied contact deposits that the mineralizers were drawn from the same parent magma from which the contact igneous rock itself came. In other words, we ascribe the source of the mineralization to the same parent magma postulated by Dr. Roesler. The divergence in our views is as to how and where the magmatic emissions deposited their metallic contact. Finally the statement is made: "One of the striking features of the larger Firmeza deposits is that the ores have almost obviously avoided the masses of

marmorized limestone." Dr. Roesler describes 11 mines in the Firmeza district, and five of those are either directly or very closely associated with limestone. When we bear in mind that the engulfed blocks of limestone make up a very small proportion of the rock mass as compared in volume to the diorite, this seems rather to indicate that the orebodies as far as possible really sought out the limestone blocks. If you further accept our assumption that where the limestone is not present now, it is due to the fact that it has been completely altered, it would seem almost inevitable to conclude that there was a distinct relation between the limestone and the ore.

The absence of ilmenite in the Firmeza ore is used as an argument to connect the mineralization with the granitic phase of the original magma rather than with its earlier dioritic representative, and in support of this, Dr. Roesler quotes a statement from Vogt in which he says that the titaniferous ores are associated with basic rocks and the non-titaniferous with acidic rocks, but this statement applies to "Erzaussonderungen", that is to magmatic segregations, and has no application to pneumatolytic or hydrothermal deposits, and hence to the deposits in question. I have paid considerable attention to the titaniferous iron ores and recognize the almost universal application of Vogt's rule to the magmatic segregations. A notable exception, to it, however, is the Tofo mine of the Bethlehem Steel Co. in Chile, which, according to my interpretation, is a magmatic segregation of non-titaniferous magnetite in igneous rock which ranges from dioritic to more basic in composition. That the rule does not apply at all to deposits other than magmatic segregations is strikingly illustrated by the deposits of Cornwall, Pa., which are non-titaniferous but occur in limestone associated with diabase and presumably derived from the diabasic magma; and by the fact that the only known example of titaniferous magnetite of contact-metamorphic origin is one of no commercial importance at Cebolla Springs, Gunnison Co., Colo., which I described a few years ago merely because it was an exception to the statement by Beyschlag, Krusch and Vogt that contact-metamorphic magnetites are never titaniferous. The non-titaniferous character of the Cuban ores affords no argument, therefore, for either theory.

In support of his own theory, Dr. Roesler cites the constant relationship between the orebodies and the aplite dikes and granite apophyses. In so far as the relationship exists, it is not surprising, you would even expect it in accordance with the other theories: that is, if the mineralizers and the aplitic dikes came from the same parent magma it would be only natural to suppose that the area of mineralization also coincided more or less with the area in which the aplitic dikes were intruded. But the relationship between the aplitic dikes and the orebodies does not seem to be intimate enough to make them competent vehicles for the mineralizing solutions, though I am not prepared to say that they had nothing to

do with it at all. If you take an orebody, such as the Lola Hill deposits, in the Daiquiri district, the amount of aplitic material is not great, nor are the aplitic dikes connected in such a way with the orebodies as would make one think that they had been the channels used by the mineralizers in producing the orebody, and if they were, one would expect the ore to be coextensive with the dike itself, and the objection, that I pointed out in the case of Prof. Kemp's interpretation of this deposit applies here—that the orebody pinches out in depth. Roesler himself, in determining the economic conclusions that might be drawn from his interpretation, says: "Since the mineralization came from below, from the granite massive, there is no reason to believe that the ore will not extend at least to the granite." The largest orebody of the region appears to terminate short of that depth.

To sum up, points of similarity in all the recent theories are that they ascribe the mineralizers to the same parent magma from which the inclosing diorite was derived and that the mineralization was pneumatolytic or nearly so. I differ from Dr. Roesler in not feeling it necessary to ascribe to the aplitic dikes the rôle of vehicle for the mineralizers. I also feel that the restricted localization of the orebodies makes it necessary to postulate special conditions at the points of localization. In many instances, the relations of the ore to limestone afford unmistakable evidence of such special conditions. The assumption that the presence of limestone has determined the localization in all cases seems, therefore, reasonable.

BENJAMIN L. MILLER, So. Bethlehem, Pa.—Dr. Singewald has well expressed the ideas which we formed of the Firmeza district while in that region. I wish to call attention to only a few additional points.

On page 1793 occurs the following statement: "The limestone cliffs, with their sea-cut terraces, have been interpreted as evidence of periodic movements of the island, but it seems to the writer that the widespread occurrence of a terrace about both Jamaica and Cuba at the same elevation above sea level indicates a movement of the sea surface, rather than of the land."

Far larger areas than those mentioned would have to be investigated before one should conclude a general lowering of the sea surface. While there are many well-developed sea-cut terraces along the Atlantic Coast of the United States, it is doubtful whether these can be correlated with those existing in Cuba and Jamaica. It is certain that uplifts and depressions of large areas in which there was little warping have taken place in the West Indies as pointed out by Dr. T. W. Vaughan, but the effects of such movements are much less extensive than terraces which might result from a general withdrawal of the ocean waters.

In regard to the origin of the orebodies, I feel, as do a number of other persons who have been in the region, that the exact origin of the

orebodies is still somewhat in doubt. All of the facts necessary to reach a definite and positive conclusion have not yet been obtained. With the evidence at hand an investigator may place particular emphasis upon certain points and minimize other lines of evidence and so reach conclusions quite at variance with views formerly held. This is what has been done thus far. The region remains an interesting one for the geologist and no doubt diverse views regarding the processes which have taken place in the formation of the orebodies will be held for a long time as there is little reason to believe that the necessary evidence for a final decision will be soon obtained.

I think Dr. Roesler has performed a very important piece of work. He has collected much valuable information, has outlined the problem in a commendable manner, and has furnished considerable evidence for the conclusions which he formulates. It is no fault of his that his views are not accepted in their entirety. In some particulars the speaker must dissent.

Opposed to the view which Dr. Roesler expresses concerning the fissures or fractured zones through which the ore solutions have ascended is the sharp delimitation of some of the orebodies in depth. If the heated solutions rose through such fissures, replacement of the country rock would seem to have taken place throughout a zone of greater depth and the lower limits would be much less sharply defined.

The shape of the orebodies seems to me to be much as one might expect if there had been engulfed masses of limestone that initiated the precipitation of the ore minerals. In many places the process continued even after the limestone had all been replaced and portions of the orebodies represent replacement of the diorite.

The fact that granite and aplite dikes have not been observed in other places in Cuba where similar orebodies have been observed, as mentioned by Mr. Cumings,² also seems to throw doubt on the intimate relation between the orebodies and the aplite dikes which Dr. Roesler maintains.

JOHN D. IRVING, New Haven, Conn.—I have not seen these deposits, but when Dr. Roesler prepared this paper I had the pleasure of going over it in very great detail with him and saw all his specimens and know some reasons why he arrived at his conclusions.

In the first place, these iron-ore deposits with their contact-metamorphic minerals, garnet and epidote, and with the magnetite associated with them, with the neighborhood of the limestones in the vicinity, at once suggest to anybody who takes them up that they belong to the ordinary category of contact-metamorphic deposits, one of the limestone masses that have been more or less completely altered as the case

² *Bulletin* No. 123 (March, 1917), 375.

may be. In that conception there is nothing that does violence to any of our previous ideas in regard to contact-metamorphic minerals. Everything fits in beautifully.

I remember the first time that I had opportunity to learn anything about these particular deposits. A former student of mine, Dean Corsa, went down there and made a careful examination and wrote a little paper which was not published because it was never completed. I said, "The only thing I do not like about your paper is that it seems to me that these things must be contact-metamorphic deposits in limestone, and you do not believe it," so that my whole idea was in favor of that interpretation because it was the customary interpretation of the conditions which existed or are said to exist there.

Dr. Roesler brought his material back and I said to him, "The first thing you want to be careful about is to see that these things are not contact-metamorphic deposits in limestone masses, because that is what I believe they are." So that you can see that in supporting Dr. Roesler's point of view, I have been personally persuaded that the contact-metamorphic idea is not correct.

Now there are two questions, it seems to me, involved in this matter. The first is, is it possible for the solutions which arise from these granite pegmatite dikes which have been given off by them or for solutions in general in that region to have altered the dioritic rocks into contact-metamorphic minerals? And the other question is whether or not that had anything to do in forming these.

In regard to the first one, I think the most significant thing is a photograph of a small dike which Dr. Roesler has in his paper on page 1824, Fig. 19, in which he shows a granite band between aplitic granite and diabase porphyry. That was the thing that convinced me that these solutions were capable of producing contact-metamorphic minerals without the assistance of any limestone in the vicinity of the dike. The picture does not give as good an idea as the dike itself. At the edges of the dike is a beautiful layer of garnet which makes its way out in the dioritic rock to a short distance and is more or less intergrown with it and appears to be a replacement or alteration of it. I can see no conceivable way in which such a band, continuous over a considerable area of the exposure of that dike, can have been developed in any way except by alteration of the dioritic rock itself.

It is not an ordinary case of contact metamorphism, because there is something added that is not in the diorite. So we must look upon it as a contact-metamorphic alteration from the dioritic rock. If that is true (I can see no escape from it) why is it not possible that that same diorite would have been able to produce larger masses of contact material without the necessary agency of limestone? Is it correct that we should think these limestone masses have necessarily been the determining

feature in this occurrence just because we have in other places found them to be?

It seems to me the most significant feature is the arrangement that Dr. Roesler suggests in his letter, where we have larger dimensions in horizontal and in two vertical directions intersecting one another.

It seems as if there were a series of rocks, and that would give rise to this linear arrangement and this development in certain directions. Of course, it is possible that limestone masses can be engulfed in such a way that they can simulate conditions which would result from a certain distribution of fissures and therefore that seems to me to be a strong point.

L. C. GRATON, Cambridge, Mass.—Without attempting to discuss the facts or the arguments in this interesting paper, because they relate to a deposit of which I have no first-hand knowledge, I am nevertheless impelled to refer to one feature of the paper in its relation to two specific topics.

It is to be regretted, it seems to me, that this much-discussed question of the origin of these Cuban iron-ore deposits, and especially their relation to the limestone, appear not to have been settled by Dr. Roesler's paper. If it could have been established to the satisfaction of all that these iron ores, so similar to contact-metamorphic deposits, have, nevertheless, been formed directly and solely by igneous influences and with the aid of intermediation of limestone, then it would be easy and reasonable to believe that true contact-metamorphic deposits rich in iron have derived that metal from igneous emanations, rather than the view, which some have advanced, that the iron is simply a recrystallization of ferruginous impurities in the limestone beds.

My own feeling in reading Dr. Roesler's paper is that his failure in some directions to carry complete conviction is due to the brevity of his presentation. I feel that back of the conclusions he has here recorded he probably has a repetition and an accumulation of evidence which, in my opinion, it would have been better to present than to omit.

The second point in which I was especially interested received similarly abridged treatment. I refer to the question which occupied so prominent a part in the discussion of these Cuban deposits at the Institute Meeting a year ago;³ namely, whether the hematite was derived from the magnetite by normal oxidation near the surface, or whether both iron oxides are products of the initial ore-forming processes. Prof. Irving tells me that he feels that Dr. Roesler has settled the question definitely, at any rate for these particular deposits, and I am glad to hear Dr. Singewald's opinion, likewise, that the hematite is in the main a primary

³ See discussion of paper by Lindgren and Ross, *Trans.* (1916), 53, 59-66.

constituent of the ore. But especially in view of the fact that no longer than a year ago Dr. Roesler was inclined to the belief that the hematite is a superficial oxidation product,⁴ it is, I think unfortunate that he has not now presented more fully and convincingly the evidence that has caused his view to change.

It is to be hoped that Dr. Roesler may be prevailed upon to devote further attention to this particular problem, and by means of microphotographs and extended discussion bring more light to bear upon this question which it seems to me is one of considerable importance.

C. P. BERKEY, New York, N. Y.—I think I ought to say one or two more words before allowing this matter to drop. I see quite clearly that Dr. Roesler has not completely silenced everybody, but it does seem to me, as the last speaker suggests, that Dr. Roesler has written too brief a paper to clear up everything that we may wish to know. Evidently he has not given all of his facts, although he is clearly satisfied and convinced in his own mind. He writes, for example, in a recent letter, as follows:

"How a bunch of entombed limestone blocks came to assume such a regular alignment as I have spoken of is not explainable, it seems to me. Besides, I have found all sorts of specimens of mineralized diabase with the retained diabasic structure; and, after living with these deposits for 6 months, I feel that I know that in 95 per cent. of the cases limestone had absolutely nothing to do with the matter."

That is a pretty strong statement from a man who has been on the ground 6 months and has been successfully guiding explorations. Some of the objections to Dr. Roesler's interpretation relates to the connection of the ores with aplites. One objection raised against his interpretation is that there is no aplite in sight at some of the ore pits. It does not seem to me that this is a very serious matter. He does not claim that aplite must be in immediate touch with these ores; it is only that there is a genetic association with these rocks as end-products from the same magma. The solutions or emanations which produced the ores seem to have been able to penetrate the surrounding rocks farther than the associated matters now forming the aplitic dikes, which may be only the residual acid contents of the original intrusions.

I take it from what I have heard today that everybody admits that the mineralizing solutions have come up through weaknesses in the rock. I believe, also, everybody has admitted that the diorite is mineralized. If it is mineralized, and if some of the deposits are in that kind of rock, there is no necessity of attaching particular importance to limestone at all. This is very nearly in agreement with Prof. Kemp's point of view. He says there are two types of deposits: those associated with limestone, constituting contact-metamorphic deposits of rather common sort; and others

⁴ *Loc. cit.*, 66.

that are mineralizations of diabase. Neither Dr. Roesler nor Prof. Kemp consider the wall-rock type at all essential.

It seems to me that the situation is fairly well defined, so far as Dr. Roesler's views are concerned. The guiding feature is the fractures and these are reasonably systematic. Physical conditions have controlled and guided the mineralization, rather than the chemical composition or nature of the rock. These ores have been deposited at points where their activity or carrying capacity has been reduced so that crystallization or deposition must begin; and mineralization has resulted wherever and whenever such conditions have been reached, no matter what the rock was. If diorites formed the walls, then diorites were affected and the deposit was made.

On the question of the hematite, he says there are two types—primary hematite intergrown with magnetite, and secondary hematite which is a simple superficial and earthy product. Where the hematite which he considers a purely secondary product is developed, the sulphides are destroyed. Elsewhere the sulphides still remain. These views look reasonable to me.

The Manganese Ores of the Lafayette District, Minas Geraes, Brazil

Discussion of the paper of JOSEPH T. SINGEWALD, JR. and BENJAMIN LEROY MILLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1745 to 1762.

L. C. GRATON, Cambridge, Mass.—I should like to ask the authors if they attempted to draw any analogies with the manganese deposits at Franklin Furnace, N. J., and whether there are any traces of zinc in these Brazilian deposits.

J. T. SINGEWALD, JR.—We did not think of them at all as analogous to the Franklin Furnace deposits and there is no appreciable percentage of zinc present. If these ores do contain zinc it could be there only as a fraction of a per cent.

F. LYNWOOD GARRISON, Philadelphia, Pa. (communication to the Secretary*).—There is no disputing the fact that the manganese ore supply situation in this country demands serious consideration and is a matter of fundamental public concern. We are consuming this material at a steadily accelerating rate and there is no hope for a proportionate increase in the domestic supply.

In the year 1916, this country imported 560,000 tons and produced only 27,000 tons from our own territory. In 1915, the local production was 9,709 tons, which on its face shows the American output to have tripled during the past year. Those of us who are familiar with the

* Received Feb' 26 1917.

character and scope of the manganese deposits in this country, however, know perfectly well that these figures are deceptive and that no such ratio of increase can be expected and maintained. It therefore behooves us to give careful and immediate consideration to the sources of foreign supply now available to us, and at the present moment this means predominantly Brazil as long as the war endures.

As is well known, steel cannot be made in large quantities without the use of ferromanganese alloys, and in view of current political and economic events, it is plain that were our ports blockaded or our ships denied the freedom of the seas, the steel production of this country would necessarily be seriously affected if not practically paralyzed, for, unlike Germany before the war, it does not seem that we have accumulated any considerable reserve of manganese ores and, as I have indicated, our domestic yield last year was about $4\frac{1}{2}$ per cent. of the consumption.

Prior to the war, we imported large quantities of ferromanganese from Europe. Inasmuch as both England and Germany have practically no domestic supply of manganese ores, like ourselves they had to depend upon importations chiefly from India and Russia. It is evident, therefore, that the situation in England and France is serious and in Germany must be desperate, for it is known that her reserves of manganese ore have long since been consumed and, failing the discovery of some suitable substitute as a deoxidizer in the process of steel manufacture, that country today must be face to face with a problem of the utmost gravity in the consequently necessitated restriction of her steel production. Naturally we hear very little about the matter, for it is a technical subject beyond the ken of the general public in all countries. Most people assume that given an unlimited supply of iron ore and coal, a corresponding production of steel is simply a question of apparatus to make it.

The United States and Canada have been combed over pretty thoroughly in hopes of finding manganese deposits of sufficient magnitude to meet our needs, but so far with uniformly disappointing results. Plenty of mangiferous iron ore has been found in the Southern States and some superficial, but never rich, manganese ore deposits in residual clays. We have nothing in this country similar to the great deposits in Minas Geraes of the Lafayette type, although there is some resemblance between our Appalachian deposits in the Potsdam sandstone or quartzite and those of the Miguel-Burnier district which are also associated with quartzites. The latter, however, are infinitely larger, although they also are evidently superficial, but nevertheless by virtue of their size and extent must contain large bodies of good, merchantable ore.

Seemingly we have no cause to hope that there will ever be found in this country any manganese deposits comparable to those of India, Russia and Brazil. Since the war, the demand for Brazilian ore has increased in leaps and bounds, and were the handling and shipping

facilities at Rio de Janeiro less archaic and primitive, the output might be greatly enlarged.

Rio de Janeiro is the only Brazilian port from which manganese ores are being shipped. About 9 or 10 years ago, attempts were made to mine and ship manganese ores from Bahia, and some small tonnage was exported, but the enterprise culminated in failure.

The existing productive districts of Brazil are today those and only those mentioned in the paper of Singewald and Miller and although quite distinct geologically they are very near together geographically. There is abundant reason to believe that the productive manganese area of this enormous country may be considerably enlarged as the result of well-directed and sustained exploration. But as yet little intelligent and systematic work of this kind has been attempted, although manganese ores have been discovered in numerous localities and the geologic data we already possess indicate the existence of many more yet to be found.

At present the Lafayette type of deposits appears to hold promise of larger ore reserves than those of the Miguel-Burnier class. From what is known of the genesis of the ores in both of these districts, it might seem that in neither case can much persistence in depth be expected of the orebodies. This would appear to be especially so with the deposits associated with and intercalated in sedimentary strata.

The genesis of the ores in the Lafayette district, as described by Singewald and Miller, follows rather closely the hypothesis laid down by Derby some 20 years ago in which he called attention to the important role limestone and calcium compounds seemingly play in the origin of the Lafayette (Queluz) ores, as well as those in the sedimentary rocks.

The intimate relations between iron, manganese and limestone beds has long been recognized. The affinity of these two metals for certain quartzose rocks such as the itabirites of Brazil, the Potsdam sandstone of the Appalachian region and the quartzites of the Dharwar series in India, is equally noteworthy. But the suggestion made by Derby as quoted by the authors on page 1750, to the effect that the orebodies of the Lafayette type present strong analogies to certain ore types in magmatic segregations, imports a new idea into the problem which it seems to me ought to receive careful consideration.

Before discussing this subject in detail, there are a few general facts connected with it which it might be well to review. Years ago Penrose pointed out that pre-Paleozoic rocks, especially the Archaen, have played a far more important role as a source of manganese than the later igneous rocks.¹

The sedimentary and residual ores of iron are fully paralleled by those of manganese, the gossan iron ores being the only ones having no true manganese equivalent.

¹ *Annual Report of the Arkansas Geological Survey* (1890), 544.

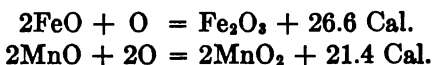
The simultaneous leaching of iron and manganese from the rocks by the same solvents in much the same way is evident.

It is also plain that although these metals are redeposited under similar conditions, the precipitation takes place more or less separately. All manganese ores contain some iron, and most iron ores carry manganese, and it is important, as a practical matter, for us to understand, as far as it is possible, the chemical reactions involved in the question.

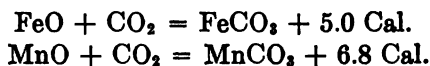
The carbonates of iron and manganese are isomorphous and are precipitated as such together. Resulting oxidation leaves the two metals associated in their relative proportions. The oxides of these metals, on the contrary, are not isomorphous and may be precipitated together only in mechanical association, as in bog ores. When iron and manganese ores occur in more or less separate deposits, different influences must have been at work from those that have thrown them down together.

According to Fresenius, the iron is precipitated first as ferric hydroxide, the manganese remaining in solution as a bicarbonate to be finally thrown down in a calcareous sinter, or, in other words, solutions of manganese carbonate are more stable than similar solutions of ferrous carbonate, and in consequence the manganese salt is carried or migrates farther, thus effecting at least a partial separation of the two metals from the same solution.

The thermo-chemical observations of Dieulafoy are in accord with the above deductions and rest upon the principle that when several reactions may conceivably take place in the same solution, *the one attended by the greatest evolution of heat will occur*. The thermo-chemical equations of Dieulafoy are important and in brief are as follows:



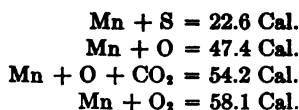
If oxygen acts on a mixture of FeO and MnO, or upon substances equivalent to them, ferric oxide will form *first* and be the more stable.



When carbon dioxide unites with these oxides, then *the manganese compound will form first and be the more stable*. If oxygen and carbon dioxide act together in considerable excess, Fe₂O₃ and MnO₂ will both be formed; but if they act slowly in small quantities, the oxygen will go to produce Fe₂O₃, and MnCO₃ can be generated at the same time. Manganese carbonate, being somewhat soluble, may then be separated from the ferric oxide by leaching, either to be deposited as carbonate or perhaps to be later oxidized to MnO₂ and CO₂.²

² The Data of Geochemistry, *Bulletin of the U. S. Geological Survey* (1911), 491, 511.

Dieulafait gives the heat of formation of several manganese compounds as follows:



Hence it appears that since the dioxide produces the most heat, it is also the most stable compound of the series and being the easiest formed becomes the principal manganese ore. Thus thermo-chemical and geological hypotheses are in harmony.

During a recent visit to Brazil, my attention was called to a manganese rock associated with granites and gneisses in a section not heretofore explored and many miles distant from the Lafayette district. On analysis this rock gave the following composition:

	Per Cent.
SiO ₂	36.04
Al ₂ O ₃	11.01
Fe ₂ O ₃	9.90
MnO ₂	37.00
Phosphorus.....	0.068

It will be noted that this material approximates closely in composition to that indicated in III, on page 1751, and to typical spessartite which according to Rammelsberg contains 36.16 per cent. Al₂O₃ and 32.18 per cent. MnO.

In the case of the ores of the Lafayette type, it is possible there may be reason to assume the manganese to be a replacement of silica rather than of lime. I have myself seen in other parts of Brazil large bodies of manganese ores associated with shales and quartzites similar to the Miguel-Burnier deposits, but no limestones, although, of course, future development may show their presence or that of calcareous shales.

In personally discussing this subject with J. C. Branner, whose work on Brazilian geology stands second only to that of Derby, he referred to an instance where he broke what seemed to be a solid lump of manganese and found that it was a fragment of itacolumite covered with a crust of psilomelane. Under the microscope a section of this core rock showed that the silica is being replaced by a brownish mineral of undetermined character which in turn was being replaced by solid psilomelane. Dr. Branner, therefore, concludes that: "in this case the ore is clearly a replacement of silica by manganese brought in in solution."

Now as to the magmatic segregation hypothesis suggested by Derby to account for the origin of certain Brazilian manganese ores: It may be recorded that a manganese rock similar to that whose analysis is given on page 1751, has been found in the neighborhood of chromic iron-ore deposits in areas composed almost wholly of granites and gneiss, which are probably Archaen.

Chrome ores are almost invariably found in serpentine rock and usually serpentines have been formed by the alteration of basic igneous rocks, chiefly pyroxenites and peridotites.

I have never heard of any chrome ores having been found in Minas Geraes, but serpentine seems to be a not uncommon rock in the manganese as well as in the iron districts.

Goodchild³ in his paper on the subject of "Laterization in Minas Geraes" speaks of serpentine boulders occurring in the gorges between the canga flats to the south of Caraca having the following composition:

	Per Cent.
SiO ₂	39.30
Fe ₂ O ₃	9.36
Al ₂ O ₃	5.94
Mn ₂ O ₄	0.65
CaO.....	2.25
MgO.....	30.71

This rock corresponds rather closely to the typical peridotite derivatives given by Kemp in his *Handbook of Rocks*, page 140. Goodchild (page 15) refers to it as outcropping in the low country at the base of the Caraca mountain in the gorges between the canga flats.

Shearer⁴ in the discussion of this paper of Goodchild, says of "the serpentine rock found in the valleys on the Morro da Mina property southeast of the Serra do Caraca", the analysis of which is above quoted, that it "appears to be an altered basic lava flow overlying the iron formation and is the only late eruptive rock in the district and covers only a few square kilometers out of hundreds, so it cannot be supposed to have any genetic significance."

I am disposed to question this remark of Shearer's, for, on the contrary, I believe the existence of serpentines among the old rocks of central Brazil is an important matter. And in this connection I will venture to quote a remark upon the subject which Branner made in a recent letter to me. "A person well posted on the geology of Brazil sits up and takes notice whenever serpentine is mentioned, for it is supposed to have some genetic relation to the diamonds. I have only seen it in one place in Brazil, and if you know of occurrences, my suggestion is that you look it over carefully."

At the present time I am not at liberty to discuss this subject at length, save to say that I have myself found serpentines in several different localities in Brazil associated with granites and gneisses, and in every instance accompanied by minerals of economic importance.

³ *Transactions of the Institution of Mining and Metallurgy* (1914), 23, 14.

⁴ *Ibid.*, 31.

Manganese Ores of Russia, India, Brazil, and Chile

Discussion of the paper of E. C. HARDER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 113, May, 1916, pp. 761 to 798.

J. T. SINGEWALD, JR., Baltimore, Md.—I might say a word further in regard to the Miguel-Burnier district. I am sorry Mr. Harder is not here. I had hoped to be able to ask him a few questions in regard to some of the features of those deposits.

This Miguel-Burnier district is the other district that I referred to that lies in the State of Minas Geraes. The interpretation that Mr. Harder gives to these deposits is the same as that given by Harder and Chamberlin in a previous paper, and that is that they consist of lenticular deposits of sedimentary origin in a series of rocks that also include lenses of limestone, but they seem to think that the manganese ores occur at a definite horizon and at a horizon different from the limestone. We found that the manganese ores occur at several horizons, as also do the limestone lenses, and the limestone masses are of the same shape as the manganese orebodies, so that our own feeling was that these manganese ores simply represent concentrations within the zone of oxidation formed as a result of decomposition and replacement of this limestone.

To the east of Miguel-Burnier there is a big bend in the railroad which cuts through, at right angles to the strike, this series of rocks. Here is shown very well both the local character of the limestone lenses and the local character of the manganese lenses in that, though the distance across the bend is only a couple of hundred yards, the limestone and manganese lenses do not match up at all, and it also shows plainly that both occur at more than one horizon. Consequently there is no reason why the manganese ore could not represent alterations of this limestone. The mine workings do not as yet give very definite information on this point in that they are all comparatively shallow and within what would be the zone of oxidation so that what they will grade into below that depth, we do not know.

On the other hand, the character of the ore suggests that it may represent such a product in the zone of oxidation in that it tends to be quite earthy; whereas, if the manganese had been there originally as such it would have undergone the same metamorphism that the other rocks have undergone and one would have expected that it would have been converted into a hard manganese ore that would today stand out pretty much as do the iron ores of that same region and not have the soft, earthy texture which is a prevailing characteristic.

E. C. HARDER (communication to the Secretary*).—I am very glad Mr. Singewald has brought up the question of the origin of the bedded manganese-ore deposits in the Miguel-Burnier region. My descriptions of these deposits have been so brief that I did not feel justified in saying much about their origin. It has been mentioned, however, that one main manganese-ore bed is found in the district and that possibly other horizons carry manganese ore. It has been stated also that limestone has not been found at the main manganese horizons but that it occurs elsewhere in the sedimentary series. Many persons who have seen the deposits, among them O. A. Derby and H. K. Scott, are of the opinion that they were formed by the weathering and replacement of beds of carbonate rock. The view, as Mr. Singewald says, is based mainly on the fact that the ore occurs in the same sedimentary series with lenses and beds of manganese-bearing carbonate rock (or limestone) and that it is prevailingly soft. Mr. Chamberlin and I have come to a somewhat different conclusion for a number of reasons.

1. Carbonate rock or limestone is found outcropping at the surface in the immediate neighborhood of the places where the manganese-ore bed crops out. The manganese-ore bed on the other hand has been mined by underground methods to a considerable depth and no carbonate rock has been encountered. There is no apparent reason why weathering and replacement should have occurred to such a depth along this particular bed while other carbonate rock beds in the vicinity were almost untouched. If it is a replacement deposit, might not one expect to find carbonate rock within a reasonable distance of the surface somewhere along the extent of several miles that the bed has been developed?

2. The ore comprising the manganese bed, although soft, does not have the consistency one would expect a replacement deposit of limestone or an oxidized decomposed manganese carbonate bed to have. In replacement deposits the manganese ore usually shows geodal, nodular, and concretionary forms. Some residual impurities from the limestone, such as clay or silica, are generally present. Manganese ore derived by the alteration of manganese carbonate is commonly earthy and porous. Material of this type is of common occurrence throughout the manganese-bearing areas near Miguel Burnier and Ouro Preto, resulting from the surface decomposition of carbonate rock layers associated with the sedimentary series. Mr. Singewald mentions these occurrences as tending to show that manganese ore occurs at several horizons. The ore in the manganese bed is generally bluish black and finely crystalline and is more or less compact and uniform. It consists of very pure manganese oxide. The bed is not hard and does not crop out prominently as many of the iron-ore beds do because the manganese minerals composing it are by nature soft. This is true of sedimentary manganese-ore beds in most

* Received Mar. 7, 1917.

parts of the world. Many of the iron-ore beds of Minas Geraes, on the other hand, are composed of hard hematite which resists weathering. However, even in the case of iron ore, it is well known that by far the largest and most numerous deposits of this region consist of soft "jocutinga" ore (soft crystalline hematite) which does not crop out.

3. Most of the so-called limestone beds of this region are not limestone but consist of a mixture of carbonates of calcium, magnesium, iron, and manganese. If such a bed were replaced by manganese oxide the calcium and magnesium would be carried away but the iron would tend to remain behind as hydrated oxide and would form an appreciable proportion of the manganese bed, which is not the case in the Miguel Burnier district. The manganese oxide may have replaced a bed of pure limestone, but that is merely speculative. Limestones as a rule are not so pure that they do not leave residual material such as clay and iron oxide behind.

4. The manganese-ore bed occurs regularly interlayered with the other sedimentary rocks and behaves like them in every respect. On one side is schist and on the other side is a bed of itabirite. Between the itabirite and the manganese-ore bed and directly in contact with the latter there is in many places a layer of soft crystalline hematite which is believed to be an original sediment. This close association would tend to indicate that the manganese-ore bed also is an original sediment.

Geology and Ore Deposits of Mohave County, Arizona

Discussion of the paper of FRANK C. SCHRADER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1935 to 1967.

JOHN CARTER ANDERSON, Kingman, Ariz. (communication to the Secretary*).—From an extended study of the Oatman and Secret Pass districts of Mohave County, Arizona, I believe that the genesis of the ore deposits, in most cases, is directly connected with rhyolitic intrusions later than the Tertiary flows; and that the developments of the future will prove these intrusives to be the primary source of the ore.

In support of this opinion I will instance the Wilhelm-Eclipse dike in Secret Pass and the Murdock, Nellie, Black Range dike and ledge in the southern part of the Oatman district.

The Wilhelm-Eclipse dike is a very fine-grained light-colored siliceous rhyolite dike with a strike of N.45W. traversing the full length of the Secret Pass district about midway between two major faults which mark the easterly and westerly boundaries of the district. These faults have lifted the block between so that erosion has exposed the pre-Cam-

* Received, Feb. 14, 1917.

brian granitic complex which underlies the range. The geological conditions present on surface are, therefore, analogous to what would probably be the 2,000-ft. level of the Tom Reed mine in Oatman, or several hundred feet deeper than the deepest opening in that mine. From the Wilhelm group of claims in the southeast end of the district, where the granitic complex is best exposed, to the Nancy Lee mine at the other end of the district, the surface rises several hundred feet. About one-third of the way from the Wilhelm to the Nancy Lee the granite is overlain by the green chloritic andesite. The dike is opened by several shafts from 40 to 200 ft. deep, the deepest being on the Eclipse claim just below the contact between the andesite and the granite. In every opening ore has been exposed. The gold is all fairly coarse as compared with the Oatman gold and some seams and streaks of high-grade have been cut which show wire gold up to $\frac{1}{2}$ in. in length.

The high-grade ore is usually found in the rhyolite along the foot-wall contact. In the high-grade the gold occurs principally along seams and fracture planes in the rock, or in recementing quartz veinlets showing visible free gold. A cross-cut from a 50-ft. shaft on the Wilhelm cuts 45 ft. of rhyolite which is very much shattered and open, with a great deal of manganese dioxide in shrinkage cracks and fracture planes, and carries from a trace to several dollars in gold. The rock is heavily impregnated with hematite, which is clearly pseudomorphic after pyrite; and carries the best values where least altered and the poorest where the manganese is most abundant. At the Nancy Lee, where the thickness of the green chloritic andesite above the granitic complex is several hundred feet, the dike cuts a quartz and calcite cross-vein similar to those of the Oatman district, some good values resulting in the quartz vein near the intersection.

Similar conditions are being disclosed by the development of the property of the Murdock Mining & Milling Co. in the Black Range section of the Oatman district. Through the kindness of Mr. Porter, the General Manager, I have been able to learn of the results of their underground work.

The Murdock property is situated just southwest of the Boundary Cone and covers the extension of a mineralized rhyolite dike. Eastward from the end line of the Murdock the dike is largely covered by wash in its course through the property of the Oatman Syndicate. Still farther eastward its place is taken on the property of the Nellie by one of the typical quartz and calcite veins of the district.

The country rock at the Murdock is the older, or basal andesite, which directly overlies the pre-Cambrian granitic complex. An exposure of the granitic complex is to be seen just south of the dike. A cross-cut on the 300-ft. level from a shaft sunk in the hanging-wall andesite, after passing through 82 ft. of rhyolite and a 12-ft. contact ledge on the foot

wall, cuts the granite. At the Nellie the country rock on the foot-wall side of the vein is the basal andesite, while the shaft sunk in the hanging wall was in the next higher flow of trachyte for 250 ft. There is a cross-fault between the Nellie and the Murdock and the block in which the Nellie is located is relatively downthrown with reference to the Murdock.

The rhyolite of the dike on the 300-ft. level of the Murdock is heavily impregnated with hematite and assays low in gold for the full width. The character of the rock, the mineralization, and the gold content are exactly similar to that in the Wilhelm-Eclipse dike coming from the same geological horizon. In both instances the gold is free, relatively coarse as compared with the gold in the producing mines, and is contained in the heart of the hematite.

At the Nellie, the vein filling on surface and where cut on the 350-ft. level is largely calcite and a pseudomorphic replacement by quartz and adularia; but one specimen from the 350-ft. level, said to be from the highest-grade streak cut, was quartz and rhyolite impregnated with hematite and identical in character with the rhyolite of the Murdock.

As the dip and strike of the dike on the Murdock and the vein on the Nellie are practically identical and the connection can be traced on surface, I believe that a rhyolite intrusion, exposed by the deeper erosion on the Murdock, came up along a preëxisting fissure already filled with a calcite vein filling and that the solutions emanating from the dike were the agencies by which the calcite was replaced wholly or in part by quartz and adularia and the vein filling impregnated with auriferous pyrite.

This connection between the rhyolite dikes and the quartz and calcite ledges of the district cannot elsewhere be traced so conclusively, as few rhyolite dikes have been exposed by the erosion in the areas covered by the Tertiary flows. In the eroded areas, as at Secret Pass, however, the intrusive dikes are as frequent as veins in Oatman, and I believe that deeper development in Oatman will result in discovery that the ore-bearing quartz and calcite-filled fissures connect in depth with rhyolite dikes having a primary auriferous pyrite in the body of the dike; and possibly a richer contact vein between the dike and the granite.

In drifting along the contact vein exposed on the 300-ft. level of the Murdock, according to Mr. Porter, alternating small shoots of rich and lean ore have been cut; the face of the drift for several feet of advance at times averaging as much as \$50. Specimens of this ore appear to be composed largely of brecciated and recemented rhyolite, similar to the high-grade ore from the Wilhelm-Eclipse dike, and show free gold in hematite and in the recementing quartz veinlets. It is probable that at points of greatest shattering, as at intersections and crossings, this recementing will be sufficient in extent to form good-sized shoots of high-grade ore.

Further evidence in favor of the rhyolite as the source of the primary

ore is found in certain silicified flows which carry low values wherever sampled. The average of a number of samples recently broken by the writer at random from a silicified surface flow of rhyolite over four claims was 82c. per ton.

What the tenor of the primary ore will be, it is impossible to say, as the oxidation of the pyrite, and possibly the impoverishment of the primary ore, extends below the deepest workings in the district. There are undoubted evidences of leaching and reënrichment in the higher horizons of the veins, probably as a result of the percolation of surface waters later than the original deep-seated oxidation of the pyrite. A similar leaching of a portion of the gold in the primary ore may have accompanied the oxidation of the pyrite to depths as yet unknown. The nearest approach to an approximation of the value of the primary ore is found in the values cut in the body of the dike at the Wilhelm on the 50-ft. level and the Murdock on the 300-ft. level, and I believe that the values which will be found below the present water level of the Wilhelm will be higher than those in the cross-cut on the 50-ft. level.

With reference to the pyritic andesite cut on the 700-ft. level of the Gold Road mine indicating the near approach of the sulphide zone, it is my observation that this pyritization of the green chloritic andesite is characteristic of the wall rock, usually on the hanging wall, in a number of instances, and the unoxidized pyrite extends practically to surface, where that formation is exposed on surface. Among the places where I have noticed this occurrence of pyrite in the wall rock is the cross-cut into the hanging wall on the 200-ft. level of the Black Eagle mine of the Tom Reed, in the hanging wall of the 300-ft. level of the main Tom Reed mine, in the cross-cuts on the 300-ft. level of the Oatman Amalgamated and in two places in Secret Pass. The oxidation that has taken place in the veins to an unknown depth seems to have had little effect on the pyrite in the wall rock. This is probably due to the fact that the water of the district is almost wholly confined within the walls of the vein itself. In only one instance that I know of does the pyritic wall rock carry gold minerals. That is at the Orphan group of claims, now the Secret Pass Gold Top Mining Co., in Secret Pass. Here the ore from a winze which enters the pyritic andesite at about 50 ft. carries \$8 in gold.

This is the property spoken of by Mr. Payne. The mineralizing solutions here have come up along a major fault which marks the westerly boundary of the Secret Pass district, spreading out irregularly into the wall rock. Several thousand dollars worth of high-grade ore from a shallow glory hole were milled in a two-stamp Tetrault mill by leasers, and a considerable tonnage of surface ore running from \$8 to \$15 per ton in gold was opened up. This fault is later than the Tertiary flows and the rhyolite intrusions which it cuts. It is easily traceable for several miles and just east of the Orphan mine has lifted the pre-Cambrian

granitic complex to a contact with an andesite later than the green chloritic andesite, which in Secret Pass immediately overlies the granite. The granite on the foot-wall side is irregularly mineralized, and in several places high-grade gold ore has been found in prospect holes.

One other ore occurrence in Secret Pass is worthy of particular notice. A rhyolitic dike of a somewhat coarser texture than the Wilhelm-Eclipse dike and having a strike of N. 35 W. which is paralleled by the major fault marking the easterly boundary of the mineralization of the district, has mineralized the granite for from 25 to 35 ft. on each side of the dike. The granite is stained a deep red by iron oxides and a sample across 23 ft. on the foot-wall side of the dike at a depth of 12 ft. assayed \$4 in gold. No deeper development has yet been undertaken on this ledge.

A Study of the Silica Refractories

Discussion of the paper of J. SPOTTS McDOWELL, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1999 to 2055.

J. W. RICHARDS, So. Bethlehem, Pa.—The paper is certainly an exceedingly valuable one and it gives detailed information which should be utilized in the following way. There are probably twenty different circumstances or conditions under which silica bricks are used. Sometimes they are used under high pressure, at others under low pressure, at times for low temperatures and others for very high temperatures. If the methods of this paper are followed out, and if the precise mechanical and thermal properties are determined and can be reproduced, the industry can furnish the proper brick for any particular purpose. There is at present only an imperfect classification made as to properties and uses, but we need a much more scientific classification of the uses to which the bricks are to be put and then should manufacture the bricks suited particularly for each of those purposes. There is one property which is not mentioned in the paper and which is of considerable importance, that is, the electrical conductivity. There may be other properties besides, which a good research laboratory can determine, and when we have listed all the various properties and noted the way in which they can be reproduced, the maker can then specify his exact requirements and the corresponding quality of brick can be selected or made to meet them.

H. D. HIBBARD, Plainfield, N. J.—About a year ago I had some experience in another country, in melting steel, where silica bricks made in that country failed, but in a different way from those in America. Here, when the bricks get too hot, they are partly melted and string down

from the roof, but there when the heat was too high pieces dropped off from them, and I would like to ask why they acted that way.

W. F. ROCHOW, Pittsburgh, Pa.—An underburned silica brick if burned rapidly will expand permanently to about the same degree as a properly burned silica brick, but would be weakened. It is possible that the difficulty you describe could have been caused by too rapidly burned brick.

H. D. HIBBARD.—It seemed to me that the trouble was that the lime was not thoroughly mixed with the ganister so that in the cementing material between the grains of ganister the lime was very high, perhaps 10 or 20 per cent. and the cementing material was therefore too easily fusible. Then when it got hot enough, it was softened, lost its strength and let the brick drop to pieces. Since then in thinking it over, it appeared probable that the trouble was in insufficient grinding and mixing of the materials in the pans. Thus the cementing material might, if it had a high enough percentage of lime, have a fusion point as low, perhaps, as $1,200^{\circ}\text{C}.$; if it got more silica worked in by continued grinding it might rise to 1,300, 1,400, 1,500 or perhaps $1,600^{\circ}$. The total or ultimate percentage of lime in the brick does not give the composition of the cementing material or the mortar between the silica grains on which the integrity of the brick really depends. It is like a chain, the strength is that of the weakest link, and the resistance of a silica brick to heat depends on the fusibility of the most fusible part, which, in this case, was the cementing material between the grains.

W. F. ROCHOW.—That is a possible explanation. There are three eutectics of mixtures of lime and silica, and a silica brick having a much higher percentage than 3 per cent. of lime will be lacking in refractoriness while one having less than, say, 1.4 per cent., will be mechanically weak.

H. D. HIBBARD.—The point I wanted settled, if possible, was whether the lime and silica which formed the mortar between the grains had the proper degree of refractoriness or not, and if not, whether it could have been cured by prolonged mixing and grinding?

W. F. ROCHOW.—Such a condition would be possible, but it is not at all likely with brick manufactured in this country in accordance with the present methods, because after the quartzite rock is ground and mixed with the lime as milk of lime, it is usually allowed to stand for some time before it is used, so that the milk of lime would have a tendency to spread uniformly throughout the mass even if the mixing could not have been thorough in the grinding pan.

H. D. HIBBARD.—Without any mixing effect?

W. F. ROCHOW.—No, not without any mixing. Thorough mixing

in the grinding pan, however, is readily accomplished without great difficulty.

J. B. UMPLEBY, Washington, D. C.—I have been particularly interested in this paper because I know something of the work of Dr. Fenner on the stability of the silica series in the Geophysical Laboratory in Washington, and although I am not sure, I suspect that in his original investigation, he did not foresee the application of the work to the manufacture of silica brick. It is probably one more instance of an abstract study having a definite commercial application unforeseen by the man who originally starts the investigation. I cannot but wonder if much more of the work being done at the Geophysical Laboratory on the two and three component systems will not have a definite application apart from adding to our knowledge of mineralogic phenomena.

The Conservation of Phosphate Rock in the United States

Discussion of the paper of W. C. PHALEN, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1901 to 1934.

E. G. SPILSBURY, New York, N. Y.—I would like to emphasize what Mr. Phalen says in his paper regarding the possible utilization of the waste acids in the West for the acid concentration of phosphate rock, and to call attention to the fact that this is already being done by Mr. Laist at Anaconda, who is producing concentrated phosphoric acid from these low-grade rocks. During the last year, he has not been able to carry this along to the extent he had expected, owing to the tremendous demand which they have had for all the acid they can produce, so instead of having a surplus of acid, as was expected, they are really short of this material for their electrolytic copper and zinc work.

The Magnetic Concentration of Low-Grade Iron Ores

Discussion of the paper of S. NORTON and S. LEFEVRE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 149 to 169.

GEORGE C. FOOTE, Port Henry, N. Y. (written discussion).—The paper by Mr. Norton and Mr. LeFevre will bear the most careful consideration by all interested in the iron business, particularly in the East. There is much information contained therein which should be emphasized.

The idea of handling iron-bearing rock in quantities comparable to the porphyry coppers does seem at first blush to be beyond the pale of commercial possibility. Yet, today in the Adirondack region 25 per cent.

iron-ore is being mined and milled in small quantities at a profit and an "even break" can be made at present prices with a 20 per cent. ore with present equipment. Large tonnages mined and milled with equipment designed to properly handle and treat this 20 per cent. ore would today show substantial returns to the producers.

Another latent point contained in this paper deserves particular consideration.

It is well known that the old available low phosphorus ores the world over have been growing scarcer each year and even with the normal demand it is only a question of a comparatively short time before furnaces will be facing a famine in this ore.

F. S. Witherbee, in his estimate referred to in the paper under discussion, gave 1,100,000 tons of magnetite ore above 30 per cent. iron content available in one section of the Adirondack region. It is entirely safe to say that of this amount one-half is ore which will average less than 0.020 in phosphorus.

Your attention is directed to the separation tests of the Jackson Hill and the Palmer Hill ores, and particularly to the reduction of the phosphorus in magnetite ores of this character, accomplished by magnetic separation. These tests, together with hundreds of others made by Witherbee, Sherman & Co., demonstrate that the phosphorus in the crude magnetite is reduced one-half by concentration; also that the iron content can easily be increased to 60 per cent. and a furnace product obtained of the same physical characteristics as the well-known old standard concentrates of this region. A crude ore, then, containing 20 to 30 per cent. iron and 0.020 phosphorus will make, by magnetic separation, a product of ideal size for blast furnaces and of 60 per cent. iron and 0.010 phosphorus content.

There are millions of tons of ore of this character in the Adirondack region, notably in the Ausable district of Clinton County, owned by Witherbee, Sherman & Co., and in the Lake Sanford district of Essex County owned by MacIntyre Iron Co., where quarrying and steam-shovel operations could be conducted and low operating costs insured.

Mining and milling costs with modern equipment and large tonnage should not average more than \$1 a ton crude ore: The cost of concentrates on board cars at the mines, not over 4 c. per unit; freight from mines to Eastern furnaces, not over 3 c. a unit, making a total cost of this special low phosphorus standard ore, free from copper, of not over 7 c. a unit at the furnace. The average selling price of this grade of ore at Eastern furnaces in normal times has been about 10 c. a unit. Present prices range from 15 to 20 c. a unit. The opinion seems to be that after the war a price of about 11 c. a unit will be maintained for a number of years.

When the enormous tonnage of ore available is considered; the

coarsely crystalline character of the ore in crude and concentrates; the cheap mining and milling costs; the freight cost to furnaces and the high iron content of the product, the possibilities of this region as a producer of various grades of iron ore begin to be realized.

JOHN L. W. BIRKINBINE, Philadelphia, Pa.—Much that I wish to mention in regard to the paper by Mr. Norton and Mr. LeFevre has already been taken up in the discussion of Mr. Foote, but there are a few points left that I would like to call to your attention. I think Mr. Norton and Mr. LeFevre did not mean to say that any magnetite iron ore can be concentrated for 10 or 12 c. a ton. They probably intended to state that coarse-grained ore, easily granulated, can be concentrated at this price.

In examining the flow sheets for mills 3, 4, and 5, your attention is called to the policy of subjecting the rock to the minimum amount of crushing. You will note that in mills 3 and 5 the rich ore was removed wherever possible without further reduction. This makes a good furnace material, while in mill 4 the barren material is rejected so as not to crush this to small size, as fine grinding not only causes great wear and tear on the machinery but also utilization of a large amount of power.

The authors' recommendation that before deciding on any method of concentrating iron ore, there should be a great deal of experimentation, should not be neglected, for at Mineville the whole system of mills has been an experiment on an enormous scale and the success of that plant has been due to the willingness of the men behind it to spend large sums of money for experimental purposes.

For the past year or so, I have been devoting a great deal of time to studying the lean magnetites that occur in the East, and although many of these deposits, especially those in eastern Pennsylvania, were formerly operated when the iron industry was in its infancy in the United States, they were afterward abandoned. In going over the deposits, we are surprised to find the great diversity in the characteristics of the ore.

In one section in Berks County, Pennsylvania, in a distance of 12 miles there are five different ranges of ore, each one distinct in its mode of occurrence and character; some being high in sulphur, some of Bessemer grade, some titaniferous, some rich in iron and high in titanium and sulphur, others lean in iron and carrying little of the detrimental substances. Of these the best known are the magnetites of the Cornwall type, which are universally high in sulphur, the gangue generally consisting of lime, but sometimes of shale. The ore of the Cornwall type occurs between the Mesozoic conglomerates and the Paleozoic limestones, near diabase intrusions, the other magnetites occurring in gneiss.

It is my hope that in the early future I may be able to present to the Institute more detailed information regarding these deposits.

FRANK L. NASON, West Haven, Conn. (communication to the Secretary*).—I shall have to depend on memory for most of the following statements, as I have not a copy of Mr. Witherbee's paper before me.

The total area of the magnetic iron ore field of the Adirondack Region is about 9,000 sq. miles. Exclusive of the gabbros, which cover about 2,500 sq. miles, inclusive of the Grenville, or white limestone series, the iron-ore gneisses cover 6,500 sq. miles. I am quite sure that Mr. Witherbee, in estimating the reserve tonnage, took into consideration only the areas in which great tonnages have been actually produced, leaving entirely out of consideration all other possible areas, including the gabbros exclusive of the Tahawus titaniferous ores. In Clinton County, the distance from Palmer Hill to Lyon Mountain is about 20 miles. The width of the field from east to west is about 15 miles. The total area is thus about 300 sq. miles.

In Essex County, the area is about 200 sq. miles; exclusive of the Gouverneur red hematite field, the Benson mine area is about 20 sq. miles; the total area covered by Mr. Witherbee's estimates is thus about 520 sq. miles. This is 8 per cent. of the total area of magnetic iron ore-bearing gneisses.

As to the tonnage per square mile, there seems to be something wrong with the writers' estimates. There are 43,560 sq. ft. in 1 acre. With a thickness of 10 ft. of iron ore, allowing 10 cu. ft. per ton, there are 43,560 tons per acre or 27,878,400 tons per square mile, instead of 2,700,000 tons as stated in the paper.

Taking one-tenth of the estimated area (not the total area) gives a productive area of 52 sq. miles. According to data assumed, this 52 sq. miles would yield 1,449,800 thousand tons, practically Mr. Witherbee's estimate. This is exclusive of the 70 to 100 million tons of titaniferous iron ores in the Tahawus field. In New Jersey, the authors' estimate of iron ore area is 900 sq. miles. They are not as conservative in their estimates as is Mr. Witherbee, for they assume that the entire belt of iron-ore gneisses is underlaid with minable ore. This gives them the total of 2,300,000,000 tons. If they had followed Mr. Witherbee's excellent example, one-tenth of 8 per cent., the productive area would be 7.2 sq. miles with a total reserve of 19,440,000 tons as against their estimate of 2,300,000,000 tons. Still farther, the area of iron-bearing gneisses in southeastern New York, including the Tilly Foster and Theal mines areas, is about 400 sq. miles. As above, this would reduce to a productive area of 3.2 sq. miles, adding about 7,640,000 tons to the New Jersey reserves, making a grand total of 27,080,000 tons in the southern magnetic iron ore field.

It seems very clear to me that Messrs. Norton and LeFevre have not

* Received Feb. 20, 1917.

thoroughly studied Mr. Whitherbee's paper. At least they have held him to "strict accountability" in his own field and have applied his methods only nominally in dealing with the New Jersey field. In dealing with this field (the New Jersey) they seem to me to have ignored some very important data. To point these out would take more time than is allowable in discussion. They may be made the subject of a separate paper later on. Personally, however, while engaged on the geological survey of New Jersey and private work later, my estimate of reserves in this State and in southeastern New York, which I have considered liberal, amount, in round numbers, to about 50 per cent. of the total given in the paper under discussion.

An Investigation on Rock Crushing Made at McGill University

Discussion of the paper of JOHN W. BELL, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 171 to 176.

R. B. T. KILIANI, New York, N. Y.—I wish to present some figures based upon actual observation extending over a few months time which seem to prove the author's conclusions. It is a comparison of two machines operating on the same ore, in the same plant, one taking the undersize of $\frac{1}{2}$ -in. screen and the other taking the undersize of 8-mesh. Each of the machines was first operated as a pebble mill, with flint pebbles, in open circuit, then with metal pebbles in open circuit and, finally with metal pebbles in closed circuit. For the first case, the percentage of oversize on 48-mesh, Tyler screen, in the discharge, varied, in the mill taking $\frac{1}{2}$ -in. feed, from 37.2 to 19.9 per cent. The second mill, taking the 8-mesh feed, had a variation in oversize on 48-mesh, from 36.8 per cent. to 0.5 per cent.; so it is seen there is considerable variation in these figures. I have computed the tons per kilowatt-day reduced through 48-mesh for each machine under each one of these conditions, giving 6 sets of figures. The first of these I called 1.0 and determined the ratio of the others to this. In the same way, I figured the relative mechanical efficiency according to Kick's law, using the figures given by Mr. Taggart in his paper, for ordinal numbers of the Tyler screens. I have also figured the efficiency in mesh-tons per horsepower according to Gates' method by plotting the screen analyses of feed and product and multiplying the area by a constant dependent on the scale used. The commercial efficiency of tons per kilowatt-day reduced through 48-mesh checks very closely with the efficiency by Rittinger's law, within 1 per cent. except in one case. By Kick's law, the efficiency checks fairly closely, within about 5 per cent., with the commercial efficiency of the mill grinding $\frac{1}{2}$ -in. feed, but when we start to grind a finer, 8-mesh material, and especially grinding very fine, with a

greater percentage of -200 mesh material produced, there is a big discrepancy. The ratios obtained may be tabulated as follows:

Relative Efficiencies

	Open Circuit		Open Circuit		Closed Circuit	
	Flint Pebbles		Metal Pebbles		Metal Pebbles	
Feed size.....	- ½ in.	-8-mesh	- ½ in.	-8-mesh	-½ in.	-8-mesh
Commercial.....	1.00	0.81	1.07	0.76	1.26	0.79
Rittinger.....	1.00	0.67	1.06	0.77	1.22	0.79
Kick.....	1.00	0.42	1.03	0.44	1.13	0.45

J. W. BELL (communication to the Secretary*).—Mr. Kiliani's results indicate very interestingly the inherent defect in Kick's law demonstrated by our own experiments that the Kick or Stadler efficiency figure is influenced far more by the size of the feed to a crushing machine than by its actual performance. Mr. Kiliani, of course, appreciates that the number of tons reduced to -48-mesh is a very rough measure of the amount of crushing performed since obviously a large variation in the percentage of the finer sizes would be possible without conflicting with this specification. He has used this method merely for the sake of the comparison of the several results.

I mention this only because if the conclusions based on experiments performed at Purdue University and McGill University are correct, a method for measuring the efficiency of rock-crushing machines is at hand, which is at once simple, dependable and practical.

C. W. MERRILL, San Francisco, Cal. (communication to the Secretary†).—I have read with the greatest interest Mr. Bell's paper and cannot compliment it too highly for the scientific method by which he has arrived at his deductions. It is my opinion that the paper will be a classic for some time to come, and I can only hope that it will stimulate further pursuit of the subject in detail in the field of fine crushing. Particularly would it be of value to the profession if, for instance, the ordinary tube mill, the Marathon mill and the Hardinge mill could be compared by the methods used by Mr. Bell.

* Received Mar. 6, 1917.

† Received Feb. 20, 1917.

Countercurrent Decantation

Discussion of the paper of LUTHER B. EAMES, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2087 to 2101.

J. V. N. DORR, New York, N. Y.—I have read Mr. Eames' excellent paper on this subject with great interest, for besides being connected with the design and installation of the first modern countercurrent decantation plant, he was associated with me in the development of the thickener in the Black Hills some years earlier, and has now been operating the largest plant that has used this method of treatment. His discussion of the influence of different factors on the results obtained is quite thorough, but several features I think may not be apparent to the reader who has not studied the subject carefully.

1. *Grade of Ore That Can Be Treated.*—Mr. Eames has shown graphically that the assay of all solutions increases directly as the grade of the ore treated. He has not emphasized, however, the fact that, as most operators prefer to keep the assay of pregnant solutions down below a certain figure in order to insure complete precipitation, an increase in ore value beyond a certain point means a proportional increase in solution precipitated and therefore not only an increase in the percentage of dissolved gold recovered, but an actual decrease in tailing loss.

The importance of the influence of the percentage of moisture in the underflow on loss is made clear, and I may add that unless it is quite evident that pulp can be safely settled to 50 per cent. solids, or more, it is usually advisable to use a continuous filter at the end of a decantation plant to reduce losses in cyanide and gold.

Dissolution in Thickeners.—The influence of a change of solution on silver dissolution has long been emphasized and most gold operators have recognized the annoying way in which a gold ore after being agitated until nothing more can be dissolved will loosen up again when almost ready for the dump after the dissolved metal has been removed by decantation or filtration. Continuous decantation gives favorable conditions for this additional dissolution and a chance to recover most of what is thus dissolved. This is especially true if an extra tank is used in the series and barren solution added at the second tank from the end instead of the first, as shown in Mr. Eames' Fig. 1.

It can be calculated that the use of this tank will reduce the mechanical loss of cyanide by one-third and cut the gold loss as well; but the saving in gold or silver that may dissolve when the pulp is first diluted with weak solution may be as important.

At one plant in the Southwest, where one of the agitators was not in use as additional dissolution would not pay the cost of its operation, the

company found that 30 c. more was being dissolved in the decantation thickeners.

It has been pointed out that decantation alone should only be applied to suitable ores, but it is becoming well recognized that when filters are used one or more steps of countercurrent decantation should precede them, as the additional dissolution will more than pay operating expenses; also, the lower-grade solution going to the filter means lower filter costs and tailing losses.

The added cost of operating an additional thickener on the average ore is so low as to be difficult to estimate and may safely be placed at less than 1 c. per ton, including reasonable amortization.

Trays.—Mr. Eames has referred to the use of trays in decantation plants. I may say that one self-supporting tray giving double capacity has been in operation for some time in a 50-ft. tank and one for a 75-ft. tank has been designed.

Tanks with several trays are also in operation, so that the problem, especially apparent in cold climates, of large mill buildings for a decantation plant, is being rapidly solved.

The Function of Alumina in Slags

Discussion of the paper of CARL HENRICH, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 2081 to 2086.

A. S. DWIGHT, New York, N. Y.—Mr. Henrich's paper is rather inconclusive as to the role that alumina really plays in slags. He gives a number of interesting instances of high alumina, with conclusions that he draws from the analyses, in the light of what he knows of the accompanying conditions, but the information given is hardly sufficient to permit intelligent criticism of these conclusions, for it is well known that the slag composition is not the only factor that can produce some of the furnace results favorable or disastrous, as the case may be, which he records. As a matter of fact, most metallurgists who have wrestled with this problem in connection with the smelting of lead and copper ores, usually arrived at some sort of a working hypothesis which met the local conditions fairly well, and then, being busy men, were inclined to leave the theoretical question very much as Mr. Henrich leaves it when he says: "A question with which I purpose dealing at some future time."

It is unfortunate that in the Golden Age of the smelting industry in the West, when we were all dealing with large tonnages of complex ores, often containing large percentages of alumina in combination with other refractory elements, the smelting companies were not generally equipped for research as they are today, and much experience that might have been

most instructive, and perhaps have led to a solution of the problem, was lost to us.

A problem of this kind must be attacked from the empirical side, and on a full working scale. That is to say, we must find a slag which works well in the furnace, flows hot and clear, melts rapidly, reduces well, and keeps the furnace open and free. Having once produced the slag that fulfills these ideal conditions, we should try and find out why it is good, and in this we can gain much help from laboratory appliances and theoretical reasoning, determining the analysis, specific gravity, melting point, specific heat, viscosity, oxygen ratios, etc. Through a knowledge of these factors we may learn how to reach the same result another time. But I maintain that if we start our reasoning with oxygen ratios, formation or melting temperatures, or even slag analyses disassociated from experience, we can never reach safe conclusions, to be applied to the solution of the practical problem.

Most metallurgists have, as Mr. Henrich says, tried figuring alumina on both the acid and basic side of the equation, and that has also been my own experience. While always decidedly doubtful of the possibility of aluminates being formed at the temperatures prevailing in the lead-smelting furnace, I nevertheless obtained better results with my slags when I figured Al_2O_3 as an acid, *i.e.*, replacing SiO_2 , thus corroborating to some extent Mr. Henrich's experience, than when I treated it as a base; but I finally adopted with most satisfactory results the theory proposed by the late T. S. Austin, then Superintendent of the El Paso Smelting Works, and with whom I enjoyed a close business association and professional acquaintance. Austin's theory was that in lead slags Al_2O_3 played a neutral role and might be considered as remaining dissolved as Al_2O_3 uncombined in the slag. I am familiar with the experiences which led him to adopt this working theory, which were briefly as follows: The El Paso plant was called on to treat large tonnages of ore from the Parral district in Mexico, containing considerable percentages of fluorspar (calcium fluoride). The slags were figured as usual, assuming that the lime of the fluorspar went into the slag as CaO , but the slag always came down off type, and showing all the physical and metallurgical characteristics of a slag deficient in lime, though by analysis it was apparently correct. As previously stated, the actual behavior of the slag itself is the only true guide to the experienced metallurgist, and in this case the addition of lime to the charge would invariably bring it around to its proper characteristics, and behavior. The amount of correction needed was equivalent to the lime present as fluorspar. Finally, by neglecting the lime present as fluorspar, Austin got his correct slag on first calculation, so he reasoned that fluorspar was not decomposed in the smelting but simply dissolved in the mass of slag as such. Some time after that he happened to encounter an ore high in Al_2O_3 ,

and obtained excellent results by applying the same line of reasoning, *i.e.*, considering it as an inert diluent. Subsequently ZnO was found to behave very much the same way, at least satisfactory results were obtained with very zinky ores when this working theory was followed. My own experience, extending now over a good many years, has corroborated the practical value of Austin's theory as applied to these three elements. But like Mr. Henrich, "I propose to deal with this at some future time," unless some one is more forehanded and gives us the true answer to the riddle.

E. P. MATHEWSON, Toronto, Ont.—I want to add a few words to what my friend, Mr. Dwight, has said, and to confirm his findings. It is some time since I had to do with very heavy alumina in slags, but when I did have to do with it, I had a great deal of trouble at the start, due to the fact that we were dyed-in-the-wool on certain types of slags and had to follow those types. A certain idea struck us some years ago in Colorado when alumina was forced on us from Cripple Creek, that we would not figure alumina as a base, would not figure alumina as an acid, and see what happened. The effect was magical. The slags, calculated as if no alumina existed in them, ran freely, and the furnaces went faster than they ever did on the old charges when alumina was not present in any appreciable quantity. We followed this up when we were dismantling furnaces and had enormous quantities of brick bats to dispose of, we got rid of these without any trouble and had the fastest running charges imaginable by disregarding the alumina entirely and calculating the charges as if no alumina existed. The same thing happened in copper smelting later, in Montana, although we had not quite so high alumina; by disregarding the alumina and treating it as an inert body, we had magnificent results.

WM. B. BOGGS, Douglaston, N. Y.—I notice in all these slags that the sum of the lime and magnesia runs up anywhere from 2 to 15 or 18 per cent. I have been forced to handle a lot of basic and siliceous ores from South America and Canada, carrying large amounts of aluminum. We make a very peculiar slag in our furnaces; that is, it is not normal, it runs from 32 to 34 per cent. silica, no lime at all, 8 to 10 per cent. alumina; and the balance is iron oxide, a very small amount of zinc oxide and magnesia oxide. We received a shipment of ore recently that ran 16 to 20 per cent. alumina, 8 to 10 per cent. silica, 4 to 5 per cent. copper and was high in iron and sulphur. Because of the shortage of sulphur we were forced to use a large amount of this ore, even to keep our grade of matte down to 55 per cent., and we found that it was all right to neglect the calculation of alumina in the slag composition, as long as the alumina was below 10 per cent. When the alumina in the slag went over 10 per cent., the furnace slowed down and dropped to about 2 tons per square

foot of hearth area. By constant analysis we found that our silica had become 26 per cent. in the slag. When we substituted for this sulphide, a clean sulphide that did not contain alumina, the furnaces picked up, that is, we could not force the furnace to run with a slag over 10 per cent. alumina. If we were running lime, it would act differently; if we substituted lime for some of the basic ores high in alumina the furnace made a slag higher in silica and this diluted the slag so that the alumina came down; with our blast furnace making a matte of about 55 per cent. copper, we found that we could not get a tonnage of over $3\frac{1}{2}$ to 4 tons per square foot of hearth with a slag containing more than 10 per cent. alumina.

JOSEPH W. RICHARDS, So. Bethlehem, Pa.—The title of this paper is too broad; it should be "The Function of Alumina in Lime-Iron Slags." The remarks of the author do not apply to the alumina in many other slags in which it is a large constituent. In iron blast-furnace slags, we have to take alumina into account; it is too large a constituent to neglect. In the siliceous slags of charcoal iron furnaces, the alumina undoubtedly acts like a base; but in the ordinary basic slag of coke iron furnaces, the alumina is best taken care of and understood if it is considered as an acid. This is the position of alumina in iron blast-furnace slag, and the qualifications which the author makes, and his remarks concerning alumina, do not apply to alumina in this important class of slags.

A. S. DWIGHT.—Following Prof. Richards' remarks, I will add to what I have already said that it is my belief, based on long observation, but without special investigation by pyrometric tests or otherwise, that at the maximum temperatures produced in the blast furnace smelting lead ores, neither aluminates nor silicate of zinc are formed. On the other hand, the temperatures in the iron blast furnace are much higher, and we must not fail to recognize the probability that entirely different combinations of the elements may take place. The lead and copper slags must be studied for themselves.

W. C. SMITH, Chrome, N. J.—Some few years ago I had charge of a small lead refinery, treating refinery byproducts. The principal material we had to get rid of was a mixture of magnesite brick, clay brick and silica brick not properly bedded. The furnace had been running along all right with about 10 per cent. of alumina in the slag. The night man called me up about 3 o'clock one morning, and said that the furnace was slagging slowly, and that the air did not go through properly, slag cold. By morning the furnace had picked up and was getting through more tonnage than we had gotten through in a number of weeks. We thought the troubles were over. The regular morning slag was sent to the laboratory, when the laboratory reported 26 per cent. of alumina. Thinking

the chemist had made a mistake, I sent it to him again, and the analysis came back with 27 per cent. alumina. The furnace continued to make this slag for 3 days, we had evidently cleaned up the material high in Al_2O_3 , and in going back to our old slag the furnace froze. I never knew why the immediate range from the high-alumina slag going down to a normal of about 10 per cent. would freeze the furnace. The SiO_2 , FeO (CaO and MgO) ratio was about the same in our slags with 10 per cent. Al_2O_3 and with 26 per cent. Al_2O_3 . If Al_2O_3 is inert and simply dissolved in the slags, why did the slags carrying between 10 and 26 per cent. Al_2O_3 give trouble?

J. W. RICHARDS.—It may not be generally known that some iron blast furnaces have been run regularly for a considerable time with 30 to 35 per cent. alumina in the slag, properly running slag which also excluded sulphur quite satisfactorily from the pig iron.

The Viscosity of Blast-Furnace Slag

Discussion of the paper of A. L. FIELD, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 307 to 332.

WOOLSEY MCA. JOHNSON, Hartford, Conn. (written discussion).—When we regard the number of British thermal units running into the billions that must be applied to metallurgical slags in the United States, we cannot but conclude that the physical chemistry of slags is a very important subject when measured in dollars and cents. A further contemplation of the subject shows us that it has been hitherto treated in too purely a practical manner. It is needless to comment at length on the cost of these heat units and on the cost of fluxes to produce these metallurgical slags, for it amounts to tens of millions of dollars annually. Accordingly, this research is of value to the country.

But we must all balance our results against our efforts. Therefore, while the speaker is unwilling to criticise unduly this meritorious research, he does criticise it in the interest of the profession for the reason that it seems to him too highly theoretical. The article is much too scientific to produce results at a minimum cost.

Let us explain our meaning. It has been said that there are always two sides to a question. It may be further said that there are always three sides to a question, the two extremes and the middle ground. In any industrial research there are the highly theoretical side, the purely practical side, and the theoretico-practical side. To study a slag and be practical in the studying, there is something needed that represents in the laboratory (or in microcosm) what actually happens macrocosmically in the blast furnace. There is such a thing as the art of metallurgical research occupying the middle ground between theory and practice.

The only direct connection to the operations of slag in the blast furnace in Mr. Feild's research that the speaker can see, after a diligent study, is that to the old practice of pulling out slag in strings with a hook.

Scientifically, the method employed is open to the criticism that the coefficient of friction between graphite and slag is a variant. Furthermore, the curves are not drawn with due regard to the theory of probabilities. It might be suggested that the influence of fluorspar on the fusibility of slags would be a fertile field for investigation.

The speaker has made many determinations of a similar nature, and along lines that he hopes were "theoretico-practical," in the period 1904-1914. He originated for this purpose two forms of apparatus. In one, the temperature at which a slag runs through a standard graphite aperture was measured (this temperature being called "the running-point" of the slag); in the other, an apparatus of Acheson graphite resembling an hour-glass was used. In this, the time taken in running a given amount of any slag at a given temperature could be measured. For a detailed account of these, the members are referred to *Electrochemical Industry* (July, 1906), 4, 262, and to *Trans.*, (1913), 47, 220. Prof. Guess has used a similar apparatus, in measuring the running temperature of mattes (*Bulletin* No. 114 (June, 1916), 1077).

By these methods, 20 determinations can be made at possibly one-third the expense and without any skill as compared with Mr. Feild's method. What is equally important, the results could be translated into practical language at once and one determination gives the practical temperature needed.

GEORGE K. BURGESS, Washington, D. C. (written discussion).—Mr. Feild is to be congratulated for his successful application of physical measurements to the important metallurgical problem of the viscosity of blast-furnace slags over considerable ranges of temperature and his paper is an excellent illustration of what may be accomplished when the methods of physics are applied to metallurgy.

Among the questions that may be raised is that of the constancy or permanence at a given temperature of the viscosity value for what appears to be a definite slag as shown by analysis. In other words, does a slag of a given chemical analysis possess the same physical properties if held for a considerable time at a temperature at which it is fluid? Mr. Feild gives some figures, but does not appear to dwell on this point.

It would be interesting also to make determinations of density of the fluid slag, for which Mr. Feild's apparatus may be readily adapted.

Although it is a minor matter, I should like to see a little more definite check as to the correctness of the temperature determinations, for his arrangement of apparatus, by the optical pyrometer sighting upon the

slag surface. The known freezing point of a definite chemical substance such as diopside or synthetic anorthite, or better perhaps of a slag itself, could have been observed, or a direct comparison made with the indications of a thermocouple immersed in the slag.

There is one item in this paper, however, to which I would take serious exception; that refers to the interpolated observations with Dr. Clement on the temperatures of slag at the blast furnace. In Table 2 there is here shown an apparent and considerable discrepancy between the values as given by the optical and thermo-electric pyrometers, and it is clearly to be inferred that the authors do not place reliance on their optical determinations. If, however, as explained in my paper on Temperature Measurements in Bessemer and Open-Hearth Practice, they had made the necessary corrections (which they do not say was done) to the readings of the optical pyrometer, the discrepancy noted would be practically eliminated. To the optically determined temperatures of Table 2 should be added some 60° C. for the emissivity of the slag (the most probable value of this emissivity is 0.65). The average temperatures are then: optical = $1,540^{\circ}$ C., thermo-electric = $1,522^{\circ}$ C. One would expect the latter to be somewhat low.

Furthermore, since it was found possible to immerse in the slag stream a closed-end graphite tube, it would have been easy to obtain true temperatures optically by sighting the pyrometer into this tube; incidentally this observation combined with the surface observation would have given the emissivity of the slag, a characteristic the limits of the numerical range of which it is highly desirable to determine for as many slags as possible.

R. H. RICHARDS, Boston, Mass.—One point occurs to me in regard to the details; I am not able to judge whether there is anything in it or not. Years ago I was asked by Gen: Paine to try different surfaces for the yacht Volunteer, when the international race was to take place. He wanted to get the one that gave the least friction. We used this very same apparatus to put our various substances upon to test the question of friction, and of all the substances we tried, graphite gave the least friction, it was the most perfect lubricant. Now the question I would ask is, whether graphite, which, in the cold, gave the most perfect lubricant, is really the best substance to use in viscosity tests? Do you want a substance that gives the greatest friction, in order to get this viscosity measure? The second question is, did the graphite stay as graphite when the test was being made?

J. W. RICHARDS, So. Bethlehem, Pa.—I would rather have seen the title of this paper "The Fluidity of Slags;" that is the thing the blast-furnace man is usually interested in most. I wish to call particular attention to the fact that, on page 324, the slag number 22,954, the first

one in the list, which is the most viscous at the low temperature and the most fluid at the high temperature, is the highest in alumina of the whole set. It contains 35 per cent. of alumina, and yet at 1,500° it is the most fluid of all the light slags tested. This is quite in line with the fact that if you reduce the silica so as to make room for the alumina, you can get high alumina slags of quite satisfactory fluidity for iron blast-furnace purposes. For low-melting, iron-lime slags, in lead or copper smelting, alumina is of course objectionable above a small amount, but this limitation does not apply to iron blast-furnace slags, which are essentially lime silico-aluminates. I wish to make the following general remarks concerning slags: In general, slags are not of definite chemical composition, it is a mistake to assign them definite formulas. A slag with three constituents, for instance, is a sort of triangular compound, and its properties cannot be inferred from the properties of any two of its constituents, taken two by two. It is a true ternary compound. In fact, if you do get a slag which corresponds to a definite chemical formula, it is usually the slag that you ought to get away from, since it is generally the least fusible of all the compositions that you can get of those ingredients; it occupies a maximum on the melting-point diagram, so that the effort of the blast-furnace manager is to get away from such a maximum which has a definite chemical composition, toward the eutectics which we all know do not correspond to definite chemical formulas. Let us further avoid considering that slags are practically chemical compounds of the ingredients, two by two; they are a combination of all the ingredients present, three, four, or five, whichever it may be, and they cannot be separated in the melted state into their constituents. When they have solidified, you may pick out the various constituents, two by two or three by three, but when in the melted state, the slag is homogeneous and does not contain any distinct constituent.

G. A. Guess, Toronto, Ont.—The paper deals only with iron blast-furnace slags, and is not of so much interest to copper and lead men. I remember that old paper of Dr. Hofman's, giving the formation temperature of various silicates. Some of our students attempted, a couple of years ago, to measure the fluidity of slag along those lines, to see how the fluidity curve would agree with the formation temperature curves as given by Dr. Hofman. We used a little graphite cup, with a hole in the bottom, similar to the cups that were used for the melting temperatures of matte as described in the paper read by Mr. Lathe and myself a year or so ago, and we checked that also by getting typical slags, such as siliceous slags from Granby, from Anaconda and from the Tennessee Copper Co. The hole in the cup was sufficiently small so that, when the slag started to run, it did not run stringy, but would run in such a condition that I concluded that if it was hot enough to run out

of that small hole, it was hot enough to run from the furnace. Although we did not get very far, we made a few synthetic slags, and work will be carried further still. It was interesting to compare what we would call the flowing temperature of a Granby slag with the flowing temperature of an Anaconda or Tennessee Copper Co. slag. I considered that those things were interesting and might be useful, but we did not get very much out of it.

A. L. FEILD.—In regard to the communication from Dr. G. K. Burgess on the standardization of furnace conditions, I conclude that he has seen only a copy of *Bureau of Mines Technical Paper 157* which described the present method. In the paper now under discussion, however, a measurement is given on diopside which shows that the temperature conditions in the furnace check with the theoretical within 20°C . This accuracy was considered sufficiently close for practical purposes, although it may be possible ultimately to secure greater accuracy. As to the measurement of slag temperature at the blast furnace, this problem was not entered into very extensively. The measurements were made simply to obtain some idea of what results are obtained by the use of an optical pyrometer without emissivity corrections, because in the literature there are a very large number of measurements made by means of optical pyrometers without corrections being applied, or at least references are made to such measurements. The correction requires a knowledge of the coefficient of emissivity. We were not equipped to obtain this value. The suggestion of Dr. Burgess, however, can be put into practice very easily with the aid of the emissivity measurement on blast-furnace slag which he has given in his discussion. Our thermocouple measurements on the other hand, were sufficiently accurate to determine what slag temperature prevailed at flush.

In regard to Prof. R. H. Richards' remarks, I would emphasize the fact that this idea of measuring viscosity by means of the torque on an inner concentric cylinder is one of the oldest ideas on the subject. It dates back about 40 years; it was emphasized in the paper that the idea was an old one. In regard to the use of Acheson graphite—while it is well known that graphite has a very small friction coefficient when a solid body is rubbed against it, the present method uses only a liquid in contact with graphite. There is no experimental evidence to show that there is a "slip." Measurements made on mercury in contact with glass with the highest precision at the University of Chicago have shown that there is no slip between the outer layer of mercury and the glass. As to the suggestion to make the surface of the outer cylinder rough, I may make the following statement: by using concentric cylinders which do not have any irregularities on their surfaces, there is obtained what is practically demanded—and by that I mean what is mechanically demanded—in

order to make this measurement what it is supposed to be, *i.e.*, a measurement of viscosity. The measurement is made without any turbulent flow in the liquid and without the production of eddy currents or impact. It is simply a question of shearing the liquid uniformly; and if a paddle is used or if the sides of the crucible are corrugated, the torque exerted on the inner cylinder is due partly to the viscosity of the substance, but quite largely to the kinetic energy and impact of the liquid, this impact being exactly the same as is obtained in ordinary hydraulic problems. It represents energy and can cause a deflection of the inner cylinder.

In reply to the remarks of Mr. Guess, I may say that in a later publication¹ of our results, the work done by him and Mr. Lathe was referred to. At the time the present article was written it had not come to my attention. Mr. Guess' method of using an orifice is undoubtedly at present the only method that can be used around a blast furnace to get any idea of the "softening temperature," apart from cone tests. While it may have certain advantages over a cone test, both of these tests are deformation tests and have the element of time entering into them. If we take a crucible with a hole in the bottom, and put in the crucible some pitch or asphalt—hard enough to cause a hammer to rebound from it—the pitch or asphalt will finally all flow out of the orifice, showing that a great many bodies supposed to be solid at ordinary temperatures have a measurable viscosity; that is, the viscosity is not infinite. To a great extent, therefore, in the case of all deformation tests, the results secured will be determined by the length of the experiment.

In reply to Mr. Fahrenwald's inquiry² as to the error introduced by the variation of the density of the slag, I would refer him to page 17, *Bureau of Mines Technical Paper 157*. Mr. Fahrenwald says that according to his experience highly basic slags have a greater tendency to adhere to Acheson graphite on cooling than acid slags. Laying aside for the moment the fact that "slip" has not been conclusively demonstrated to exist between a solid and a liquid for any single case, it is evident that the difficulty of removing a *cooled* slag from the graphite crucible is determined by the coefficient of expansion of the particular slag and can have no relation whatsoever with the *liquid* slag in its behavior toward the walls of the crucible.

While it is possible that further research may prove the existence of points of inflection in the first derivative of the temperature—viscosity curve which can be given a definite physical significance, the experimental data given in the report which is the subject of the present

¹ A. L. FEILD: The Viscosity of Blast-Furnace Slag and its Relation to Iron Metallurgy, Including a Description of a New Method of Measuring Slag Viscosity at High Temperatures, *Transactions of The Faraday Society*, 1917 (communicated by Sir Robert Hadfield, F. R. S.).

² *Bulletin* No. 123 (March, 1916), 390.

discussion are not sufficient to warrant a conclusion of such a hypothetical nature.

The amount of iron oxide present in blast-furnace slag lies usually between 0.05 and 0.50 per cent., so that the separation of metallic iron caused by the carbon monoxide atmosphere of the furnace is of no serious consequence. In the case of copper or lead blast-furnace slags, which contain considerable iron, the separation of metallic iron in undesirable quantities would doubtless be noted. This absence of easily reducible oxides in the iron blast-furnace slag precludes the existence of the undesirable "boiling" mentioned by Mr. Fahrenwald.

The practicability of the viscosity work which is the subject of the present paper is not confined to the possible operation of the furnace and apparatus at the blast-furnace by an operator of average skill and ability. The measurements recently made in this laboratory and other measurements now in progress will furnish the blast-furnace man with an accurate knowledge of the viscosity of his different slags, so that the only requirement imposed at the furnace is a chemical analysis of the slag. If the slag composition is known, our data will furnish complete information regarding the viscosity of the slag at all practicable furnace temperatures.

Dr. Sosman's suggestion³ that the viscosity or fluidity of pure oxides and of mixtures of two or three oxides should be examined is an excellent one. Such work has been in progress for the last 4 months and shows promise of very valuable results.

The Seasoning of Castings

Discussion of the paper of RICHARD MOLDENKE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 229 to 233.

A. E. OUTERBRIDGE, JR., Philadelphia, Pa. (written discussion).—The fact that iron castings improve with age has long been known. Many years ago the late Sir Frederick Bramwell, a distinguished engineer, formerly in charge of the metallurgical and gun-making department of the Woolwich Arsenal, England, determined to break up a large number of cast-iron mortars of an obsolete type that had been in stock for years. It was then reported to him that it was found to be much more difficult to break these old gun castings under the "drop" than it was to break similar gun castings recently made, under his own supervision; he changed the composition of the iron and still found that whenever castings made recently were broken up (on account perhaps of superficial defects) great disparity was observed in apparent strength between the old and new castings. Sir Frederick then made many other comparative tests, such

³ *Op. cit.*, 393.

as heavily charging with powder and firing these mortars repeatedly until destruction occurred by fracture. In each such test he was surprised and mortified to discover that the old guns, made years before he assumed charge of the works, withstood many more firing charges than any of the new guns that he submitted to these tests.

After thorough investigation of all of the brands of iron used, besides tests of "coupons" cut from the guns and other tests, physical and chemical, he finally arrived at the conclusion, which he incorporated in one of his official reports, that the difference in strength between the old and the new guns was not due to inferior metal or methods, but solely to the seasoning of the castings, which had long been made and allowed to remain undisturbed. Sir Frederick stated that in his belief the cooling strains existing in the castings when new gradually disappeared in course of years and the full strength of the metal in the castings was then secured.

In the year 1881 I was engaged at the large car-wheel works of A. Whitney & Sons (no longer in existence) in some metallurgical investigations which necessitated my personal attention frequently very late at night. It was quite a common occurrence for me to hear a sudden sound, like the firing of a pistol in the foundry yard, which was caused by the bursting, or breaking to pieces, of condemned chilled cast-iron car wheels that the inspector had caused to be thrown out while red hot, during the afternoon, instead of passing into the annealing ovens, owing to some molder's defects that were clearly apparent.

Castings of this nature are subject, of course, to abnormal cooling strains, as compared with ordinary gray iron castings, so that they almost always break apart unless put into an annealing pit as soon as they can be removed from the mold, and cooled very slowly during a period of several days. Then they are extremely strong castings if the cooling strains have been thoroughly released.

En passant, I may say that I have known a watchman who happened to be passing near a rejected car wheel at the moment it burst, in the night, to have his leg broken by the flying portion of the heavy wheel.

In 1888 another phase of this matter presented itself for study. At that time it was customary at the large machine tool works of William Sellers & Co., Inc., with which I had become connected as metallurgist, to "pickle" all small castings, such as pulleys below a certain size, in large vats containing sulphuric acid diluted with water, in order to remove the sand. A hard and fast rule obtained at that time that such castings must not be cleaned in the tumbling barrels for fear of breakage.

It was and is the daily custom to pour test bars from different portions of each melt of cast iron as it comes from the cupolas, and a card pattern was then made allowing 12 bars 1 in. by 1 in. by 15 in. to be cast side by side from one ladle of iron in one mold. These were all, of course, about

the same size and quality and would break under nearly the same transverse strains under normal conditions. Four of these bars were cleaned with a scratch brush, four were pickled and four were cleaned in the tumbling barrel; all of the pickled bars showed about 10 per cent. loss of strength as compared with the companion bars cleaned with the scratch brush, while the bars that were "rumbled" all showed surprising gains in transverse strength, varying from 15 per cent. to more than 30 per cent. increase.

This led me to undertake a long and elaborate investigation, covering several years, partly to ascertain the cause of these marked differences.

In 1896, the late William Sellers, then president of William Sellers & Co., Inc., Philadelphia, asked me to furnish an account of my tests in response to requests that he had received.¹

The claim was made in the paper that all iron castings are subject to cooling strains and that these strains can be, in great measure, quickly removed by subjecting the castings while cold to repeated shocks or vibrations. The opening sentence of this paper is as follows:

"It has been generally accepted as a fact that cast iron, under the influence of repeated shocks, becomes brittle, and will finally break under a blow which otherwise it would have withstood. It will probably surprise metallurgists, therefore, to learn that experiment disproves the supposed fact and establishes its exact opposite. The result of about a thousand tests of bars of cast iron of all grades, from the softest foundry mixtures to the strongest car-wheel metal, enables me to assert with confidence that, within limits, cast iron is materially strengthened by subjection to repeated shocks or blows."

The paper, though brief (comprising less than 10 pages) gives a résumé of a large variety of tests, together with tables, all showing that what I ventured to designate as "molecular annealing" of cold iron castings actually occurs from the effect of shocks or vibrations.

The rule at the Sellers' works was reversed so as to read that no small castings should go into the pickle tubs but must be cleaned in the tumbling barrels. Pickling, indeed, has long since been discarded entirely in this establishment.

After publication of the paper in the *Transactions*, the Franklin Institute in 1896 appointed a committee of Science and Arts to investigate these discoveries, and they made their own castings. Their report, dated May 5, 1897, entitled "On the Molecular Changes in Cast Iron Caused by Vibration," was printed in the *Journal of the Franklin Institute*, July, 1898. The report states "where the proper conditions are observed there will be a decided gain in strength in almost every case, at times running as high as 35 or 40 per cent. increase in the rumbled

¹ A. E. Outerbridge, Jr.: The Mobility of Molecules of Cast Iron. *Trans.* (1896), 26, 176.

over the unrumpled bar. Many others have made similar experiments with the same results, both with high and low silicon irons." A table is given of their tests, showing results of breaking 166 bars, of which half were rumpled and half were not rumpled, showing gains in strength up to 40 per cent. The deflection showed as high as 48 per cent. increase in more than one case.

A later development, which pleased me greatly, was the insertion of a new clause in the revised specifications for tests of iron castings prepared and adopted by the American Society for Testing Materials, to the effect that test bars must not be rumpled or otherwise subjected to shocks or vibrations before breaking, but must simply be cleaned of adhering sand with a scratch brush. This clause was inserted through the influence of Dr. Moldenke, who has shown a sympathetic interest in all of these investigations as well as in others that have grown out of them.

I have also been informed that similar clauses have been inserted in some private contracts abroad for iron castings made to specification as to strength of test bars.

In 1904, at the Atlantic City meeting of the Institute, a paper under the same title as before was presented as a sequel,² giving results of tests of extraordinary molecular movement of cast iron caused by repeated heating and cooling. Companion test bars that were cast in molds 1 in. by 1 in. by 15 in. were there exhibited in which some bars had been caused to "grow" or increase in cubical dimensions over 40 per cent. without distortion or material change in appearance of the metal (unless highly magnified) as compared with untreated bars of the same original size.

A number of interesting and valuable practical applications have been made of these observations, none of which were presented at the patent office for the securing of any proprietary rights therefor before publication. Some of these uses have been made by others who independently perceived applications of the principles involved that probably never would have occurred to the author.

In conclusion, I would say that Dr. Moldenke's paper is a timely one and is well worthy of careful consideration.

THE CHAIRMAN (HENRY D. HIBBARD, Plainfield, N. J.).—This suggestive paper brings up an important though obscure subject regarding which more knowledge is needed. In addition to the internal stresses due to cooling from the casting temperature in the cases cited by the author, there are others which occur in certain kinds of steel articles, which may properly be considered with them in connection with seasoning. Thus, a piece of cold-drawn shafting if turned in a lathe will not be straight because the resultant of the internal stresses remaining is

² *Trans.* (1904), 35, 223.

not the same as that of the stresses in the original piece. Again, a hardened steel shot contains internal stresses which may cause it to crack spontaneously, the most common case being that its point breaks off. This usually happens within a few weeks after hardening, if at all.

In the castings and shot, the internal stresses are due to different rates of cooling of the various parts, as the surface always cools in advance of the interior and must do so unless the rate of cooling is infinitely slow. In the cold-drawn shaft the internal stresses are set up by the cold work done in drawing the bar through the die. How do these stresses compare, and are they to be lessened or eliminated by the same seasoning? If so, what seasoning is best? Will a permissible time alone do it at uniform atmospheric temperature? Probably not. Will time alone do it at some moderately elevated temperature, say not over 200° C.? Possibly. Should the article be repeatedly heated moderately and then cooled? Probably. The daily fluctuations in temperature may be enough to effect seasoning in time. Old articles do not break spontaneously, nor presumably have excessive internal stresses.

The intensity of the internal stresses in a newly made or cooled article which seasoning might lessen are due to:

1. The rate of the change of temperature.
2. The shape of the piece.
3. The bulk or volume of the piece.
4. The elastic limit of the metal.
5. The ductility of the metal.
6. The coefficient of expansion of the metal.

It seems probable that heat, even if only of atmospheric degree, must be needed to effect the readjustment of the various parts of the article which results in diminishing or eliminating internal stresses. If, at any degree above absolute zero, heat causes vibration of the molecules, it seems reasonable to suppose that given a sufficient intensity of molecular motion ultimate positions will be such that each will be equally crowded or equally free on all sides, causing the annealed state which results in the greatest degree of softness and the highest specific gravity the metal is capable of possessing. Outerbridge's method of strengthening castings³ which he calls annealing, by a succession of light blows may be founded on increased molecular movements, as the title of his paper assumes.

Iron castings have been seasoned by being kept for some days or weeks in a core baking oven with the temperature fluctuating from atmospheric up to 110° or even to 150° C.

Some projectile specifications require that the hardened shot be heated to the temperature of boiling water. Those that pass this ordeal without cracking are not likely to break spontaneously thereafter. The reason may be either that the heating increases the internal stresses

³ *Trans.* (1896), 26, 176.

so that if a projectile is in a dangerous condition it will break then as a consequence, or that the heating if accomplished without rupture, by its annealing effect lets down the internal stresses so that the danger of spontaneous rupture is avoided.

Delayed suicide of a shot when it occurs no doubt comes from fatigue of the metal, as it often happens when the temperature, and therefore also the stresses, are stationary.

Stresses due to cold working may be less amenable to seasoning. The strength of wire is often due more to the drawing operation than to composition. Thus, wire for the cables of suspension bridges might be seriously weakened by seasoning if it took place to any considerable extent in a few years or even decades.

Howard found, in retesting iron test pieces which had been pulled beyond the elastic limit a quarter of a century before, that the elastic limit remained at the point at which the previous test was discontinued, showing no seasoning effect due to that length of time and ordinary temperatures.

On the other hand, as far as I have been able to find out, no pieces of good tempered steel are known which are more than a few centuries old and one surmises that seasoning has taken place in the old weapons. A friend of mine who tried with a file the hardness of some old pieces of steel in the British Museum came near being arrested therefor.

The whole subject needs investigation, the work being started now and being finished perhaps by future generations.

R. MOLDENKE.—We have with us Mr. Outerbridge, who came in while his discussion was being read, and I think he has something to show us.

A. E. OUTERBRIDGE, JR.—Owing to a delay in arrival of the train from Philadelphia I reached this room in time to hear the latter portion of my own paper being read. It reminded me of a visit I paid to Menlo Park, N. J., about 40 years ago, on invitation from Mr. Edison, to hear the human voice reproduced on his first phonograph. After adjusting a sheet of tin-foil on the cylinder he asked me to speak into the mouth-piece, then reversing the cylinder and again turning the crank, I was astonished to hear the words "Mary had a little lamb" proceeding from the phonograph in an entirely unfamiliar voice.

I should, perhaps, apologize for the somewhat personal nature of the paper that has just been read, but as it is a reminiscence of original investigations along the lines of Dr. Moldenke's address this could scarcely be avoided entirely. I will not repeat any of the matter contained in the written discussion, but I now take pleasure in exhibiting some of the same cast-iron test bars which I had the honor of first showing before this Institute at its annual meeting in February, 1904.

Here are two bars of gray cast iron which were poured in one mold, from one hand ladle of iron, from identical patterns; they were both originally the same size, one bar remains as cast, the other has been caused to "grow" or increase in cubical dimensions 40.98 per cent., while still retaining its shape and metallic appearance.

R. MOLDENKE.—Tell how these bars were treated.

A. E. OUTERBRIDGE, JR.—The expanding of these iron bars was done by putting a dozen bars into a steel pipe which was 20 in. long, the ends stopped up. That pipe was put into a gas furnace with a pyrometer measuring the temperature, heated for 4 hr. and then taken out and allowed to cool. When it was cold, the bars were measured. After three heatings, the original shrinkage of these bars had been overcome and the bars were then 15 in. long, i.e., the length of the mold. After 40 heatings, this bar which was originally $14\frac{13}{16}$ in. long became $16\frac{1}{2}$ in. long, and 1 in. square section became $1\frac{1}{8}$ in. It showed an increase of almost 41 per cent. in cubical dimensions while still retaining its metallic properties. That is the way in which these bars have been caused to grow, and a great many practical applications have been made of that property since 1904.

LEONARD WALDO, New York, N. Y.—What was the change in the specific gravity?

A. E. OUTERBRIDGE, JR.—The change in specific gravity bears a very close relation to the amount of expansion; pieces of cast iron originally 7.25 sp. gr. became 5.49. The increase in weight of the expanded bar is so small that I considered it negligible.

A MEMBER.—Do I understand that the specific gravity was diminished a third as the bar expanded?

A. E. OUTERBRIDGE, JR.—It diminished in proportion to the amount of expansion but not absolutely, 5.49 is the lowest specific gravity that I found of a gray iron bar which, before expansion, was 7.25.

R. MOLDENKE.—The figure you have given is practically the one obtained in the briquetting process for cast-iron borings, in which a pressure of perhaps 35,000 lb. per square inch is applied. The loose material is thus brought together to a density corresponding to that given gray cast iron which has been made to "grow," or open up in structure.

A MEMBER.—How was the specific gravity ascertained?

A. E. OUTERBRIDGE, JR.—Your question recalls another very interesting thing which I had entirely forgotten. In order to determine the specific gravity of these bars, pieces were cut off from the ends of a bar before expansion and from the ends of the same bar after expansion, from

less than 15 in. in length to nearly 17 in., with corresponding increase in cross-section. I used the ordinary method in taking the specific gravity of the bar before expansion by weighing in air and then in distilled water but I experienced unexpected difficulty in taking the specific gravity of the expanded metal for when immersed in distilled water bubbles of gas came off from the entire surface until I coated the expanded dried piece with a little waterproof lacquer to prevent the water from oozing in through the intermolecular spaces, driving the air out. It was just as easy to take the specific gravity in that way, of the expanded bar as of the other bar.

LEONARD WALDO.—May I ask what the temperature was that caused this remarkable growth or permanent expansion?

A. E. OUTERBRIDGE, JR.—The most efficient temperature is from 1,450° F. to 1,550° F., at lower temperatures you must heat the bars a great many more times, and at still lower temperature, say below visible redness, you can heat forever and get no such effect, unless heated in superheated steam, which causes considerable permanent expansion, as in the case of cast-iron valves.

LEONARD WALDO.—Suppose these growths took place in a vacuum?

A. E. OUTERBRIDGE, JR.—Prof. Carpenter of Manchester, England, while investigating these discoveries of growth of gray cast iron, heated bars with an electrical heating device in a vacuum, but his experiments were very inconclusive. He said that in some cases they did not seem to grow and in others they did.

MR. WALDO.—I understand that your remarks apply only to cast iron.

A. E. OUTERBRIDGE, JR.—Yes, only to gray cast iron, not to white iron, not to steel, not to wrought iron, not to bronze, not to copper. In fact, I have tested nearly all metals and alloys (except the rare and precious metals) and have found that gray cast iron alone possesses the peculiar property of "growth" or continued increase in cubical dimensions after being repeatedly heated and cooled.

M. H. MEDWEDEFF, Springfield, Mass.—If castings grow upon annealing, I would like to know how it is that castings are made to size, and retain their dimensions on annealing.

A. E. OUTERBRIDGE, JR.—All gray iron castings that are subjected to alternate heating and cooling above "cherry red" increase in size. There is nothing new in it, only that these experiments were made to find out why it was that grate bars warped and other castings cracked and twisted on being heated and cooled. The cause was heretofore supposed to be warping. We now know that there is actually an extraordinary increase in cubical dimensions. White iron (or "malleable iron castings")

made of white iron) in which practically all of the carbon is chemically combined with the iron, does not, on the other hand, expand, when converted into gray iron by heat, sufficiently to overcome the original shrinkage of the metal. I stated this fact very plainly in my paper presented to this Institute in 1904 but it was questioned by the English investigators who brought it out with an unimportant modification as an entirely new discovery some time later. I said that the free carbon in converted malleable iron was quite different from graphite existing in gray iron normally and is known as Ledebur's "temper carbon."

Among the practical applications that have been made of this property of gray cast iron that I recall at the moment, is that of a large number of cast-iron radiators that were too short to couple up with other similar radiators owing to an error in the new pattern. These were condemned and were being melted in the cupola when it was suggested that several of the castings be subjected to red heat over night in an annealing oven; after two such treatments the radiator castings had grown to the proper length and all that had not been destroyed were then treated and were actually placed in service in a large hotel in your city. Cast-iron pistons of internal-combustion engines and of pumps that have become worn too small to be efficient have been restored to size by this simple process of permanent expansion of gray iron by heat. These investigations all illustrate the mobility of molecules of gray cast iron and have therefore close relation to the subject of Dr. Moldenke's paper.

Temperature Measurements in Bessemer and Open-Hearth Practice

Discussion of the paper of GEORGE K. BURGESS, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 293 to 306.

J. W. RICHARDS, South Bethlehem, Pa.—I think my affections are still rather with the radiation pyrometer than the optical pyrometer, for practical use, and I wish that Prof. Burgess would use the two together, under the same conditions, using the optical as his standard, if he prefers. Then, assuming the optical emissivity which he feels is correct for the optical pyrometer, he could calculate the thermal emissivity for use with the radiation pyrometer. If we could thus get reliable values for the thermal emissivity, we could then use the radiation pyrometer, and would avoid the use of the standard lamp, which has to be frequently calibrated for use in the optical pyrometer.

It is very advisable to distinguish carefully between the terms optical emissivity and thermal emissivity. There is considerable confusion caused by the bare statement that molten iron has an *emissivity* of so much, meaning by that its optical emissivity, for a light wave of a certain length, whereas what we want in the radiation pyrometer is its total ther-

mal emissivity. I think it would be better to always use the terms optical emissivity and thermal emissivity. I should prefer to go on using the radiation pyrometer where possible, for it does not need a standard of comparison, or a standard light, but needs only the information as to what is the total thermal emissivity of the material being examined.

G. K. BURGESS.—Regarding the use of the optical or radiation pyrometer, I purposely avoided the discussion of their relative merits for this purpose. I am perfectly willing that Prof. Richards should use the radiation pyrometer if he wants to get up as close to the hot metal stream as he has to do, to use it. The main difficulty in the use of the radiation pyrometer, as at present constructed, is the danger of not getting sufficient aperture when sighting on a small stream and the optical angle is such that you have to put the instrument within a few inches, almost, of the stream. That is a rather serious objection for convenient use. I have taken observations within 10 ft. of the stream from an open-hearth furnace, but I do not care to do it as a matter of practice; whereas, with the optical pyrometer, you can be 50 ft. away. I took some observations with the two together only the day before yesterday, and my preference has not been changed by that experience.

R. C. DRINKER, Pittsburgh, Pa. (communication to the Secretary *). —A brief survey of the interesting paper by Mr. Burgess indicates that any discussion I might offer would be of little value, since my work of late years has been in the direction of a measuring apparatus so simple, and yet fairly accurate, that it may be placed in the hands of an operator just able to read figures—a condition so frequently found among steel melters.

Although my methods are crude, in comparison to the able work of Mr. Burgess, yet my contention "that melting temperatures are quite as critical as those of heat treatment," makes it immaterial whether those determinations are accurate temperature measurements, so long as they are comparatively accurate, *i.e.*, enable one to duplicate tomorrow, a satisfactory temperature of today, and thus in time establish valuable data.

* Received Feb. 20, 1917.

The Manufacture of Weldless Steel Tires for Locomotive and Car Wheels

Discussion of the paper of G. AERTSEN, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 69 to 74. •

THE CHAIRMAN (HENRY D. HIBBARD, Plainfield, N. J.).—This excellent paper fills a gap in our *Transactions* and is most acceptable.

In the early days another way of casting tire ingots, sometimes called cheeses, because of their shape, was to use a 22-gage sheet-iron cylindrical can for each ingot of the proper size to give the weight desired. These cans were bottom cast in groups, usually of four, each can being surrounded by a heavy iron ring somewhat taller than itself, the top of the can being covered by a circular cast-iron plate having a small conical riser in the center, about $2\frac{1}{2}$ in. in diameter at the bottom and 10 in. high. The top plate was clamped down to resist the upward pressure of the fluid metal and the space between the can and the outer ring, usually about 2 in. wide, was filled with dry sand. The riser permitted the escape of some of the air displaced by the steel and also increased the pressure in the steel when the mold was filled, and so diminished somewhat the tendency for gas holes to form.

But little was known of segregation and pipe in those days, though the center of an ingot was understood to be the worst part, and it was separated in part in the disk which was punched out in perforating the ingot. This disk was about 8 in. in diameter and from $\frac{3}{4}$ to 2 in. thick, the variation in thickness being to rectify the weight. When an ingot was too heavy, a thicker disk was made by not driving the punch in quite as far as when the ingot weight was right. Then the bloom was turned over and the punch driven in on the other side, which finished the punching and detached the disk. The tire bloom was then "becked" and later reheated and rolled as described in the paper. The forging was done with a 10-ton double-acting steam hammer.

The steel generally speaking had lower carbon, from 0.45 to 0.55 per cent., than tires today have, and the tires were smaller, the most common size of ingot weighing 1,060 lb. each.

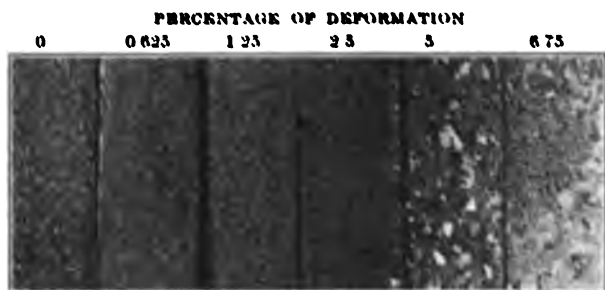
BRADLEY STOUGHTON, New York, N. Y.—This paper was specially requested by the Iron and Steel Committee, because that Committee thought so important an industry should have its history written by one who was familiar with it, and before he had forgotten, or all the rest of the world had forgotten, its early stages. Mr. Aertsen has said that it is not an exhaustive research. I think it would not be any different if it were an exhaustive research, because our library made a careful attempt to find some written light on the history of these tires, but very little was found.

Those who knew anything about it kept it to themselves or else were unable to put their ideas into writing, so that I really think we have a very valuable paper in this one by Mr. Aertsen giving us an account of the early days of this industry and the many important phases that it went through at the beginning and in the middle stages.

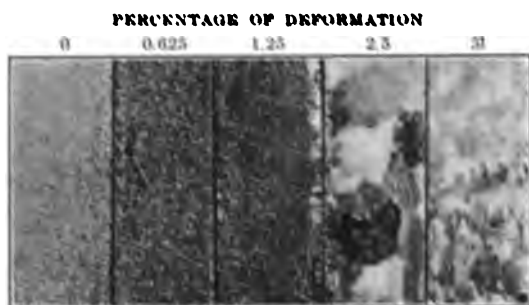
Recrystallization after Plastic Deformation—Grain Growth Phenomena in Metals—On Grain Growth

Discussion of the papers of HENRY M. HOWE and ZAY JEFFRIES, presented at the New York Meeting, February, 1917; and printed in *Bulletin* No. 118, pp. 1851 to 1860, *Bulletin* No. 119, pp. 2063 to 2073, and *Bulletin* No. 120, pp. 2111 to 2117.

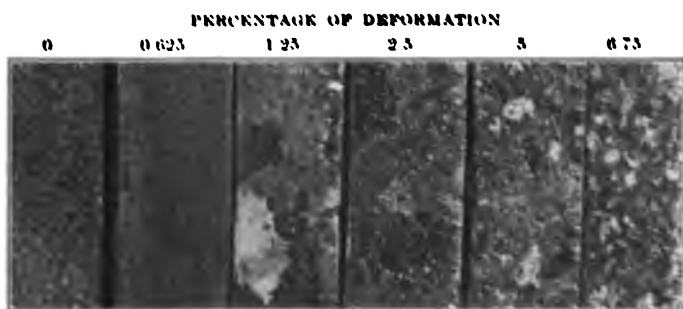
W. E. RUDER, Schenectady, N. Y.—In 1913 I presented a paper before this society on Grain Growth, and at that time it seemed to me that the only explanation for grain growth was that of critical strain. I do not think I published the photographs at the time, but I made some experiments on the effect of stretching sheets of metal. This allowed for fairly accurate determinations of the amount of cold deformation. At the time I worked with silicon steel because one could see the change in grain growth with the naked eye. I stretched about 10 samples from 0.5 per cent. to 10 per cent., and, after heating to $1,050^{\circ}\text{C}.$, I found that the maximum grain growth occurred at about $2\frac{1}{2}$ per cent. cold deformation. This I designated "the critical strain" for maximum grain growth. About a year ago I talked with Prof. Jeffries, who explained to me his ideas of grain growth. It occurred to me that it would be a very easy thing to check the strain hypothesis against the strain and temperature hypothesis, if we took a certain number of these strips, strained exactly alike and heated to different temperatures; in that case the maximum grain growth ought to appear at a different percentage of deformation for each temperature. These experiments showed that there was, as Mr. Jeffries shows in his present paper, a decrease in germinative temperature with increased cold deformation. The samples were stretched from $\frac{1}{2}$ to 8 per cent. and heated for 10 hr. in hydrogen. Samples were introduced into a furnace previously heated to temperature. A maximum grain growth was obtained at 5 per cent. deformation, when heated to $750^{\circ}\text{C}.$; at $2\frac{1}{2}$ per cent. when heated to $800^{\circ}\text{C}.$; at $1\frac{1}{4}$ per cent., if heated at $950^{\circ}\text{C}.$; and at $\frac{5}{8}$ per cent. when heated at $1,100^{\circ}\text{C}.$ This shows very clearly that the germinative temperature decreases with the increase in cold deformation. The growth was very much more marked, however, at the high temperature, *i.e.*, the grain grew more rapidly and attained a much larger size than at lower temperature. In considering the photographs (Fig. A) those along the lateral edges should be ignored because



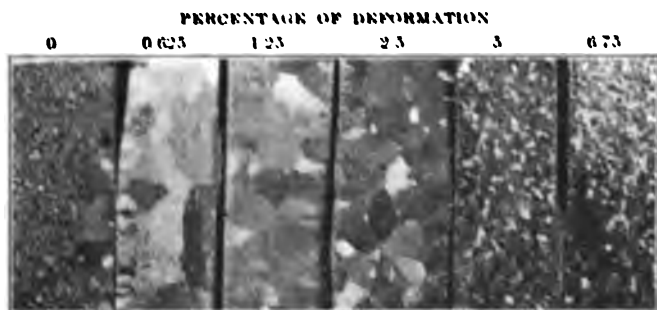
ANNEALED AT 750 °C. FOR 10 HR.



ANNEALED AT 800 °C. FOR 10 HR.



ANNEALED AT 950 °C. FOR 10 HR.



ANNEALED AT 1100 °C. FOR 10 HR.

FIG. A.

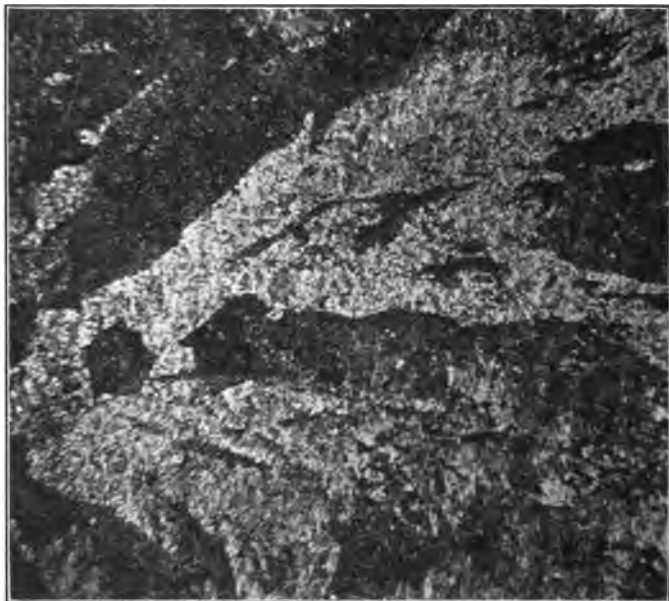


FIG. B.—PRESSED ELECTROLYTIC IRON ANNEALED AT 1000 °C. FOR 3 HR. $\times 50$.

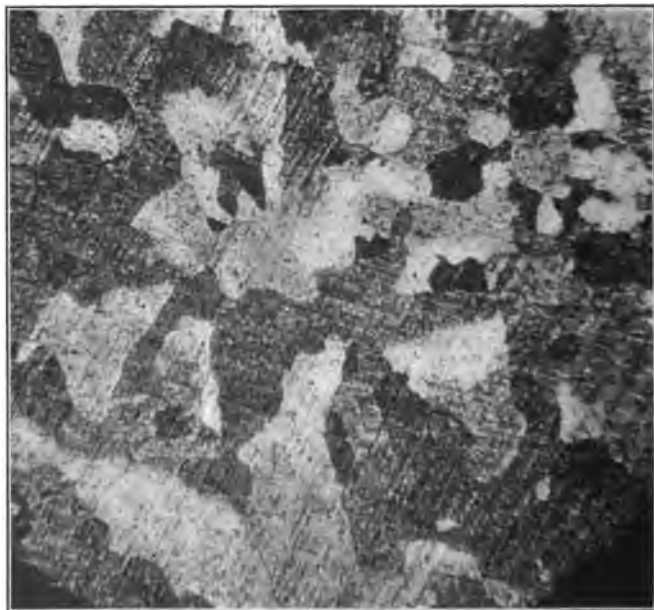


FIG. C.—SAME SAMPLE AS IN FIG. B. REHEATED TO 1250-1300 °C. FOR 3 HR.

they were started by the strain gradient set up by the cutting of the strips. The amount of strain so induced is of course not measurable. Another point which was brought out by Prof. Jeffries' hypothesis is shown in one of the photographs (Fig. 14) which I published at that time. This was the only sample upon which I tried the experiment of the effect of temperature gradient, and in that experiment I placed a piece of sheet steel in the furnace which was at a temperature of $1,340^{\circ}\text{C}.$, at the hottest zone, and was graduated from there on out to room temperature at the cold end. In this case I found that I always got an enormous grain growth in the range of about $900\text{--}1,100^{\circ}\text{C}.$, and always at the same point, in any number of pieces I put in. The grain growth in this case was always much greater than it was in any other heating. This, of course, falls in with Prof. Jeffries' hypothesis, or is easily explained by it, because at the point of germinative temperature, the grains had an enormous number of inactive grains to feed upon, *i.e.*, those that were below the germinative temperature; and, therefore, they always formed enormous grains. There is one case I have not been able to explain altogether and that is the formation by cold deformation of small grains inside of large grains (cf. Fig. 9, *Trans.* (1913), 47, 569–86). I found at that time that these small grains never form except at a high temperature ($1,050^{\circ}\text{C}.$) a temperature much greater than that at which the original strained sample began to show growth ($725^{\circ}\text{C}.$). I am at a loss to see exactly why the large grains which completely surrounded the small ones did not re-absorb the grain fragments at this high temperature. Another case which came up recently was one of which I was reminded by Prof. Jeffries' reference to pressed metals; namely, in case of a slug of pressed pure electrolytic iron powder, I was able to get large grains in the pressed metal by heating at $1,000^{\circ}$ (Fig. B), but this large grain was broken up on further heating to $1,300^{\circ}$ —broken up into smaller, more even grains, which we might call normalized grains (Fig. C). I do not know why they should change at that high temperature, particularly since the first heating was above the critical range.

FREDERICK E. CARTER, New York, N. Y.—I am interested more particularly in this question in its application to platinum. Until now very little has been done on the platinum group of metals, one reason being that it has been very difficult to get pure metals. I am fortunate enough to be able to get them pure, and have just started some work on the metallography of the subject. Some preliminary work on the subject of the platinum alloys which I did with Prof. Campbell, of Columbia University, fits in nicely with some points that have been raised here. I found that with pure platinum you readily get recrystallization at about $1,000^{\circ}\text{C}.$ The platinum-iridium alloys, however, will not recrystallize at that temperature, and even at $1,100$ and over, recrystalliza-

tion only slowly takes place, the temperature depending on the percentage of iridium, *e.g.*, alloys of 25 and 30 per cent. iridium require a higher temperature for annealing than 10 or 15 per cent. iridium. I also obtained a regular series of crystal sizes, the higher iridium alloys showing much smaller grain. That is, of course, to be expected. I have been led to believe, too, that there seems to be a maximum size for these crystals. With the alloy containing 25 per cent. iridium the grain does not seem to grow even with continued heating, and the grain of the alloy with 30 per cent. iridium does not grow with continued heating to the same size as that of the 25 per cent. alloy. Just over $1,150^{\circ}$ there is no change at all in these higher alloys, so it must mean that their "germinative temperature" is between $1,100^{\circ}$ and $1,150^{\circ}$. Does it mean that at just over $1,150^{\circ}$ I have reached the temperature at which the grains of the higher alloys would no longer increase in size?* Several points have come up that were of interest to me, like the gas content of the metal and the temperature gradient tending to cause an increase in the size of crystals; these of course are very important points in the use of thermocouples. A good deal of what I have heard today aroused my curiosity and gave me much information which may be applied to studies of the behavior of the alloys of the platinum group.

J. W. RICHARDS, South Bethlehem, Pa.—I think metallurgists will find some new methods of studying grain growth if they take cognizance of the advances recently made in the study of crystals by the Roentgen rays. Most of you know that the Nobel prize was presented to Prof. Bragg, of Cambridge, for investigation on the structure of crystals by the Roentgen rays, and we actually know many times more about the structure of crystals through this investigation than was ever known before; in fact, the actual molecular structure of the crystals is now known. I think the use of these methods might obviate the necessity of depending upon the uniformity of etching tint to find the orientation of the crystals. Recrystallization *evolves* energy, as Mr. Jeffries has well said, and therefore it should take place very rapidly at the germinative temperature, if the temperature is not passed through too rapidly, because it is an exothermic reaction, and therefore has the high reaction velocity characteristic of exothermic reactions.

* As the result of further experiments, I desire to correct my statement that, at $1,150^{\circ}$ C., the grain sizes of platinum and iridium alloys remain constant even with continued heating. I have since submitted them to a further prolonged annealing and have found the grains showed a small growth; the grains are, therefore, still within the germinative range.

In connection with this discussion of grain growth, it is interesting and suggestive that precious-metal thermocouples become defective due to crystallization when kept for long periods at about $1,100^{\circ}$ C. more readily than when they are used at either higher or lower temperatures. F.E.C.

THE CHAIRMAN (H. M. BOYLSTON, Cambridge, Mass.).—There is one little point that occurred to me in connection with Prof. Jeffries' remarks about the speed with which grain growth can take place. He says that the probable reason that, in Prof. Sauveur's bent bar, the outside layers on the convex side of the bend did not show any growth was that they had gone too rapidly through the germinative temperature. I am quite certain that his pieces were heated from the cold, in a cold furnace, so that the heating to 650° must have been comparatively slow. It seems difficult to realize that growth can take place so quickly as to cause enormous grain size in 6 sec. at the germinative temperature, while that piece failed to show any growth. Prof. Sauveur's bent bar must have been kept for at least 6 sec. at most of those intermediate temperatures between room temperature and 650° .

J. W. RICHARDS.—Is it not possible that the growth may have occurred during the heating of the metal and been subsequently destroyed by the higher temperature?

ZAY JEFFRIES.—No, that could hardly have taken place, because, in Prof. Sauveur's experiments, the temperature was never raised above 650°C. , and the breaking up of the coarse grains due to allotropic modifications occurs only in the region near 900°C. ; but the explanation for Mr. Boylston's apparent exception in this case is very easy: In the first place, this remarkable growth in the 6-sec. period which I have previously mentioned, occurred at a high germinative temperature. By special means of introducing obstructions to grain growth, we can change the germinative temperature and even raise it so that theoretically it would be above the melting point of the metal. As the temperature of the germination increases, the speed of the growth increases very rapidly. In a certain example it took an hour and a half, or thereabouts, to cause considerable coarsening in a particular kind of metal at a relatively low germinative temperature; whereas it took about 1 min. at a relatively high temperature, to form very coarse grains with the same metal. The difference between these two samples was that certain non-metallic obstructions were used in one case to raise the germinative temperature, so that these enormous speeds of coalescence of the grains took place only at a relatively high temperature. If the germinative temperature is low, as is the case in Prof. Sauveur's samples, then the rate of coalescence of the grains is slow, and that partially accounts for the absence of coarseness in the more severely deformed portions of his samples.

In my previous remarks, I should have mentioned another important factor, namely, with increasing degrees of plastic deformation, coarsening by means of furnace heating becomes more difficult. Prof. Sauveur's samples were heated in a furnace. Furnace heating practically eliminates the temperature gradient from axis to surface, of the piece of metal. As

the degree of deformation increases, the strain gradient decreases; *i.e.*, the recrystallization temperatures in different portions of the same deformed sample become more nearly the same, for instance, in severely cold deformed metals, the recrystallization temperature, or germinative temperature, of the surface of the sample is the same as the germinative temperature of the axis, so that, if the temperature is uniform from surface to axis, we can get no marked grain-coarsening condition.

I can explain, I think, Mr. Ruder's inclosed grains along this same line of reasoning. He mentions that at high temperatures he usually gets small grains inclosed within large grains. The fact that a grain is small and abuts a large grain, does not necessarily mean that that small grain must be absorbed. The general rule is that it will be, but it may have certain characteristics which permit it to exist as an individual grain, even in contact with a larger neighbor, such as Holland abutting Germany. It may coalesce after a considerable time, but the fact that it exists as an individual small grain in the interior of a large grain is explained, in my opinion, in this manner: The very high temperatures which Mr. Ruder used would cause remarkably fast growth. I imagine that a few seconds would be sufficient, in certain of the instances which he cites, to cause a very large area of the metal with which he was working to form into one grain. Let us call this a germinant grain. Supposing that another grain had started to form as a germinant grain, not far distant from this particularly large and active grain; this second germinant grain would have reached a size which would make it somewhat resistant to coalescence with the larger grain but yet might not reach such a size that it could absorb the adjacent smaller inert grains as rapidly as could the larger, more active, germinant grain. In that event, the larger, more active, germinant grain would absorb the inert grains at a rapid rate and would advance its boundary toward the smaller germinant grain. Meeting a temporary obstruction in this smaller germinant grain, due to its rather large size, when compared with other adjacent grains, the larger germinant grain would actually grow around the smaller germinant grain and would even absorb the small inert or inactive grains surrounding the smaller germinant grain. In that manner there would be one grain entirely inclosed within another larger grain. The condition may obtain in which several small grains are inclosed within one large grain.

It seems that after a grain is entirely inclosed within a larger grain, the former becomes even more resistant to coalescence with the larger grain than when the two simply abutted. I attribute this to a balance of forces which act in all directions on the same grain at the same time and establish an approximate equilibrium; whereas, when a large grain abuts a smaller grain, there is not sufficient resistance in the smaller grain to withstand the force of the larger grain, thus producing a state of unbalance or unstable equilibrium and the only way to establish more

stable equilibrium is for the smaller grain to be absorbed by the larger one.

I think that Mr. Ruder's experiment with pure iron can be explained according to the allotropic changes, and in that respect I would like to refer to some work by Stead and Carpenter,¹ on the grain size changes in electrolytic iron due to the allotropic modification which occurs at about 900° C. They found, I think, something similar to what Mr. Ruder seems to have found in this particular case. Heating to a high temperature, Mr. Ruder formed, not alpha or beta iron, but large grains of gamma iron; then, in cooling past 900°, these large grains of gamma iron broke up into non-gamma iron from several centers at the same time and hence caused a small grain structure of alpha iron which Mr. Ruder observed. On the other hand, when he heated to 1,000° C., he formed small gamma-iron grains, and on cooling below 900°, these small gamma-iron grains would form germinant centers of non-gamma iron and these germinant centers would absorb the small gamma grains as fast as the allotropic change took place from gamma to non-gamma iron. In my opinion, this is an ideal condition for a rather wide range of grain size with change in the thermal treatment due to the allotropic modifications and consequent change of structure due to the change from one modification to another.

Prof. Richards' remarks are very much to the point concerning Prof. Braggs' X-ray spectrometer. In my paper a year ago, I brought that out and advocated the study of these crystals and the study of orientation with the Bragg X-ray spectrometer. Unfortunately I have not personally been able to get an instrument with which to work, but I know that work is being done on that line at the present time in the United States and I think that it will yield some very excellent results.

Regarding the remarks of Mr. Carter on platinum and iridium alloys: There is a general tendency to form small grains in solid-solution alloys, such as the alloys of platinum and iridium, and the general law of the heightening of annealing temperatures or recrystallization temperatures, here applies up to a certain point at which will be found the maximum, but the grain size should usually increase above 1,100°C. with extended time at higher temperatures. It may be possible under the germinative temperature laws, to produce the largest grains at a temperature between 1,000 and 1,100. On the other hand, the purity of the substance may be such that the actual fast-growth temperature due to the germinative laws will be in the neighborhood of 1,200 or 1,300° C. or even higher than that, according to the special conditions. This matter of equilibrium grain size is not yet well understood. Prof. Howe puts the case very well when he states that it is doubtful. Some metals seem to change in grain size at a high temperature very much more rapidly than others. I have one particular series of tests in mind where I found after repeated

¹ *Journal of Iron and Steel Institute* (1913), **88**, 119.

tests practically no difference in grain size, as the result of heating for 30 sec. and for 10 hr. On the other hand, I have found other examples in which the grain size had changed enormously during the interval between 30 sec. and 10 hr. A grain may be in equilibrium when it is 10,000 or more times as long in one direction as it is in another, or it may be in equilibrium when it is exactly equiaxial. Probably the final structure with infinite time, would be toward one grain in all pieces of metal, regardless of shape or size, if the temperature were sufficiently high and the time sufficiently long. However, the rate of grain growth is so rapid at the beginning of recrystallization that in some metals an apparent equilibrium grain size seems to obtain after a few minutes heating or in extreme cases even a few seconds is sufficient. I call this an "apparent equilibrium grain size" not because no further grain growth will take place with extended time at the given temperature, but because such grain growth is so slow as to be negligible in the time periods ordinarily used in industrial heatings, such as annealing or heating to obtain definite structure.

JOHN A. MATHEWS, Syracuse, N. Y. (communication to the Secretary).*

—I have read with much interest and profit the papers by Profs. Howe and Jeffries in regard to the phenomena of grain growth in metals which have been subjected to stress.

The authors rightly conclude that much work remains to be done, and, in connection with future work, I would offer the following suggestion as a method of approach which might yield quantitative information in relation to the effect of stress, time and temperature.

My suggestion is, that the metals to be examined be prepared in the form of torsional test-pieces, for example, in the form of 1-in. or $1\frac{1}{4}$ -in. round bars, 12 to 18 in. long. These bars could then be twisted in the torsional machine, stopping the operation before fracture, and before any reduction in area had taken place at any particular point. In this way, an estimate of the stress could be had for all zones of metal from the center to the outside. The bars thus twisted could then be cut into disks, say, $\frac{1}{4}$ in. thick, and these disks could be further quartered or halved. In this way, a large amount of material that had been subjected to uniform conditions would be available for variable time and temperature experiments. The pieces could be examined, microscopically, lengthwise and transversely, and variable conditions, produced by differences of stress from the center to the outside, could then be studied, and very nearly the exact stress at each zone could be calculated.

I shall be glad to coöperate with either of the authors in performing the torsional work on any materials they care to submit.

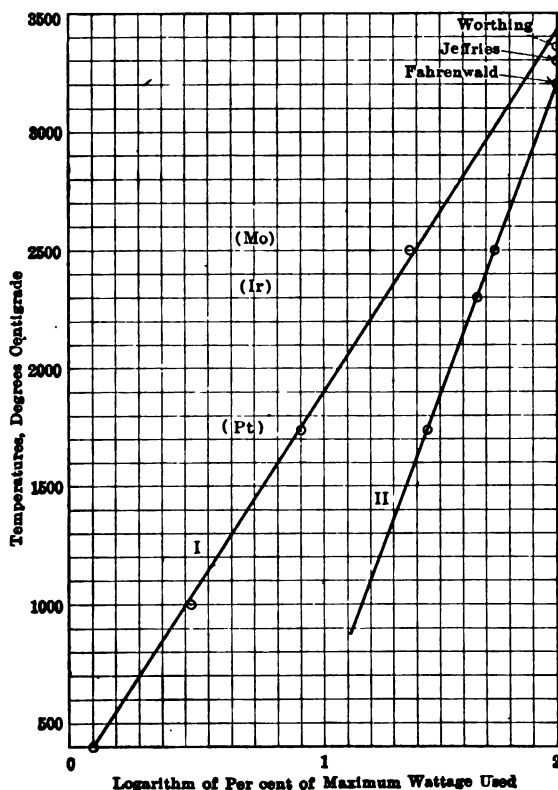
* Received Mar. 6, 1917.

Tungsten-Molybdenum Equilibrium Diagram and System of Crystallization

The System Tungsten-Molybdenum

Discussion of the papers of ZAY JEFFRIES and F. A. FAHRENWALD, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 115, July, 1916, pp. 1225 to 1236, and *Bulletin* No. 114, June, 1916, pp. 1049 to 1055.

J. W. RICHARDS, South Bethlehem, Pa.—I think that the diagrammatic results can be much more clearly set forth if they are plotted as follows: The present diagram, shown on page 1227 of Mr. Jeffries' paper,



plots the variation of wattage with temperature in a very inconvenient and uncertain way. If, however, you plot as abscissas the *logarithms* of the wattage, you get very nearly a straight line, so that you can interpolate the values with much greater accuracy than you can by using

simply the wattage. In the accompanying diagram the abscissas run from 0 to 2, representing the logarithms of 1 to 100 per cent. of the wattage employed. The ordinates are the melting points. Line I represents Jeffries' results, which extrapolated from 2,500° C. seem to point to 3,430° C. as the melting point of tungsten. Line II gives Fahrenwald's results, which register perfectly for the melting points of platinum, iridium and molybdenum, and also with Fahrenwald's assumed melting point of tungsten, 3,200° C. Worthing gives the latter temperature as 3,360° C.

ZAY JEFFRIES.—I would like to say that that is a very helpful suggestion.

THE CHAIRMAN (H. M. BOYLSTON, Cambridge, Mass.).—I notice that Prof. Jeffries draws a full line for the liquidus in the equilibrium diagram and dots the solidus, and that Mr. Fahrenwald does just the reverse. I wonder if there is any explanation of that.

ZAY JEFFRIES.—The points as actually observed are indicated by little circles, and I noticed, by microscopic examination, a little coring. By coring is meant the separation of the tungsten-rich material from the molybdenum-rich material. Coring indicates that there is a difference in temperature between the solidus and the liquidus; what that difference is, I have no means of knowing, so I suppose that I really should not have put in the indicated difference in temperature between the liquidus and solidus. If I am not mistaken, however, I mentioned the fact in the paper that the liquidus and solidus curves were only tentative.

Roll Scale as a Factor in the Bessemer Process

Discussion of the paper of A. PATTON and F. N. SPELLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 139 to 148.

E. T. McCLEARY, Youngstown, Ohio (written discussion).—Perhaps there is no question before the steel manufacturers of America today that causes them more worry than that of maximum production, together with good quality, and Messrs. Patton and Speller have clearly shown in their excellent paper how this may be accomplished. Any operation that tends to uniformity is certainly conducive to good practice, and as the use of roll scale in the converter eliminates to a large extent the variations in temperature due to varying silicon content in the pig iron, it should certainly give a more uniform product.

At the plant of The Youngstown Sheet & Tube Co. the use of roll scale was started in the year 1908. A great many tests have been con-

TABLE 1.—Average Results from Various Tests at the Youngstown Sheet and Tube Co.

Heat	Mixer Iron				Cupola Iron				Average Iron to Vessel				Weight of Charge in Pounds															
	Total C		Silica		Mn		Total C		Silica		Mn		Total C		Sil		Mn		Iron		Pig Iron		Steel Scrap		Roll Scale		Total Weight	
No. 1 Without scale.....	4.06	1.96	0.65	1.10	3.60	1.10	0.45	3.93	1.70	0.59	25,500	None	4,000		29,500													
	4.06	1.94	0.67	1.03	3.82	1.03	0.43	3.99	1.67	0.60	25,240	None	2,910	1,000	29,150													
No. 2 Without scale.....	4.04	1.97	0.68	1.06	3.64	1.06	0.43	3.94	1.75	0.62	24,420	None	4,000		28,420													
	4.06	2.00	0.69	1.03	3.64	1.03	0.42	3.96	1.77	0.63	25,260	None	3,000	1,000	29,260													
No. 3 Without scale.....	4.10	1.93	0.68	1.08	3.70	1.08	0.43	4.00	1.72	0.62	24,000	None	3,985		27,985													
	4.02	1.95	0.66	1.05	3.72	1.05	0.45	3.93	1.71	0.60	24,360	None	3,070	1,130	28,560													

	Per cent.		Blast Press., Sq. In.	Time of Blow, Silica		In Minutes, Total Blow	Average Analysis of Slags							
	Si in Charge			Blow			Converter Slag		Ladle Slag					
							Silica	Iron	Mang.	Silica	Iron	Mang.		
No. 1. Without scale.....	1.48	23	4.70	11.00	No. 1 Without scale.....	61.60	16.50	9.18	56.60	16.60	11.92			
With scale.....	1.48	23	2.50	9.00	With scale.....	56.20	20.30	9.66	57.00	16.05	12.52			
No. 2 Without scale.....	1.52	23	4.55	11.02	NOTE.—The average weight of all slags was 2,270 lb. per heat or 8 per cent. of total charge.									
With scale.....	1.57	23	2.50	8.80										
No. 3 Without scale.....	1.49	23	4.62	10.55										
With scale.....	1.50	23	2.50	8.40										

It is apparent from a study of the above table that the tonnage is increased approximately 20 per cent. when roll scale is used, due to the shortening of the blowing time and that the results in general agree with those shown by Messrs. Patton and Speller.

ducted as to the results obtained by its use, and our conclusions have been practically the same as those of Messrs. Patton and Speller.

We have followed through to the finished product, heats, the temperature of which has been controlled by the use of roll scale alone, roll scale and pig-iron scrap, roll scale and steel scrap, and heats in which the temperature was controlled by use of steam blown through the molten metal, and in every case, all conditions being equal, those made with roll scale and scrap (either pig iron or steel) have given the best results.

The use of roll scale, as the authors have stated, must be a judicious one. It has been the writer's experience that the main determining factors in its use, in so far as it relates to the Bessemer process, are as follows:

- (a) The silicon content of the pig iron to be blown.
- (b) The combined and free carbon content of the iron.
- (c) The manganese content of the iron.
- (d) The initial heat of the iron to be blown.

We have found that iron with a silicon content under 1.15 per cent. does not warrant the use of roll scale and that it does not follow that the higher the silicon content the greater the amount of roll scale that may be used; for example, the writer has seen pig iron with 3 per cent. silicon with which no roll scale at all could be used. This I believe to be due to what the practical man calls "dead iron," or, from a theoretical standpoint, iron which is low in initial temperature, low in combined carbon and undoubtedly carries a large amount of occluded gases.

The use of roll scale with iron containing over 1 per cent. of manganese and the normal amount of silicon causes a sloppy blow and, quite often, steel which is not uniform. Increased silicon content of the iron tends to offset the effects of high manganese.

Table 1 shows the average results obtained from various tests made at the Youngstown Sheet and Tube Co.

H. H. CAMPBELL, Steelton, Pa. (written discussion).—About the year 1900, the Bessemer department at Steelton was short of scrap, and we tried using large, hard, compact lumps of ore in the converter. These, were thrown in immediately after blowing began and the heavy lumps, plunged down into the molten iron. It was to be expected that they would disintegrate under sudden heating, but this same ore when thrown into the bath of an open-hearth furnace did not vigorously decrepitate, so that no trouble was looked for from this source. There was no outward sign of loss at the nose of the vessel, but the deposit in the stack showed plainly that some of the ore was being projected out of the converter in particles about the size of a small pea. It is not possible to make even a guess at the loss from this cause, or to give any figure regarding the gain from the reduction of the iron in the ore, because the amounts involved were so small.

The charge of pig iron was about 22,000 lb. while the average ore addition was about 400 lb. This ore carried about 68 per cent. of iron, so that the metal in the ore represented only 1.2 per cent. of the weight of the pig iron. If one-quarter of the ore had been lost, the weight of steel produced would have been only 0.3 per cent. short, and we can not be sure of such accurate weighing in a small number of Bessemer heats. The slag from the vessel was normal, showing that all the ore that stayed in the converter was reduced; but it brings neither pleasure nor profit to put good ore in the vessel and then blow it up the stack.

When these experiments were made, the pig iron contained only a moderate proportion of silicon and a normal heat required about 1,000 lb. of steel scrap. At the same moment an alternate heat blown in another vessel would be satisfied with 400 lb. of ore. According to the authors of this paper, about 650 lb. should have been necessary to balance 1,000 lb. of pig iron; but there is a chance for quite an error of judgment on the part of both of us. The thermal calculation involved will be left for others to investigate.

It must never be forgotten that the scrap used in a Bessemer vessel represents a profit, for it is converted into steel without a cent of expense, except the cost of the recarburizer. Ore will furnish a little cheap metal; but 400 lb. of ore do not contain over 270 lb. of iron, and if a good part of that is blown up the stack, and if this ore addition makes it impossible to convert 1,000 lb. of scrap into steel at no cost at all, then it would seem that the argument is not all onesided. The experiments just cited were not pushed, and the writer did not give much personal attention to the matter. He merely suggested to the superintendent of the Bessemer department that ore should be tried, and it was used a few days on one converter. The experiment was not a failure, but the Bessemer superintendent reported that there was no particular advantage gained and so the story ended.

M. R. STEVENSON, McKeesport, Pa. (written discussion).—I am heartily in accord with the facts that have been so ably presented in this paper and from the standpoint of the furnace operator can assure you that the high-silicon iron furnished to the Bessemer Department did not work for bad practice in the blast-furnace operation. Although it may require a very slightly larger quantity of coke to produce this higher-silicon iron as compared with the lower-silicon, yet the results obtained in all the succeeding departments, ending with the finished product at the tube and pipe mills, show a fairly large saving over the practice obtained with the use of lower-silicon iron.

HENRY D. HIBBARD, Plainfield, N. J.—This paper deals with an advance in Bessemer steel practice, an art which was assumed by most steel metallurgists to have reached its highest attainable development.

The three features:

1. Of using iron oxide instead of a part of the far more expensive crude iron of the charge;

2. Of the greater cooling effect of that oxide as compared with scrap, thus lessening the amount of scrap needed or permitting higher silicon in the crude iron;

3. Of shortening the time of blowing so as to get a greater output; are each of them important and worthy to be considered a distinct step forward in the art.

The first feature owes its value in part to the supplying of oxide of iron for the requirements of the slag instead of causing metallic iron to be oxidized for that purpose.

The second feature, greater cooling effect, comes from the absence of heat ordinarily generated by the oxidation of some of the iron of the charge and to the absorption of heat in the reduction of metallic iron from its oxide. This latter takes place to a small extent through the reducing power of the silicon while that element is still plentiful in the charge.

The shortening of the blow comes from the presence in the vessel at the start of available oxygen in the roll scale which lessens the quantity of air that it is necessary to blow in to complete the oxidizing processes.

When the basic process was being worked out about 1877, Bell tried in England the effect of adding iron ore to an acid-lined converter to see if he could maintain a sufficiently basic slag to eliminate phosphorus from the iron. He added so much ore that the ganister lining of the vessel was badly scorified, and an excessive quantity of slag was made which contained so much silicic acid that no phosphorus was taken from the metal. The experiment failed and no further trials were made on that line.

The calorific effects of oxidizing the commonly determined elements in crude iron by magnetic oxide of iron or roll scale at 1,350° C., which may be taken as the temperature of the charge at the beginning of the blow, are given approximately in Table 1.

TABLE 1

Element	1 Kg. Oxidized to	Heat of Combustion, Calories	Heat Consumed, Calories	Heat Lost in Gases, Calories	Surplus, Calories	Deficit, Calories
Silicon.....	SiO ₂	7,595	4,836		2,759	
Phosphorus.....	P ₂ O ₅	5,892	5,642		250	
Manganese.....	MnO	1,653	1,612		41	
Carbon.....	CO	2,430	5,674	1,169		4,413
Carbon.....	CO ₂	8,100	11,348	3,270		6,518

One kilogram of iron burned by oxygen to Fe₂O₃ gives 1,612 Cal. The "heat consumed" column gives the heat spent in reducing enough of the Fe₂O₃ to give the quantity of oxygen required for each reaction considered.

In the Bessemer works at Steelton the cooling effect of ore in the converter was years ago found to be three times as great as that of an equal weight of scrap, which is about the proportion found by calculations. That is 100 kg. of ore absorbs about 105,000 Cal., while 100 kg. of scrap absorbs 37,000 Cal. in melting.

The table explains why the silicon is the chief heat producer of the acid process and why its heating power is decreased to little more than a third when it gets its oxygen from Fe_2O_3 instead of from air. It also shows the great heat absorption in the reaction of carbon on Fe_2O_3 by which metallic iron is reduced.

The benefit from the lessened cost per ton of ingots is, in times of great demand, like the present, overshadowed by the profit on the increased production. For a dime saved in costs a dollar is made in profits.

C. S. ROBINSON, Youngstown, Ohio.—Mr. McCleary's paper is not conducive to much discussion, but it is a very pleasant thing to be able to so thoroughly corroborate the main points brought out in the paper of Messrs. Patton and Speller. Referring to what Mr. Stevenson has said, it seems to me that the range permitted in the analysis of pig iron is such that it enables the blast-furnace man to make more pig iron, suitable for the purpose, and also that he makes a greater amount of a good grade on a more acid slag at a somewhat less cost. He has to carry less lime and he gets into fewer difficulties in his operations, thus working to a more uniform steel due largely to the use of scale; and this is reflected in the welding properties of the skelp and the general quality of the pipe.

J. W. RICHARDS, South Bethlehem, Pa.—It is extremely interesting to hear the details of this application of the open-hearth ore reaction in the Bessemer converter. It undoubtedly functions by limiting the amount of air which is necessary to oxidize the iron, and thus limits the length of the blow, and the heat losses by radiation during the blow. There seems to be, however, a contradiction which I hope will be solved by a study of the conditions. The reaction of the scale upon the iron must undoubtedly be a reduction of the scale by the silicon in the iron. Whether the scale is reduced to ferrous-oxide or to metallic iron, its reduction by silicon is an exothermic reaction, against which you have only the heat necessary to raise the scale to the reacting temperature which is a much smaller amount of heat than is evolved. I cannot, therefore, imagine why the use of the scale should have one and a half to three times the *chilling* effect of putting in cold scrap. There is some contradiction here which needs to be cleared up by a further study. Then there is the further great saving in the heat otherwise carried out by the gases. The heat lost by the nitrogen of the air, which would ordinarily do the oxidation, is entirely suppressed and that is an enormous factor. Therefore, I think that when we get this heat balance straightened out, it

will be found that the use of the scale is not in itself a chilling reaction. I think it must be an exothermic reaction, the silicon of the iron being the reducing agent.

A. PATTON.—Prof. Richards questioned the loss, or the difference between that which goes into the bath as Fe, and that which is lost as an oxide or combined iron in slag; that is, a part of scale used is not accounted for. As Mr. Speller has told you, the combined iron in slag is just about the same, regardless of whether scale is used or not. Now if we use, say, $1\frac{1}{2}$ per cent. of charge (metallic) scale, that probably will take care of the combined iron in slag. If 3 per cent. is used, then there must be $1\frac{1}{2}$ per cent., or half of your scale used, that is taken care of and reduced to Fe. We do not know, of course, whether the scale goes with the slag as combined iron and takes the place of what we would make to satisfy this slag if the scale was not used, but one will offset the other, as we find that the iron in the slag is just about the same when the scale is not used as when it is used, so in this way you can find that we recover practically all the iron in the roll scale as Fe.

J. W. RICHARDS.—In that case, I must object to the statement of the chilling effect of using the scale, because the heat reactions show that the reduction of the scale to metallic iron by silicon and the formation of the silicate slag provides considerably more heat than would be necessary to raise the scale to the reacting temperature. There is some contradiction here which needs to be straightened out by more information.

The Significance of Manganese in American Steel Metallurgy

Discussion of the paper of F. H. WILLCOX, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 199 to 207.

THE CHAIRMAN (HENRY D. HIBBARD, Plainfield, N. J.).—This paper is timely because of the changed conditions due to the great war, but apparently its scope is limited to oxidation process steels. Referring to the four numbered observations on page 199:

No. 1 seems to ignore Johnson's discoveries about oxygen in cast iron.

No. 2 ignores crucible steel.

No. 3 ignores red short steel containing enough gas solvents to make it solid, while

No. 4 assumes that a deoxidizer is necessarily a gas decomposer or solvent and *vice versa*, which it not always is.

From these observations it is deduced that the only hole-forming gas in steel is an oxide of carbon (CO presumably is meant), which is not true.

The assumption on page 200 that the more boiling of steel (presumably in the molds) the more segregation, is hardly warranted, as it would seem that agitation should tend to check segregation rather than promote it.

It is true that marked segregation exists in a large proportion of ingots containing gas holes, if not in all, but the steel may have been so dead that the top froze over quickly without any escape of gas bubbles or boiling after turning was finished.

Referring to the paragraph on manganese, page 200, its use in steel making by an oxidation process may be considered and amended as follows:

1. Its use as a deoxidizer.
2. Its use to give the composition desired either for the purpose of meeting specifications or to give certain physical properties.
3. To prevent gas holes in some degree.

Use No. 1 requires time because the oxygen or oxides are so dilute, and because the manganimiferous products formed, oxides and silicates (sonims), require time to precipitate, separate, agglomerate and float up to merge with the slag.

Nos. 2 and 3 require only time enough for an intimate mixture of the manganese alloy.

The fundamental reason why manganese is not wholly replaceable in making steels of the class under consideration seems to lie in the fact that its oxide forms with silica and iron oxide very fusible silicates which wet each other when they touch and so collect together, and when large enough float to the surface.

Other elements such as silicon, aluminum, calcium, and magnesium, have stronger affinity for oxygen than manganese has, but their oxides are insoluble and infusible in the molten steel and when these elements are used for deoxidizing without manganese the steel is likely to be red-short because of the widespread diffusion of the sonims which cannot, or do not, escape.

As for aluminum, the proportion the author calls very small, about 0.05 per cent., would better be called very large, amounting as it does to about 18 oz. per ton. Three ounces to the ton is common in moderately well-made open-hearth steel and 6 oz. with about 0.20 per cent. of silicon will kill any ordinary ingot steel. If it does not the furnace practice is not good. Half an ounce per ton will often determine whether or not a well-made steel will "rise" or "stand" in the ingot molds.

The present scarcity of manganese and manganese ore in Germany has, as the author states, been met by the utilization of blast-furnace slag accumulations of the past. These slags are smelted in electric furnaces, the alloy produced at one plant analyzing about 60 per cent. Mn, 20 per cent. Si and 2 per cent. C. The phosphorus, of course, is

very low, as all that originally in the ore went into the metal on the first smelting.

Economy of manganese is now demanded. In the first place, let it be realized that freedom from redshortness is due not to the presence of manganese but to the absence of impurities, both soluble and insoluble, which cause it. Steel made from wrought iron and charcoal or blister bar, would seldom, if ever, have any manganese, yet such steel was regularly made for half a century before Heath used manganese ore in the crucible.

Acid open-hearth steel may have much lower manganese than is ever specified and be free from redshortness.

In the first practice I personally knew of following Martin's first plan, a bath of pig and scrap was melted and then diluted with charcoal direct-process blooms made in the Catalan Forge and in 3 years, during which I knew about it, no steel showed any redshort tendency. We heard of redshort steel elsewhere but otherwise knew nothing about it. Our soft steel for boiler plate contained from 0.2 to 0.3 per cent. of manganese. The slags were extremely vitious, apparently without any uncombined iron oxide and the continued cleansing effect with such a slag, due to time, with some manganese in the bath, and the stirring caused by working the blooms (which were preheated) in the bath practically eliminated the oxides of every sort which, if present, would have tended to make the steel redshort.

The content of manganese is still incorporated in some specifications for structural steels, but it is doubtful if those who do it can give any proper reason therefor. They apparently do it because their predecessors did. Much good steel has been rejected because the manganese was too low, when a little more haste in casting would have kept it within the specification by retaining a part of the manganese which was, as it ought to have been, consumed in reducing iron oxide, making the steel so much the better.

When steel to be tempered requires low manganese, no objection is found against so specifying, but superior or varied furnace practice by which steels may be made free from redshortness and low in manganese is barred or manganese is wasted.

I may say also that we have read of a manganese substitute in Germany which I understand to be carbide of calcium. They cut the manganese down to about half of what was previously used, and by the use of carbide of calcium and that amount of ferro-manganese, made of blast-furnace slag, they are able to make fairly good steels, but the control is not as good as it used to be when the supply of manganese was not cut short.

J. S. UNGER, Pittsburgh, Pa.—When we speak of a metal, alloy or com-

pound as a deoxidizer in steel manufacture, the term is a relative one only, as oxidation or deoxidation is largely a question of temperature.

Tin and lead resist oxidation at ordinary temperatures much better than iron, but when the first two are raised to their melting points they are quickly covered by their oxides, while iron is but slightly affected at the same temperature.

Metallic silicon changes very little in the air; at a dull red it oxidizes fairly rapidly to silica, but at a bright red the oxidation is quite slow.

Steel is melted and to a large extent oxidized in a stream of oxygen after the reaction has been started, while 80 per cent. ferro-manganese burns very slowly under the same conditions, even through the manganese is supposed to have a much stronger affinity for oxygen.

A bar of copper and steel may be bolted together, and the steel completely burned away from the copper by a stream of oxygen with very little injury to the copper.

Oxygen is found in practically all steels. If it exists as an oxide or as a gas combined with carbon, it is very difficult to understand how molten steel or a piece of steel heated to a bright red heat in a vacuum will show some free oxygen in the gases evolved.

Cast iron with a high percentage of carbon, silicon and manganese, all powerful deoxidizers, contains oxygen. In fact certain irons are improved by blowing a little air through the molten iron.

The determination of oxygen in steel is a very difficult operation. There is a grave doubt whether any particular method can be considered an exact quantitative one, but enough work has been done to show that oxygen may exist as oxides, as gases, carbon monoxide, carbon dioxide and as free oxygen.

Any investigation of the effects of what are usually called deoxidizers is likely to lead to confusion. It is impossible to say when a deoxidizing agent ceases to act as a deoxidizer and begins to confer certain physical properties on a steel.

Manganese may be beneficial in conferring hot or cold working properties, and the quantity is usually specified, but steels are made in large quantities every day which do not contain manganese. Again, manganese affects the qualities when added above the usual amount. A well known example is manganese steel.

Silicon, another well known deoxidizer, is found in nearly all steels. When added in fairly large proportions, as in silicon tool steel, silico-manganese steel, or in certain steels of special magnetic properties, it completely changes the character of the steel as the manganese did in the preceding example.

Such examples can be multiplied. All elements which can be alloyed with steel confer some property.

Very few steels are made by the addition of manganese alone. The

three common deoxidizers, manganese, silicon and aluminum, are nearly always used in ordinary steel making. Sometimes but two are used, very rarely one alone. It would appear that each is necessary and has its function when applied at the proper time and place.

Experience has shown that manganese and silicon are best added to the bath or ladle, and aluminum to the mold. It is of interest to know that in some experiments made by using an alloy of these three metals and iron, the effect was to make a better product than when the constituents were added alone at different stages of the heat.

Many elements have been tried as substitutes for the three mentioned. Magnesium has been proposed as a substitute for aluminum, as it has a stronger affinity for oxygen. A simple calculation will show that 1 lb. of magnesium will take up 0.66 lb. of oxygen, while 1 lb. of aluminum will take up 0.89 lb. of oxygen.

When the magnesium in sticks $\frac{3}{4}$ in. square was added to the mold, it resulted in an explosion, blowing part of the steel out of the mold, making it a dangerous operation. These disadvantages, together with an extra cost of 100 per cent. over aluminum (before the war prices), condemned its use.

Calcium, sodium, titanium, vanadium and boron alloys have been tried in various combinations. My experience with those I have tried is as follows:

1. Their cost is too high to permit of ordinary commercial use.
2. When used alone, the steel is not of good quality.
3. To show favorable results they must be assisted by one or more of the common deoxidizers.

Slight honey-combing or sponginess, which must not be confused with piping, is rarely injurious, as these small cavities if unoxidized, unite into a common mass if the steel is rolled or forged. Their presence can not be detected by physical tests, nor does the finished material give poorer service when put into actual use.

A very minute quantity of gas may produce a cavity $\frac{1}{4}$ in. in diameter. An ingot containing about 0.001 per cent. of oxygen, existing as carbon monoxide, will at the freezing temperature of the steel occupy a volume equal to the volume of the ingot.

On page 170 of the excellent work by H. M. Boylston, referred to by Mr. Willcox at the close of his paper, is a table in which the relative value of the deoxidizers used is ranked. By adding the ranking of each deoxidizer and making a comparison, the results are as 3 is to 4 in the extreme cases. While I do not agree in using the mechanical tests as a measure of the quality of the deoxidizer, owing to the effect such material has on the steel itself, the results show very little difference.

To my mind, a good deoxidizer must have these qualities:

1. For the work it will do, it must be cheaper than any other deoxidizer.
2. It must alloy with iron in any proportion.
3. Adding a small excess should not influence the quality of the steel; or perhaps I can make it clearer by saying that an overdose does no particular harm.
4. It must confer better hot working qualities.
5. The resulting product of the reaction must be easily fusible and of much less weight than the steel, in order that a prompt separation may take place.

Piping, segregation, soundness, specific gravity, etc., are secondary qualities, and can be almost entirely removed by cropping. No single one of the common deoxidizers possesses all the qualities specified above, but a careful use of them at the proper time is not alone cheaper, but has given better results than any materials tried thus far.

J. W. RICHARDS, South Bethlehem, Pa.—The author's remarks on the use of deoxidizers in steel are brief, and in many cases faulty, perhaps due to their too great brevity. I wish particularly to call attention to the remarks on the use of aluminium in steel, where it is said that the heat of reaction is appreciable and possibly renders the steel more fluid momentarily. In actual practice the amount of aluminium used in steel is not sufficient, by its oxidation, to raise the temperature of the steel 1°, so that the increased fluidity is due to other causes than the increase of temperature; it is due to the removal of the ferrous oxide, which reduces the fluidity. He also says that aluminium seems to increase the solvent power of the steel for gases, and thus its use prevents blowholes to a marked degree. I believe that its use in preventing blowholes is due to an entirely different principle than that of increasing the solubility of the gases in the steel. At least nine-tenths or more of its action is due to removing the oxygen of the dissolved ferrous oxide.

Coming to the significance of manganese and our supply of it, there is a remark on page 204 which may lead us in this country to think of how our supply of manganese for ferro-manganese can be increased. It is the statement that during the past year piles of slag from an old ferro-manganese furnace, running 5 or 10 per cent. manganese, have been drawn upon and reduced for making ferro-manganese. The slags from ferro-manganese, while high in manganese, are very low in iron, and therefore they are a sort of concentrate produced by smelting which gives a material from which you can get a higher per cent. of ferro-manganese than from the original ore from which the slags were made. In this country we have very few supplies of rich manganese ores, and if our supply were cut off during a war, we would be at a great loss for material from which to make directly high grade ferro-manganese. However, it seems to me much more logical and quite possible to take the leaner manganese ores which contain considerable iron, and to give them a double treatment, first extracting almost all the iron with part of the manganese, leaving a rich manganiferous slag, making perhaps very little

attempt to get much of the manganese out, in fact, taking as little as possible. Then re-treat the slags so as to get from them a much higher percentage of ferro-manganese in the product than would otherwise be obtained. There are immense deposits of manganese ores, such as the Cuyuna Range in the Middle West, which could be utilized for that purpose and which would supply all the manganese for high ferro-manganese that the country would need. It is even possible that they could be thus utilized at other times than in case of war and the suspension of our imports of manganese ores.

ALBERT SAUVEUR, Cambridge, Mass.—Dr. Unger made the statement that blowing air through cast iron improves its quality; I think that some of us would like to know whether this is Dr. Unger's expression of his own opinion or merely the expression of the opinion of others?

J. S. UNGER.—That is simply a statement of another man's opinion.

JOSEPH W. RICHARDS.—I have read carefully Mr. Johnson's paper on that subject, and I do not agree with him on the general proposition; but I think it is quite possible that a gentle blowing of air through melted iron may improve the quality of the iron, not by leaving oxygen in the iron but by removing some of the deleterious impurities by differential oxidation.

A. E. OUTERBRIDGE, JR., Philadelphia, Pa.—In the last month there has come to me a keg of ore from a very minute island in the Caribbean Sea that is occupied by a planter with a few negroes who cultivate the land upon the shore, and there is a great mountain that rises up like a peak. There have been rumors for quite a number of years, that this mountain was a solid mountain of manganese. The planter sent a keg of the ore to New York, to the shippers; and they asked me if I would look after it. The ore was shipped directly to Booth, Garrett & Blair, very old chemists in Philadelphia. As I remember, it is an ore, not the pure pyrolusite or kidney manganese ore at all, but contains 30 per cent. of manganese, only 2 per cent. of iron and, I think, about 20 per cent. of silica. There is no evidence whatever that that ore was selected with any intelligent supervision. I have done nothing with it and I am not in position to do anything further than to send it back, which I have done, to the shippers in New York. If this matter interests anybody, I will be glad to give them the benefit of what information I have about it.

LEONARD WALDO, New York, N. Y.—Many years ago, when I was engineer in charge of the Cowles Electric Smelting & Aluminum Co. at Lockport, at the beginning of the aluminum industry, the inertia of the market had to be overcome in the introduction of aluminum. We searched assiduously in every direction and finally found an outlet for aluminum in the manufacture of steel. A pamphlet was published at

the time, but we found that if we attempted to introduce aluminum into steel, that the logical way and the only way was through the use of an alloy of aluminum with iron. A standard alloy was made, and the first introduction of aluminum into steel was made in that way. Later, the use of pure aluminum itself has taken place. I speak of that now because, at a hint from Baron Von Jüptner, I suggested, some years ago, the use of magnesium as a steel deoxidizer and its attempted use was followed by precisely the results which Mr. Unger has described, the trouble with magnesium being that under open air conditions it burns before it will vaporize. The rapid production of its own vapor causes an explosion, and therefore it becomes dangerous to use in the ordinary way, but there are certain theoretical advantages in the use of magnesium which necessitate a continued study of that subject which is now being made, and I very much hope that we will be able to add magnesium to steel in such a way that the full effect of its light specific gravity and its qualities as a deoxidizer can be brought fully into play. I say that now because magnesium is just commercially coming into the world, and various forces have contributed to make it an important material at this time. Several companies in the United States have started its manufacture. The deposits of magnesite are being carefully examined and chemical by-products from which metallic magnesium can be made have been carefully searched for. To show you what progress has been made, I would like to give an analysis of present commercial magnesium which is sure to find its use in deoxidizing. This is a sample from a lot of 22,000 lb. of magnesium.

	Per Cent.
Magnesium.....	99.867
Silicon.....	0.032
Iron.....	0.037
Aluminum.....	Trace
Copper.....	0.028
Cadmium.....	None
Bismuth.....	None
Lead.....	None
Nickel.....	None
Cobalt.....	None
Zinc.....	0.036
Specific gravity.....	1.74

It appears, therefore, that magnesium has arrived at the first stage; it is a commercial product, it is not too high in price, and, with our better knowledge of its use, it is sure to be a deoxidizer of value, not only in the ferrous metals, but in the non-ferrous metals. I might say that some experiments which were made without much thought, not by me but by a colleague in steel making, in the attempt to introduce it into crucible steel, were very disastrous. Men were hurt, but the reasons for the ex-

plosion were well known, afterward at least, and I do not apprehend that the difficulty is a serious one of getting the magnesium into the molten steel. The other uses of magnesium in aviation and the problem of the military engineer are so great that whatever encouragement we can give it from the iron and steel side should be given.

RICHARD MOLDENKE, Watchung, N. J.—I would like to add that I also have made tests with metallic magnesium added to molten cast iron. This on the advice of Baron Jueptner von Jorstorff, then in Budapest, in 1900. I had the same experience that Dr. Unger had, when I introduced a piece of rod magnesium inserted in the end of a wooden bot-stick. The metal blew up, but enough was saved to note a wonderful fluidity in what was almost pasty and badly oxidized material before. Moreover, when cast into chills, the metal exhibited a clearness of crystallization that was remarkable.

In sending out the material for analysis, to see whether magnesium had remained—as I wished to make a ferro-magnesium for subsequent use in de-oxidation experiments—it was unfortunately lost. The price of magnesium then went up and I never took the subject up again, my belief being anyhow that it is far better to keep oxygen out of cast iron than to remove it after poor melting practice has introduced it.

J. W. RICHARDS.—That simply moves the problem one step further; how are you going to make the iron magnesium alloy? But I think that problem will be solved, and when it is solved, it is quite likely that magnesium will come in as a fourth member to supplement manganese-silicon aluminium in the final deoxidation of steel.

I wish to confirm Mr. Hewett's statement about the manganese deposits of Costa Rica. A former student of Lehigh University, Mr. Yglesias, is president of a company that is mining the manganese ores, and I have had some correspondence with him. He has a high-grade ore which can be delivered in shipload lots on the Atlantic Seaboard. The problem of getting our manganese ore from abroad, in ships, would be one for peace, but in time of war we would have to fall back on our own supplies of manganese, which would bring up the importance of the other low-grade deposits in this country.

Notes on the Heat Treatment of High-Speed Steel Tools

Discussion of the paper of A. E. BELLIS and T. W. HARDY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 61 to 68.

THE CHAIRMAN (ALBERT SAUVEUR, Cambridge, Mass.).—Any information likely to throw light on the constitution and proper treatment of high-speed steel in order to obtain maximum results, should surely be welcomed. All users of high-speed steel, I believe, will agree with the authors that, in order to obtain satisfactory results, it is not sufficient to heat all brands of high-speed steel to one and the same temperature; apparently very slight differences of chemical composition seem to call for marked differences in temperature. I think that all of us agree that a properly treated piece of high-speed steel should have a fine austenitic or polyhedral structure and that an appreciable quantity of free carbide may be present in the case of very high tungsten steels. What we seem to lack is a satisfactory method, quick and reliable, of ascertaining when the proper structure has been obtained. I think Mr. Bellis' paper is a step in the right direction. To the untrained eye, however, it is not an easy matter to select from some of his photomicrographs the one representing the best structure; for instance, in the case of steel A, we wonder why temperatures of 2,250 and 2,300 do not yield the same results, seeing that both structures are apparently alike, and the same might apply to some of the other samples. I have no doubt, however, that the experienced observer is capable of detecting structural differences that are not made plain here and by which he is able to select the proper quenching temperature.

COLIN G. FINK, East Orange, N. J.—On page 68 the authors say that "the use of a barium chloride bath to eliminate this difficulty has the disadvantage that the surface of the tool becomes *decarbonized*." I would like to ask whether they actually found that the surface was decarbonized, or whether they base this conclusion on softening alone?

A. E. BELLIS, Springfield, Mass.—No, we have never demonstrated that it was a decarbonization, it is just the softening of the steel. The addition of ferrocyanide to the ferrochloride bath has been suggested. I think that is not altogether satisfactory, because the ferrocyanide bath changes composition very rapidly at these high temperatures, and it would be just a question of luck whether or not it had too much or too little ferrocyanide in the bath giving equilibrium with the carbon content.

COLIN G. FINK.—I would suggest that the authors take a piece of tungsten and put it into the barium chloride bath, weighing it before and after they put it in. If they want to use a barium chloride bath for high-speed steel, it is best to saturate the bath with tungsten; they will then find that the steel is not "decarbonized."

J. A. MATHEWS, Syracuse, N. Y.—There is no doubt of the practical bearing and importance of this paper to users of high-speed steel. It is perhaps slow and expensive, making a series of photomicrographs of each steel you get in the shop, but a visual examination would be sufficient after a complete series of standard photographs had been made.

A. E. BELLIS.—It is not necessary to make photographs.

J. A. MATHEWS.—I did not get clearly which structure you consider the best one, or do you have to sacrifice sometimes the maximum cutting efficiency to preserve your clean surface?

A. E. BELLIS.—I will repeat that the sample, if not heated hot enough, will show the black background with the excess carbide spots after the 10 min. etching. The best structure is the finest austenitic network with the excess carbide. As the sample is overheated, a coarser and coarser network results; as in orthodox metallography, the "grain" is coarsened by overheating. If overheated very much, a very coarse network results and the carbide spots become blackened. The best treatment, according to our experience—it should be modified with different forms of tools—is that which gives the smallest network structure.

I would like to remark that we can take, say, four samples from each bar of, say, six bars of steel, and have them sawn, polished and viewed under the microscope in an hour or so; after one is experienced and has the laboratory boy do the polishing and etching, the actual work is not very great. I think the greatest advantage of the test is its simplicity.

J. A. MATHEWS.—In connection with lathe tools, it has generally been considered the best practice to bring them to a sweating heat, and that is far beyond the point at which you get your minimum grain size. Is that on the assumption that you grind away most of the metal where fusion has started in?

A. E. BELLIS.—No, there seem to me to be two problems: One is obtaining the best structure, and the other is to maintain the surface.

H. M. BOYLSTON, Cambridge, Mass.—I would like to say that I have examined some of Mr. Bellis' specimens. The structures under the microscope are clearer than his photographs show. It is much easier to judge from the actual specimens than it is from a set of photographs such as he has shown.

MARSHALL H. MEDWEDEFF, Springfield, Mass. (written discussion).—The subject of heat treatment of high-speed steel tools is of exceeding importance to the large users of such steels, and any knowledge brought to light as to their successful treatment would be of help in unraveling the difficulties that arise in the art. Any discussion on this subject can only be with the hope of bringing such to light.

It is difficult at the present state of our knowledge to establish a standard practice in hardening tools made from such steels. It is apparently true that every "brand" of high-speed steel has a definite maximum as well as minimum hardening temperature which will impart to it the desired qualities. At the same time, a consideration of the design of the tool, its size, and one may add the function it is to perform, are of utmost importance in gaging the temperature from which it is to be quenched. Thus, while a given brand "A" for a roughing tool or a large milling cutter will give excellent results when quenched from its maximum temperature, say 2,300° F., a form cutter of 1/4-in. diameter, will be ruined by so high a heat; 2,150° F. would probably be a safe temperature, at the same time imparting to it the desired cutting qualities. The reasons for it one can perhaps attach to the very design of the cutter. In all probability the cutting edges, being relatively fine, heat up very quickly and by the time the main body of the tool is heated up to the same temperature, the edges develop a coarse structure and brittleness results. The higher the temperature, the greater the danger of overheating the edges. In a large cutter, the cutting edges, being comparatively large, are not exposed to the same danger.

The successful hardening of high-speed steel tools will depend to a great extent upon the proper manipulation in the furnace and also on the manner of quenching, *i.e.*, the position of the particular tool as it is immersed in the quenching medium, and at times upon the quenching medium itself. An ordinary cutter is not difficult to heat-treat, but it is the intricate form cutters that tax the mettle of the hardener. The writer has had occasion to examine a considerable number of cutters failing in service; some with their cutting edges either broken or worn off. In the former case the failure might be due either to overheating with its consequent coarse grain causing brittleness, or to underheating, where the brittleness is due to the excessive free carbides which were not given the opportunity to diffuse. Where the edges are worn off, the cause can easily be traced to surface decarbonization. But the hardening room is not always responsible for failure of tools. Careless adjustment in the machine, subjecting it to undue stress or careless grinding will frequently ruin a good cutter.

Uniform heating is of utmost importance, and the hardener overcomes the defects in the design of the furnace by manipulating the object heated with his tongues, so that no part be overheated. Surface decar-

bonization is largely prevented by properly preheating to about 1,500° F. and then placing in the high-temperature furnace, as the absorption of heat after the above temperature is reached is very rapid. The tool should be taken out of the high-temperature furnace as soon as the desired temperature is recorded.

The writer at one time started an investigation to determine the combined effect of time and temperature in bringing out the desired structure in high-speed steels. A number of test pieces, $\frac{1}{4}$ in. in diameter, after being preheated in an electric furnace were placed in a high-temperature furnace heated to 2,100° F. One of these was quenched as soon as it reached the furnace temperature. Three others were held for 1, 2 and 3 min. respectively after reaching the furnace temperature and then quenched. No conclusions are drawn for the present, but it appears as if a lower temperature and a longer time in the furnace produce the same results structurally as on heating to a higher temperature and allowing little or no soaking time in the furnace as soon as that temperature is reached. Thus, a sample held in the furnace for 2 min. at 2,150° F. and quenched in oil seems to have the same structure microscopically as one heated to 2,250° F. and quenched at once. From theoretical considerations this might well be expected. The object of heating such steels to high temperatures is to diffuse the carbides in the austenitic matrix, producing the well-defined network structure. The longer the metal is left in the furnace, the greater opportunity there would seem for this diffusion to take place, and the characteristic structure to appear. As the structure under these conditions appears at a lower temperature, it will necessarily be finer. But it is evident that the above procedure can only be followed on such tools as allow for regrinding after hardening, as the dangers of decarbonization are considerably increased.

The writer hoped to be able to carry out determinations of the critical temperatures of a number of high-speed steels, but other duties prevented their accomplishment. It is his conjecture that at the temperatures at which diffusion of the carbides takes place there must be a retardation in the temperature analogous to the retardation in the melting of solids. M. Yatsevitz, in Prof. Sauveur's laboratory, made a number of determinations of the critical temperatures of such steels, but his working temperatures were much too low, not exceeding 1,050° C. or 1,921° F. At those temperatures the structure of these steels do not appear to assume any definite form, being largely cementitic. The above remarks refer to the regular high-speed steels of high-tungsten content.

A few words about pack hardening high-speed tools. In the writer's opinion, pack hardening of tools made of such steels should be resorted to only in case of very delicate cutters which cannot be exposed to the hazards of the high-temperature furnace. The disadvantage of pack hardening is the danger of carbonization, unless the packing material be

noncarbonizing, such as charcoal ashes. The practice of using a mixture of burned leather and charcoal is not advisable, as this in itself is an excellent carbonizing material. Charcoal is used successfully, but the temperature must not be over 1,900° F., as at higher temperature carbon is evidently absorbed. A carbonized pack-hardened cutter behaves like an overheated carbon steel, becoming brittle. The writer has had occasion to examine a number of such failures. Upon regrinding and hardening in the usual way they usually give excellent results.

Effect of Time in Reheating Hardened Steel below the Critical Range

Discussion of the paper of CARLE R. HAYWARD and S. S. RAYMOND, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 277 to 285.

CARLE R. HAYWARD.—I do not want it understood that I think that the conclusion that the time of tempering temperature is immaterial has been definitely proven, but since these are the first definite figures put out, I thought before continuing on more work ourselves, it would be well to present them as progress and for information.

J. A. MATHEWS, Syracuse, N. Y.—I think that if Prof. Hayward would consult the paper of Profs. Barus and Strouhal,¹ 1885, he would find about 250 pages, mostly figures, on this subject, in which they investigated the tempering effect by electrical and magnetic methods. There are so many figures in the paper that it is very difficult reading. One point particularly brought out was that even at a low temperature, such as the boiling point of alcohol, about 60° C., there is a tempering effect, increasing up to 4 hr. One reason they obtained those results, with which the present authors are at variance, was that they started with a high-carbon steel which hardens thoroughly, whereas the steel Prof. Hayward and his collaborator used was a comparatively mild steel which does not take a great deal of hardening. About 3 years ago Mr. Stagg and I published² a few figures bearing on the same subject, but we used a low-carbon alloy steel of high tempering qualities and found a great deal of improvement, particularly in ductility, by continuing the drawing up to 4 hr. We used a chrome-vanadium steel showing after heat treatment 250,000 tensile strength. On short drawing, we got almost no ductility, but by continuing to 4 hr., we raised it to 12 per cent. I believe Prof. Hayward's results are due to using comparatively mild steel, whereas Prof. Barus used drill rod and carried out a wonderfully exhaustive

¹ *Bulletin of the U. S. Geological Survey* (1885), 14.

² *Transactions of the American Society of Mechanical Engineers* (1914), 36, 845.

investigation, using electrical conductivity methods for measuring the softening.

BRADLEY STOUGHTON, New York, N. Y.—No one will deny the very great scientific and practical importance of the subject treated by Prof. Hayward and Mr. Raymond. The results are astonishing to anyone familiar with the tempering and annealing of steel, but the reputation of Prof. Hayward, the confidence he expressed in the work of the graduate student working under him, and the circumstance that the paper offers an opportunity for the discussion of a very timely and important subject, decided the Committee on Publications to accept the paper. A preliminary study of the literature by the Committee, as well as the searches made by the authors, as announced on the first page of their paper, failed to disclose any definite evidence in contradiction of their conclusions. After the paper was accepted, however, time was available for a much more extensive search, which has brought out data which, if not generally known, will alone justify the publication of this paper and its presentation here.

In the appended bibliography are given articles dealing with the effect of time on the tempering of steel at various temperatures. A study of these articles throws very important light upon the conclusions of Prof. Hayward and Mr. Raymond, and we may say that they corroborate the following conclusion given near the end of the paper: "for reheating quenched medium carbon steel to temperatures below 500° it is only necessary to heat it through. Longer heating has little or no effect on the tensile strength, ductility or hardness. At temperatures above 500° increasing the time of treatment causes a slight falling off in hardness and tensile strength with a corresponding increase in ductility." This conclusion seems to be justified not only for hypoeutectoid steels investigated by these authors, but also for hypereutectoid steels. But in the first paragraphs of their paper, under the heading "Discussion of Results," the authors give other conclusions which seem to be at variance with previous investigations, and I do not think that these are warranted.

I do not think the conclusion is warranted that there is no important difference in the results as to elastic limit, breaking load, yield point, and hardness, between hardened steels which have been reheated for 15 min. and those which have been reheated for 4 hr. at temperatures of 300°, 400°, 500° and 600°. Nor do I think that the authors are warranted in saying, as they do in the last paragraph of their paper, that "time was found to be of so little importance, it is not probable that the results were influenced appreciably by the time the specimens were in the furnace below the desired temperatures."

The authors say that they placed nine bars in the reheating furnace, which was unfortunately of so small a capacity that the introduction of the specimens caused a large drop in temperature, and it required about

40 min. to bring it back to the desired point. It would appear from the authors' own results and from the results of previous investigators, that the heating which the steels underwent during the 40 min., while they were being heated to the desired point had an important annealing effect upon them. Therefore, when the authors heated the steel 40 min. to the desired point and then continued the heating for 15 min., instead of giving the time of heating at the desired temperature as 15 min., it would actually be 15 min. plus the equivalent at that temperature of the 40 min. required to heat the steel to the desired point. We have no way of knowing what the equivalent of that 40 min. of heating at various lower temperatures is, but we may assume it to be 10 min., 20 min., 30 min., or even 40 min. Let us for the sake of argument assume it to be equivalent to 20 min., then the reheating of the steel at the desired point would be equivalent not to 15 min., but to 35 min. Now the results of other investigators have shown that 35 min. is practically sufficient to completely temper steels of $1\frac{3}{16}$ -in. diameter at 300°, 400° and so on. In other words, the authors practically completed all the tempering effect they were capable of producing at that temperature on the steels when heating them a length of time which they call 15 min. It is, therefore, natural to expect that the reheating for a longer period would not have any appreciable effect, and this is in fact what is shown by the paper under discussion. The paper is important in corroborating the results of others, and in indicating that for reheating steels to temperatures below 500° it is only necessary to heat them through. This fact has been indicated by the results of previous investigators also, but the authors of this paper have added confirmatory evidence.

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H. M. BOYLSTON, Cambridge, Mass.—I would like to offer another possible explanation in part of the failure to show greater differences at the times mentioned. I would like to ask Prof. Hayward whether, when he examined, under the microscope, the steel after the first quenching, it was all martensitic in structure?

CARLE R. HAYWARD.—It was not entirely martensitic.

H. M. BOYLSTON.—Did it not show white spots on a dark background?

CARLE R. HAYWARD.—Yes.

H. M. BOYLSTON.—I thought so, because the manganese is very low, 0.056, and I feel that 800° C., the quenching temperature he used, is practically in the critical range, instead of being above it. I believe he used, not a martensitic steel to begin his tempering on, but a troostite-martensitic. He thus had less tempering to perform. It seems to me that the three explanations given all work together to account for the failure to show greater differences.

J. A. MATHEWS.—I would like to inquire of Prof. Hayward if that chemical analysis is right, if it is 0.05 manganese it must be steel of special character, because commercially they are rarely made that way.

CARLE R. HAYWARD.—I did not make that chemical analysis; it was done by the chemist and I accepted his figures as the report made.

WILLIAM CAMPBELL, New York, N. Y.—I have read this paper with a great deal of interest, and especially so because I did some work along the same lines. I started out with a steel of 45 carbon, 1 per cent. manganese, and I got the result that I expected. I had another steel of 45 carbon and 0.1 per cent. manganese, and I went at it the same way and the results were not at all what I had expected, and on making an examination under the microscope, as Mr. Boylston mentioned, I found a good deal of the ferrite was either unabsorbed or had precipitated out; in other words, the small amount, or the lack of manganese, had such a great effect on the hardening qualities of the steel. In regard to the question of heating, I mean the time that it takes to heat up the steel, I think the paper would be made of somewhat more value if Prof. Hayward would take a sample of the hardened material with a thermocouple in the center, and repeat the heating in order to get a time temperature curve to show the heating of the steel itself, and in that way we would be able to judge what the effect of the time heating was. I do not think that the explanation that it took so long to heat up can be the only one to give such smooth curves as are given on page 283. I would also like Prof. Hayward to add the hardening figure for the sample quenched at 800, which appears blank on page 283, because I think that would tell us also something about the amount of softening brought about.

THE CHAIRMAN (ALBERT SAUVEUR, Cambridge, Mass.).—I should like to ask Prof. Hayward if he noticed the tempering colors of the steel at different times?

CARLE R. HAYWARD.—The colors were affected because of the fact that the work was done in a muffle furnace and no attempt was made to keep the atmosphere neutral. A scale of perhaps $\frac{1}{32}$ in. was formed on the specimens; all were blue. When this scale came off, as it did in the quenching, the color underneath was usually about a straw color; it varied somewhat because part of the scale stuck and we did not take the trouble to chip it all off and study the color. I would like to add that we found it practically impossible to get the hardness with that hardened specimen; the ball on the Brinnell machine broke and we finally gave it up. It seemed to be very hard. I would like to call attention to my last paragraph, which I will read, if I may. I have recognized the fact that this heating for a long time in the furnace to bring the specimens up to temperature was unfortunate, and in doing the experiment again, I think I should do it in a little different way.

"It is unfortunate that the heat capacity of the furnace used was so small that the introduction of the specimens caused such a large drop in temperature and a relatively long time in bringing it back to the desired point. However, in view of the fact that time was found to be of so little importance, it is not probable that the results were influenced appreciably by the time the specimens were in the furnace below the desired temperature."

I think that it is very important to get the specimens heated through. I do not know how we are to determine just how long that does take. The temperature, 800°, was determined by getting a cooling curve to decide as to the upper critical point and we found that that was just below 800°, and that was the reason it was taken. We assume that, with specimens of that size, it was impossible to get a martensitic structure throughout. I recognize the fact that in the center it was troosto-martensitic; at the surface it was martensitic, and I doubt if it is possible, by ordinary water quenching, to take a specimen the size of a standard tensile specimen, and get it martensitic throughout.

M. H. MEDWEDEFF, Springfield, Mass.—One gathers from Prof. Hayward's statements that manganese has no influence upon the time and drawing temperatures. Actual experience in large-scale production demonstrates the contrary. Manganese has indeed a decided influence on both the drawing temperature and time in the furnace. An actual case in illustration: In tempering heat-treated gun barrels, it is found that to obtain certain desired results, *i.e.*, with reference to their physical tests, a variation in the manganese content of 0.10 per cent.

necessitates a change in both the time and drawing temperature. In general, for any steel the time element is of utmost importance in heat-treatment work, depending largely on size and to a considerable extent on composition.

CARLE R. HAYWARD.—Of course, I recognize that the figures do not apply to any alloy steels, but that they have to have a special heat treatment. I think it is also undoubtedly true that high-carbon steel would be affected by time, and the title of the paper was therefore not made general, but applied to the effect of time on reheating. The paper applies to medium-carbon steel, but we do not wish to make it general and covering high-carbon steel or alloys.

J. A. MATHEWS.—I am afraid that some of the users of steel will make it general and assume that the time of tempering does not make much difference. I would hate to have some of our customers do that.

O. A. KNIGHT, Cambridge, Mass. (communication to the Secretary*).—It is needless to say that with the great industrial importance of steel at the present time, and the variety of properties which may be obtained from a steel of given composition by varying the heat treatment applied, too much cannot be learned in connection with the subject presented by Prof. Hayward. A subject so practical deserves the most careful study and research. In conducting our researches, however, we should approach as nearly as possible the conditions best suited to actual practice, and new experiments should be conducted with at least an accuracy equal to the older ones.

Let us consider briefly the method of drawing the quenched specimens that was applied in this investigation. The specimens, after having been quenched in water at 4° C., were plunged immediately into a furnace whose temperature was the maximum to be used in the drawing operation, viz., 300° C., 400° C., 500° C. and 600° C. When we consider the severely strained condition of a steel after quenching in water at 4° C. it becomes apparent that this is a more or less drastic treatment to be applied. True enough, it may produce no visible defects with pieces of uniform dimensions, such as were used in this research, but with pieces whose dimensions are less uniform, as is the case with many commercial articles, we might find, and in many cases do find, that the piece will crack even under the most suitable conditions. Even though no visible defects occur, the internal strains which might be set up will most certainly work to no good advantage.

* Received Mar. 3, 1917.

From these considerations it seems as though a more suitable method would have been to place the hardened specimens in the furnace at atmospheric temperature, and bring about the desired temperature more gradually, thus easing up the internal strains already present, due to the quenching operation.

It might be argued that these experiments were carried out primarily to determine the effect of the exact time at the temperature of drawing. This being the object, it seems that a better method of procedure would have been to plunge the hardened specimens into a salt bath already at the desired temperature, and sufficiently large to insure its temperature not being appreciably lowered by introducing the specimens. This method, by agitating the specimens, would bring about the maximum temperature in a few minutes instead of 40. Portevin¹ shows that by this method of heating, the desired temperature of specimens of the diameter of those used in the present research may be obtained in a period of only a few minutes. He also gives the various salt mixtures used to make up the bath.

Another part of the paper by Prof. Hayward and Mr. Raymond to which I should like to call attention is paragraph three of the first page, the first sentence of which reads as follows:

"A search of the literature disclosed several brief statements that the tempering effect was a function of both time and temperature, but in no case were any figures given."

I would like to refer the authors of this paper to an extended research performed by Barus and Strouhal, *Bulletin 14, U. S. Geological Survey, 1885*, entitled "On the Physical Characteristics of Iron Carburets." They would find on page 43 a sub-title, "On the Bearing of the Time of Exposure on the Efficacy of Annealing." Under this heading vast numbers of figures are given on the effect of time of annealing at different temperatures.

These experiments were conducted with steel wire in the form of rods 30 cm. long and from 0.03 to 0.01 cm. in diameter. The steel was known as "English silver-steel." The wires were heated by being made part of an electrical circuit, and sufficient current was used to bring about the desired temperature, after which they were water-quenched by turning on a sudden flow of water, the specimens remaining in the heating device, and were surrounded by a thin glass cylinder. CO₂ gas was passed through the tube during heating to prevent excessive oxidation.

After hardening the steel, it was connected thermo-electrically to a copper wire which itself had previously been compared to that of silver. The thermocouple thus formed was the source of electromotive force sub-

¹Influence Du Temps De Chauffage Avant La Trempe. *Revue de Métallurgie* (Jan.-Feb., 1916), 13, 9-63.

sequently determined. The thermo-electric power measured was supposed to bear a direct relation to the hardness of the sample.

After hardening the wires and making thermocouples of them as above described, their "hot junction" was inserted in the various mediums used for drawing (steam, vapor of boiling methyl alcohol, vapor of boiling analine, and molten lead) and the value of the thermo-electric constants noted from time to time. The constant whose value is recorded in the following data is the thermo-electric constant "a" of Avenarius. The "glass-hard" steel gave negative values to "a" (with one exception which was a low positive value) which approached zero and increased in positive value as the drawing temperature and time were increased, this phenomenon giving a relation for comparing the relative degrees of hardness.

The following table, taken from the paper of Barus and Strouhal, gives the variation in the value "a" of two hardened samples heated in steam at atmospheric pressure for varying lengths of time.

Sample No.	0 Hr.	1 Hr.	2 Hr.	3 Hr.	4 Hr.	5 Hr.	6 Hr.
24 - a =	-2.83	+0.61	1.26	1.61	1.76	1.70	1.92
25 - a =	+0.13	2.75	3.64	3.55	3.77	3.90	4.02

The wide difference in the two samples is clearly seen, but the relation of time is likewise apparent.

They then say: "From this grouping of parallel results, or, more evidently still, from a graphic representation (time as abscissæ, thermo-electric constant as ordinate, mean values of No. 24 and 25, Fig. 9)², it will be seen that hardness varies continuously with the time of exposure of the glass-hard rod to the given annealing temperature; that the amount of thermo-electric change rapidly decreases as the time increases, until finally a definite and superior limiting value is asymptotically reached."

Similar results were obtained in the other investigations. The factor *time* plays a less important part as the temperature of annealing the hardened metal increases. A fair understanding of their work may be gained from a study of the mean value of results which are given under the title of "General Discussion of the Results of This Annealing." The following is taken from page 54 of their paper. "The results thus far given adequately exhibit the general physical character of the process of tempering. For the sake of clearness and with a view to partially eliminating such discrepancies as are due to incidental errors, the three individual values of thermo-electric power and specific resistance for each of the temperatures of annealing will be combined and their mean chosen for discussion. We thus arrive at the following relations:

² *Bulletin of the U. S. Geological Survey* (1885), 14, 46 (Paper by Barus and Strouhal).

TABLE 28.—*Mean Results*

1. For annealing in vapor of boiling methyl-alcohol (66°)

Time of annealing =	0 hr.	1 hr.	2 hr.	3 hr.
a =	-1.62	-1.07	-0.80	-0.60

2. For annealing in steam (100°)

Time of annealing =	0 hr.	$\frac{1}{8}$ hr.	$\frac{1}{2}$ hr.	1 hr.	2 hr.	3 hr.
a =	-0.91	+0.17	0.95	1.50	2.08	2.36

3. For annealing in vapor of boiling aniline (185°)

Time of annealing =	0 hr.	$\frac{1}{8}$ hr.	$\frac{1}{2}$ hr.	1 hr.	2 hr.	3 hr.
a =	-0.95	+4.19	4.71	5.00	5.18	5.40

4. For annealing in molten lead (330°)

Time of annealing =	0 hr.	$\frac{1}{60}$ hr.	$\frac{1}{2}$ hr.	$\frac{3}{4}$ hr.
a =	-0.37	8.11	8.26	8.30

"*Deduction.*—If these functionalities are constructed graphically, as has been done in Fig. 10 (time as abscissa, thermo-electric constant as ordinates)³, we obtain a family of typical curves, the general character of which is distinctly pronounced, and may be thus expressed:

"The degree of hardness retained by a glass-hard rod, after having been subjected to the operation of annealing, is dependent both on the temperature to which it has been exposed and on the interval of time during which this exposure has taken place, in such a way that the effect of time, though of predominating importance in the case of small values of temperature, is more and more negligible in proportion as these values increase. The operation is always most effective in the earlier stages, and this efficiency decreases very slowly where the temperatures are low—very rapidly, indeed almost suddenly, where they are high. If the action of any temperature be indefinitely prolonged, the rod under its influence ultimately reaches an inferior and limiting degree of hardness, characteristic both of the temperature chosen and the type of steel under experiment."

Again I should like to call attention to a paper by J. A. Mathews and H. J. Stagg, Jr., published in the *American Society of Mechanical Engineers, Transactions* of 1914, the subject being "Factors in Hardening Tool Steel." On page 861 we find paragraphs 43 and 44 reading as follows:

43. "Regarding the effect of time on drawing the temper, we submit the following: Standard $\frac{1}{2}$ -in. round A. S. T. M. test pieces were quenched from constant temperature into the same medium, and the

³ *Op. cit.*, 55.

temper drawn in the same salt bath at constant temperature for 5 min., 15 min., etc.

Table of Results

Elastic Limit	Maximum Strength	Elongation	Reduction	Brinell Hardness	Remarks
228,750	260,137	2.5		425	1,550-Oil-800° F. 8 min.
201,125	214,562	11.6	45.4	390	1,550-Oil-800° F. 20 min.
175,000	183,187	12.0	49.35	340	1,550-Oil-800° F. 40 min.

44. "Each of these results is the average of four closely agreeing checks. A study of the above table shows that time at the drawing temperature has a marked effect. The act of breaking down the marten-site is progressive and not sharply defined. Both time and temperature have their effects."

That time plays a less important part as the drawing temperature is augmented is not only logical but seems already well established. The temperatures employed in the research under discussion were sufficiently high in all cases to minimize the effect of time. Undoubtedly the reason that time had no more marked effect is that owing to the temperatures employed, a great deal of tempering took place during the period of heating.

I hope that I may not be misunderstood in this discussion, as I do not advocate that further carefully conducted research along this line might not prove of value, but I hope the researches above referred to will be of benefit in correcting the erroneous statement of the authors, and also to reveal the fact that as early as 1885 it was quite definitely proven that the time factor is of minimum value in the range of tempering temperatures employed in current practice.

In conclusion, I wish to say that the points which I hope my discussion will bring out are (1) that the authors' conclusions are based upon a method which is not as accurate as methods employed by earlier investigators, and (2) their statement concerning the literature reveals the fact that they made an inadequate search.

The Effect of Sulphur on Low-Carbon Steel

A discussion of the paper of CARLE R. HAYWARD, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October 1916, pp. 1841 to 1850.

THE CHAIRMAN (ALBERT SAUVEUR, Boston, Mass.).—I am sure we are indebted to Prof. Hayward for his addition to our knowledge of the influence of sulphur on steel. As he has said in this paper, sulphur has had many defenders in recent years. It seems to me, however, that, while it may be old school to believe in the detrimental action of sulphur, it is not yet old school to believe in segregation; that while it may be that 0.1 per cent. of sulphur has no very great effect on the physical properties of steel, a ladle analysis of 0.1 per cent. of sulphur may readily result in 0.2 or 0.3 per cent. sulphur being present in the segregated portion of the ingot, and that, it seems to me, might be a rather serious matter.

J. S. UNGER, Pittsburgh, Pa.—Prof. Hayward has presented the results of an investigation along a line to which I have devoted considerable time and attention, and I am particularly pleased to note that his results are in accord with my own and further tend to prove the fact that the old established prejudice against sulphur is based more on belief than actual facts.

I have felt for years that too much stress was placed on the harmful influence of certain elements in steel such as sulphur, phosphorus, copper, etc., but it has only been within recent years that efforts have been made to establish the truth. Practically all the investigations made to date have proven that these elements within reasonable limits are harmless and, in fact, that for certain purposes the addition of these elements is beneficial.

In referring to copper, J. E. Stead, Vice-President of the British Iron and Steel Institute, said at their last meeting: "Even today one comes across steel specifications in which copper is barred, which can only be regarded as an indication of ignorance, if not stupidity, of those who prescribe the composition of steel, for it has been long ago proved that copper in steel, instead of being an evil, is quite harmless, and is sometimes distinctly beneficial."

Sulphur owes its evil name to the early days of the manufacture of steel. Chemical analyses were crude, and failure due to either poor raw materials or metallurgical treatment were many. Sulphur being a comparatively easy element to determine with reasonable accuracy, particular attention was called to the amount of sulphur, and the failure was attributed to it, without giving any consideration to other causes. A pre-

judice against sulphur was thus formed, which has existed to this day. As a result, the permissible sulphur content in steel is so low in some cases that it is questionable if the excessive purification necessary to produce such results is not detrimental to the steel.

It has been my experience in the working of steel, that occasionally a high-sulphur steel would roll or work much better than a lower-sulphur steel, yet the steels would be apparently the same in composition, excepting the sulphur. At other times the reverse would be true, and the lower-sulphur steel would roll better than the high-sulphur steel. If a certain amount of sulphur produces a particular effect, to be consistent it should always produce the same effect.

My investigation of the influence of sulphur in steel covered tests of considerable quantities of several kinds of steel in various sizes ranging from small roofing nails to 8-in. channels and railway axles, with sulphur contents of from 0.025 to 0.254 per cent.

The products tested were rivets, machine- and hand-made chains, sheets, wire, nails, tubes, pipe, drop forgings, channels, plates, axles, rails and wire cable. Owing to the variety of products investigated, it was sometimes necessary to use steel a little softer or harder than that ordinarily used for the same purpose. This, however, had no influence on the comparative results.

In the fabrication of these products, the usual works practice was followed in the production of the finished article, no preference being given in the treatment of steel of one sulphur content over that of another. The material was subjected to the tests regularly prescribed for steel products of the varieties mentioned and, in addition, special tests were made on most of the products.

It was not my object to study the state in which the sulphur existed in steel, or its microstructure. What I wanted to learn was what effect different amounts of sulphur had on the working of the steel and its effects on the properties of the finished article. This is what concerns the user of the steel.

A paper read before the Society of Automobile Engineers and published in their *Proceedings* for 1916 and in the *Iron Age*, January 13, 1916, gives the results of a great many of these tests, particularly those of interest to the automobile industry. Another paper presented before the American Boiler Manufacturers' Association and published in their *Proceedings* for 1916 and in the *Blast Furnace and Steel Plant*, July, 1916, describes an extensive series of tests on the influence of sulphur in rivet steel.

I will not review these results here, but will briefly refer to some of the tests which were not covered in these papers; such as tubes, plates, rails and wire products.

Referring first to the tests on tubes, a series of fifty 2-in. boiler tubes

were made of each sulphur content, and all met the regulation hydraulic pressure test without failure. The welding practice on the tubes of 0.050 and 0.090 per cent. sulphur was better than the practice on the tubes containing 0.030 per cent. sulphur. A series of 4-in. seamless tubes were also made and subjected to different tests. The tubes of 0.068 sulphur as a whole gave better results than the 0.032 per cent. sulphur tubes. In the higher-sulphur steels there was a slight falling off in tensile strength, but an increase in ductility as shown by the elongation.

The investigation of plates consisted of an extensive series of longitudinal and transverse tensile, torsion and bend tests, welding and hot and cold flanging tests. The results of these tests were practically the same for plates of all sulphur contents.

Steel of 0.51 per cent. carbon with varying sulphur contents was rolled into 100-lb. rails. Tensile and drop tests were made. There was a slight decrease in the tensile strength in the high-sulphur rails, but an increase in ductility as shown by the elongation in both the tensile and drop tests. Rails of each sulphur content were then put into actual service in a railroad track, and although lower in carbon than steel ordinarily used for rails, which reduced the life of the rails, all of them, regardless of the sulphur content, showed about the same amount of wear.

In the investigation of wire products, one of the studies made was the influence of sulphur in the manufacture of roofing nails with a very large, thin head. The upsetting of the head is a particularly severe operation, yet the steel of the highest-sulphur content made as perfect a head as the lower-sulphur steels.

Another study was the electrical welding of wire. When welding the wires for continuous drawing or in the manufacture of fencing, where the wires are welded at right angles to each other, it was found that steels of all sulphur contents could be perfectly welded.

In making the regulation twist and bend test of galvanized wire, the wires all met the requirements of this test, and the coating adhered perfectly, regardless of sulphur content.

The tests of wire cable showed that cables containing up to 0.10 per cent. sulphur gave practically the same results in the tensile, twist and life tests.

Attention has been called by other investigators to the fibrous condition of high-sulphur steels. This fact was particularly noticeable in our study of case-hardening. After the steels had been case-hardened and quenched, all showed the same depth of case and hardness. When fractured, the low-sulphur steels would snap off square and show a crystalline fracture, while the higher-sulphur steels, instead of snapping off square, had to be bent backward and forward before they would break, and the fracture would be irregular and have a fibrous appearance.

The results of this investigation confirmed my belief that steel could

contain a great deal more sulphur than that usually permitted and still be of good quality. The work of Wahlberg, Arnold, Stead, Cooper, Hayward and other investigators has done much to strengthen this belief. There is still a large field for investigation and a much larger work in distributing the knowledge thus gained to all those interested. This seems to be the only way by which any erroneous beliefs can be replaced by actual facts, which later find a practical application.

GEORGE F. COMSTOCK, Niagara Falls, N. Y. (written discussion).—The results of tests given in Prof. Hayward's paper are interesting, and are worthy of serious attention by those who advocate an appreciable increase in the permissible limits of sulphur content in steel. It does not seem, however, that Prof. Hayward in his conclusions has rendered an entirely impartial verdict. The beginning of the paper would indicate that he started out with the intention of proving if possible that sulphur up to 0.1 or 0.15 per cent. does no harm to steel, and the tensile tests were extremely encouraging to such a conclusion. It should be noted, however, that the high-sulphur steels were no better on the whole in the tensile tests than the low-sulphur steels.

The impact tests, on the other hand, do show a marked difference in favor of the low-sulphur steels, but Prof. Hayward is apparently unwilling to admit in his conclusions that any particular importance attaches to this fact. The excuse is given that the shock test is new and its value not thoroughly appreciated by all engineers. This, I suppose, is a matter of opinion, but it surely would seem that a glance through the published records of technical societies interested in metals or their testing, both in this country and Europe, should be sufficient to show that the shock test is not new but is well known and extensively used. Looking at the matter from the viewpoint of the designing engineer, it might be asked how metals usually fail in service, whether by slow, gradually increasing tension, or from small repeated stresses which cause fatigue, or from sudden shocks? It seems obvious that failure from pure slow tension practically never occurs in service, and that fracture from shock is far more common, although probably not so common as failure from fatigue. When looked at in this light it is hard to see why Prof. Hayward's shock test results should not be given serious attention in judging of the practical value of high-sulphur steels.

The actual results of the work described in this paper may be summarized as follows: Tensile tests showed on the whole no difference between high-sulphur and low-sulphur steels; impact tests showed a distinct difference in favor of low-sulphur steels; microstructure not discussed. In view of this summary, where nothing is shown in favor of the high-sulphur steels, but one point of decided superiority is brought out for the low-sulphur steels, is there any logical escape from the con-

clusion that steel low in sulphur is better and safer to use than steel with high sulphur?

The failure of the tensile tests to show a difference between these classes of steel is very similar to its results on steels with varying amounts of phosphorus, or on steels heat-treated differently. It has been shown by very eminent authorities that steel high in phosphorus may give good tensile strength and elongation, and still show great brittleness in shock tests. This is a case directly in line with Prof. Hayward's results with sulphur, and if high-phosphorus steels are considered unsafe to use in spite of fairly good tensile results, then Prof. Hayward's paper can only be regarded as showing that high-sulphur steels are equally unsafe.

I notice that Prof. Hayward does not discuss the microstructures of these samples, although photomicrographs were made and are reproduced in the paper. I would like to ask what conclusions are to be drawn from them? It always seems unfortunate to me that photomicrographs should be considered as proving anything in themselves in cases like this, for of course no two photomicrographs even of the same sample would be exactly alike. The true use of a photomicrograph is, I believe, in illustrating points that would be less clear without the actual views, and I would like to ask whether Prof. Hayward considers that these photomicrographs show the structures of high-sulphur and low-sulphur steels to be alike, or, if not, what differences are shown?

If photomicrographs had been made before the samples were etched, there is no doubt that much greater differences would have been shown in the cleanness of the metal. The excessive amount of non-metallic manganese sulphide in the high-sulphur steel could be easily seen in an unetched sample, and would illustrate well the reason why such steel is easy to machine, and also why it is brittle and unfit for severe service. Sulphide inclusions are as harmful as any other non-metallic impurities in steel, and more so than some because they tend to segregate. Unless we disagree entirely with the eminent authorities who maintain that non-metallic inclusions are harmful in steel, we cannot logically admit that high-sulphur steel is all right.

It is unfortunate that Prof. Hayward did not continue the investigation to include alternating stress tests, even if new stock would have been required. Surely he does not consider his results of no general value and inapplicable to any but the identical steels on which he worked, and it is not clear why the results of fatigue tests on high-sulphur and low-sulphur steels should have any "uncertainty in their interpretation" merely because the stock was different from that used in the impact and tensile tests. It is to be hoped that such an investigation will soon be made, in order to show, more clearly even than Prof. Hayward's present results show it, why high-sulphur steel is unsafe to use for severe service.

M. H. MEDWEDEFF, Springfield, Mass.—I recollect when Dr. Ungers' paper on the "Effect of Sulphur in Steel" appeared in the *Iron Age*, I intended to offer a criticism more with the hope of having some more competent authority challenge the rather revolutionary statements contained therein. It was, however, never published, as I hoped to secure substantiating figures in defense of the current specifications for sulphur limits.

While connected with a steel works producing merchant open-hearth bars, splitting ingots in the process of rolling were repeatedly found to be caused by the high sulphur in the steel, the other constituents being normal. The ingots at the same time appear to have a fibrous appearance. This may perhaps be ascribed to the rolling out of the sulphides into filaments. Sulphur exists in steel as MnS, or perhaps both as FeS and MnS, particularly when the manganese content is low. The melting points of these sulphides are considerably below the melting points of hypoeutectoid steel. Now if the steel is high in sulphur, it contains a considerable amount of MnS or MnS plus FeS; it is reasonable to suppose that as these impurities are of lower melting point, they would be thrown out to the grain boundary of the metal on cooling. Granting that high sulphur may even favor an increase in the tensile strength and elongation, the question rises in one's mind, will such steel stand transverse strains or shocks? Prof. Hayward answers that question himself in his Charpy tests figures. From the above considerations of the discontinuity in the structure of such steels, one may well expect it.

One cannot help but wish that Dr. Unger would give us the secret of how to weld high-sulphur steel; it is the general experience that such steels are stringy and unweldable.

CHAIRMAN SAUVEUR.—I would like to ask Dr. Unger about those different sulphur steel tests. Do I understand they were all less than 1 per cent. sulphur?

J. S. UNGER.—I had steels of about 0.09 per cent. carbon that ranged in sulphur from 0.030 to 0.254 per cent.; of 0.32 per cent. carbon, from about 0.032 to 0.230 per cent. sulphur; and of 0.51 per cent. carbon, that ranged from 0.025 to 0.230 per cent. sulphur. Those figures can, however, be obtained in my former paper. I do not recollect the exact figures, but in making these tests I want to say, if you are selling to a customer steel that he proposes to weld into chain, the man that finally uses the chain is not interested to a very large extent as to what tensile strength a bar may possess. He is interested, however, in the strength of that chain.

In making this investigation I found that there was a greater possibility of making a poor weld in a chain, whether it was of low sulphur

or high sulphur, than any slight change that might exist in the physical properties of the steel.

If we make a beam, channel or an angle, the usual method of testing is to make a tensile test. A beam is very rarely subjected to a tensile strain; it is either used as a column or as a girder and it is subjected to compressive or transverse loading. In testing the channels we made, we took full-sized channels, 6 ft. long, and tested them transversely to determine at what point they would take a permanent set, which imitated service conditions.

In testing the rails, we did not pay a great deal of attention to what they might show on tensile or drop tests; but we put them in a track that had to carry 2,000,000 tons of freight a month, and ran them for a year until they were practically worn out. We then compared them with other rails of a normal composition installed at the same time and location to find out whether they gave the same results. I assure you that until I learned they were safe, I watched the high-sulphur rails very carefully to observe the least signs of failure; so that they might be removed before an accident happened.

In making tests, the thing you must consider is, what is the customer going to do with the steel? You may make tensile, hardness, impact and all sorts of tests, but what you want to know is whether the steel is suitable for the purpose for which you are going to use it, not whether it is going to pass a particular physical or mechanical test.

G. AERTSEN, Philadelphia, Pa.—As I understood Dr. Unger, some of these tests on high and low steel were made on plate steel?

J. S. UNGER.—Yes.

G. AERTSEN.—Did you notice any greater difference between longitudinal and transverse tests on the high-sulphur steel than on low-sulphur steel?

J. S. UNGER.—We know, of course, that there is a pronounced difference between the longitudinal and the transverse tests on all plates, but there was not any greater difference on the high-sulphur steels than on the low-sulphur steels between the longitudinal and transverse tests.

G. AERTSEN.—Was there very much difference noticeable in the surface appearance of the high-sulphur plates? In other words, were they cleaner or less clean plates with low sulphur than with high?

J. S. UNGER.—I will answer that in this way: I kept on adding sulphur to the steel until I got to a point at which the steel was difficult to roll. I believe I could have rolled the highest-sulphur steels that I had, had I been willing to lower the rolling temperature, but my object was to roll these plates at a regular working temperature, not to make a special

provision on account of their being high in sulphur. I am satisfied that I could have rolled them if I had reduced the rolling temperature about 100°. But to answer your question, after the sulphur had exceeded about 0.150, I obtained a surface that was inclined to be slivery, it was what we call snakey in the mill.

M. H. MEDWEDEFF.—I believe you said it did not make any difference on the welding properties of the steel?

J. S. UNGER.—You are correct. I brought out in my paper why it seems that sulphur was the only possible explanation, the reason being that if we cannot find any other reason, we pick out the sulphur and blame it on that.

LEONARD WALDO, New York, N. Y.—There is another connection in which the paper Dr. Unger published a year or so ago and this paper of Prof. Hayward are papers of great importance. The fuel near the Atlantic seacoast for open-hearth steel practice in the future will be Mexican crude oil. Mexican crude oil runs $3\frac{1}{2}$ to 4 per cent. of sulphur; in the manufacture of many thousands of tons of steel, I found it practically impossible to prevent the addition of perhaps 0.01 per cent. of sulphur to the ingot steel, from the fuel alone. The education which has been persistently spread—whether because of gas fuel existing in certain localities (which has no sulphur) or not, I do not know—makes open-hearth superintendents hesitate over the addition of 0.01 per cent. of sulphur to a steel, which, in its ordinary product, would give 0.03. Dr. Unger has pointed out the error in this point of view, and he has only stated in words what every maker of steel rails has known for several years back, that the slight addition of sulphur has no practical effect on the product. I think, therefore, that we ought to be especially grateful to gentlemen like Dr. Unger and Prof. Hayward for putting this thing in a professional paper so that the facts and the reasons may be stated to inquirers who otherwise would be face to face with a difficult fuel situation as the years go on.

A Method for Distinguishing Sulphides from Oxides in the Metallography of Steel

Discussion of the paper of GEORGE F. COMSTOCK, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2103 to 2110.

H. M. BOYLSTON, Cambridge, Mass.—It might be asked, since this test is so simple, why it was not discovered before. It seems to me the explanation is that when we have used the sodium picrate solution, it has generally been on a very high-carbon or hypereutectoid steel, to detect the presence of free cementite, and when we have looked for sulphides, it

has generally been in the strictly hypoeutectoid steels. We have never applied this reagent to low-carbon steels, and where it was applied, the sulphides were not examined with any degree of care.

GEORGE F. COMSTOCK, Niagara Falls, N. Y. (communication to the Secretary*).—Since writing this paper I have discovered in the literature a reference¹ to a paper by Siesching in *Metallurgie*, 1910, page 566, in which a method for darkening sulphides is given, which is somewhat similar to my method. Siesching gave a dark-brown color to sulphide inclusions by etching first with picric acid, then with nitric acid, and finally with hot concentrated caustic soda. It is not stated in the reference that this treatment attacks no inclusions other than sulphides, and it would seem to be more cumbersome in application than my method, as well as being open to the objection that pearlite would be darkened by the acid treatment, thus hiding to a greater or less extent any sulphides that might be embedded in it.

Erosion of Guns—The Hardening of the Surface

Discussion of the paper of HENRY FAY, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2237 to 2251.

THE CHAIRMAN (ALBERT SAUVEUR, Cambridge, Mass.).—In forcing us to face and to discuss the important question of erosion of steel guns, Prof. Fay is performing a public service. His investigation has been fruitful and his results illuminating and suggestive.

It is not in a spirit of adverse criticism, therefore, that I venture a few remarks suggested by his paper, but rather to add my mite to the discussion in the hope that it may not be entirely without value.

Prof. Fay considers it conclusively demonstrated that the hard layers forming in the inside of guns of large calibers after they have been fired a number of times consist of martensite. Still, he admits that he never was able to detect in the hard layers the structural characteristics of martensite. This leads him to conceive the existence of amorphous martensite.

We may well hesitate before accepting this theory on the strength of the evidence offered. Martensite, as we know it, is essentially a crystalline condition assumed by a close association of iron and carbon, possibly if not probably of the nature of a solid solution. Lacking this crystalline character we are hardly justified in calling it martensite on the ground, merely, of its being very hard.

* Received Feb. 20, 1917.

¹ G. Rohl: The Constitution of the Sulphide Enclosures in Iron and Steel and the Desulphurization Process. *Iron and Steel Inst. Carnegie Scholarship Memoirs* (1912), 4, 39

The author appears to have demonstrated that the hard layers are thickest and hardest in those portions of the gun that had been subjected to the greatest amount of cold deformation. To account for this occurrence, he offers two possible explanations. In the first he assumes the formation of the amorphous state in proportion to the mechanical deformation, and he attributes to these amorphous layers the power of dissolving carbon and of being thereby converted into amorphous martensite. This, as the author undoubtedly realizes, is highly speculative.

I should like to ask Dr. Fay whether, in his opinion, this transformation of an amorphous aggregate of iron and carbon into amorphous martensite might take place above or below the critical range or both below and above it. It would seem as if it could occur only below the range, seeing that above the range the amorphous state no longer exists.

In his second explanation he conceives the possibility of sorbite being converted into amorphous martensite by work alone, or by pressure and heat combined. It is hardly thinkable, however, that pressure alone could convert sorbite into martensite—were it so, severely cold-worked wire should be martensite—and any piece of sorbitic steel could be made martensitic by crushing it in a testing machine.

Has not the author overlooked the important part that might be ascribed to cementation and quick cooling? He tells us that the products of combustion resulting from the burning of the powder contain much carbon monoxide, that the heat generated is sufficient to carry the skin of the metal, at least, above the critical range, and that the cooling of that skin must be very rapid because of the mass of metal abstracting heat from it. If we add to this that the gases are under great pressure which would greatly promote cementation, does it not seem that we have all the needed requirements for case hardening, and for the hardening of the carbonized layers, and that probable cementation and rapid cooling must be taken into consideration when attempting to account for the erosion of guns?

HENRY FAY.—I want to say that recent work has shown that we have been able to detect the martensitic structure. The reason why I did not get it in the first place was that I always had taken a cross-section which was at the maximum 0.0015 in. in thickness, and I never found it in that section. Polishing the face of the section, I have been able to detect martensite in it, not well-established martensite, but undoubtedly martensite. And what seems to me as conclusive is the fact that you can get well-developed troostite by tempering. The troostite can then be converted into sorbite, and then into pearlite.

HUDSON MAXIM, Brooklyn, N. Y.—When a gun is fired using a charge of modern smokeless powder—I refer, of course, to large guns—the heat

supposed to be generated by the combustion of nitrocellulose smokeless powder is about 4,000 to 4,500° F., possibly 5,000°, varying, I should say, with conditions, such as density of loading, weight of projectile, etc. The heat developed gives the products of combustion in a gun employing a nitroglycerin compound, that is to say, a compound of nitrocellulose and from 25 to 50 per cent. nitroglycerin, a temperature, it is supposed, of about 5,000 to 6,500° F.

Now, steel melts at 2,650° F., so you can see there is a large margin of temperature above the fusion of steel.

Then again, we must take into account the great density of the products of combustion under a pressure of about 35,000 lb. to the square inch. The result is that there is supposed to be a thin film of metal on the surface of the gun, fused, practically instantly, as far as our senses of measurement are concerned. When a 12-in. cannon is fired, the projectile is in the gun about $\frac{1}{16}$ sec.; the longer the gun, the longer the time and the greater the amount of erosion in the gun. These are some of the factors to take into account in attacking this problem of erosion in guns.

As soon as the projectile has left the gun, of course there is a terrific outrush of gases, and this thin film of metal is washed away, to a certain extent, by the outrushing gases; and when a big gun, of 12 or 15-in. caliber, is fired with the usual service charge of nitrocellulose smokeless powder, there is practically about 0.001 in. of erosion at every shot; that is to say, a film or layer of the gun has been fused and washed away enough to increase the caliber of the gun by about 0.001 in. at every shot. With the powders containing a large percentage of nitroglycerin, like those used abroad, especially in England, when they used 58 per cent. of nitroglycerin in the cordite, the erosion of the guns was many times greater than with pure nitrocellulose powder. The enormous advantages of employing a nitroglycerin compound are so great that it was very easy to err in the direction of the employment of nitroglycerin powder.

When one of our big, long, 12-in. naval guns is fired with a 1,000-lb. projectile, the amount of heat represented by the striking energy of the gun is, say, 50,000 tons—enough to melt 750 lb. of cast iron, which has to be taken from the powder gases, which are cooled by the act of propelling the projectile just that much. Now then, the time is so brief that there is not much depth to the heated layer in the gun. If you take a rod of steel and put it up to a very hot object, with the temperature of the electric arc or the temperature of a bar of carbon heated to the point where it will bend in an electric circuit, you can shove the steel bar up to it, melt off the end, volatilize the metal and not warm the bar to any considerable distance beyond the zone of fusion; consequently, immediately below the layer of heated metal in the gun, the gun is practically cold.

Now, when the projectile has left the gun, followed by the gases, the

thin film of heated metal necessarily contracts. In the act of heating and melting, of course, the metal has to displace itself upon itself, or crush itself, so that when it cools, it necessarily sets up terrific tension and cracks. Take a pane of glass, spread some glue over the surface of it and let the glue dry—the surface of the glass will be pulled off. The same thing is true if you dissolve some guncotton in acetone and pour it over the glass and let it dry, it will pull off the surface of the glass. The thin layer of this film adhering strongly to the surface of the glass will pull away the surface. Now then, when this thin film of heated metal shrinks in the gun, it not only cracks, but also it tends to deepen the cracks in the metal underlying or at the bottom of the cracks. We see that thing illustrated in a similar way when a puddle that has had in suspension a lot of earthy matter or clay has dried up and the surface has dried and cross-cracked, or crazed. The cracks not only go through the dry part, but they also pull apart the underlying clay and carry the cracks down into the wetter portion. The same thing takes place, it is presumed, in a gun. There is a tendency, also, to do work on the wall of the gun or the bore of the gun, due to the tread of the driving ring over the surface of the gun and the torsion set up by rotating the projectile, which serves to harden the surface, but I do not think that the mechanical work on the gun is anything like as destructive as the erosion from the powder gases.

Now, in this country, we have far less erosion of our guns than they have in any other country in the world. Erosion is, nevertheless, a very serious problem, even with us, but it is not so serious with us as it is with other nations who do not have a powder as good as ours. Before this war came they used to think, over on the other side, that their powder was better than ours, but they had to buy our powder after the war came, and then they learned that our powder is better than theirs. Of course, where they have built their guns over there to take nitroglycerin powder, employing smaller powder chambers, they have had to buy nitroglycerin powder from us and have it made for them, but we have, to a certain extent, in this country, met the problem of erosion with improvements in our smokeless powder.

I do not think it is any secret to say that when we took up the problem of rifle powders in this country we first tried a powder containing, say, 25 per cent. nitroglycerin, and I think the gun lasted about 3,000 rounds. Then we got some powder from the other side, fired that, and the gun lasted about 5,000 rounds. The DuPont Powder company then made some powder with which you can fire a rifle 20,000 times, and it will still be ready for more work, so the question of erosion of small arms is practically solved. Of course, there is room for still greater improvement, for there is great need of firing the maximum number of rounds with a rifle.

BRADLEY STOUGHTON.—There is a great difference in the thermal conductivity of nickel and the thermal conductivity of manganese steel. Both are lower than the thermal conductivity of pure steel. Now, if the life of the manganese-steel gun was very much lower than the life of the nickel-steel gun, it would tend to strongly corroborate the theory that Mr. Addicks has advanced¹ and perhaps he has some information on that point which he could give us.

LAWRENCE ADDICKS, New York, N. Y.—I regret to say that I have not.

HUDSON MAXIM.—May I say one word more about that matter of conductivity? I am very firmly impressed and believe that there is not very much conductivity. I should say that there is not time for a very great amount of conductivity from the heated surface of the metal. The condition we have when we fire a big gun is practically this: it is just like firing a shot out of a gun made of ice, using superheated steam for a propelling means. Now you get something of the idea of the condition in a cannon. If you figure it out, you will find that the result would be very much in favor of the ice gun using superheated steam, provided the ice were as dense and hard as steel.

LEONARD WALDO, New York, N. Y.—Some years ago it was my pleasure to be at Fishers Island, and while there in connection with the officers in service, I made a series of photographs of the interior of 12-in. guns after being fired. The photographic records are exceedingly interesting and very easily made. The structure of the edges is particularly clearly shown, and if the Department has no objection, I shall be very glad indeed to reprint these because they have never been made public. They give a very distinct idea of the nature of the disappearance of the edges of the rifling itself in the act of firing. With the erosion of guns, there also came up the question of a material to resist erosion. Breech blocks were made of aluminum bronze, and when I say aluminum bronze, I mean about 9.85 per cent. of aluminum and the balance pure copper. It was found to answer admirably for breech blocks. When it was proposed to use it for a gun, I pointed out that the modulus of elasticity was so low in any of the copper alloys, that it would not do for the gun itself, but that it might do for an inserted lining because it was shown to be very free from the action of these highly erosive smokeless powder gases under pressure. To try the experiment, I sent the rolled material for a rifle barrel to the Winchester Arms Co. Instead of using it as a lining, the company made the gun itself out of the aluminum bronze and sent back word that it was a very good barrel so far as resisting the erosion was concerned, but that it expanded under the impact of firing charges,

¹ *Bulletin* No. 123 (March, 1917), 416.

which was, of course, exactly what one would have expected; but in the aluminum bronzes, I venture to think there is the best of the untried materials for making the linings of these large guns, because if you carefully inspect the whole theory, you will find that it works better than anything that has been proposed, and in the auxiliary parts, breech blocks and those things, it has been shown to be highly resistant to the temperature and effect of corrosive gases. I am very much impressed with the fusion of the skin-deep stuff in the lining of the gun, but I do not quite catch the argument of our friend, Mr. Addicks,² about the failure of copper wire to take more than 40 twists in 6 in. on account of the exceedingly microscopic skin. I think that in the case of the copper wire, the working has gone pretty well through the wire itself and it is an integral cause of the failure of the wire to take this 60 or 100 twists as it should.

H. C. WILSON, Springfield, Mass.—This valuable paper deals specifically with the erosion of great guns, presenting problems quite different, in many respects, from those involved in small arms, with which I am immediately concerned. However, there are some questions of common interest which I would like to ask Prof. Fay.

Prof. Fay first says that after continued firing the surface of the bore becomes hard and brittle, cracks and wears away. Mechanically, to what extent may this deterioration be accelerated by the "whip" or intensive vibrations of the unsupported breech end of the gun, when we consider that a 12-in. 50-caliber gun is unsupported for 36 ft. beyond the trunnion mount to the muzzle, with an actual deflection of 0.2 in. below the bore-axis; equal to 4 or 5 min. muzzle angular deflection? Could the pounding or hammering of the traveling projectile be considered as "working" or forging the surface-metal of the bore? This axial vibration is sometimes so severe as to "ring" off the muzzles of our great guns. Would it be advisable to redesign the gun-chassis to better support this present outboard mass or otherwise dampen or flatten out these wave vibrations?

General Rohne has noted that the very fine skin veins of the fractured bore-surface appeared as ridges when magnified, and concluded that the metal had reached the melting point and had extruded under the high internal stresses.

It is interesting and valuable that Dr. Fay notes the impropriety of trying to compare tests made by firing powder charges in bombs or shells equipped with drilled plugs, and guns actually propelling projectiles; the conditions as to the erosive effect of the gases of combustion are quite different, largely due to the presence of the projectile in the gun, with no similar mass and obturation in the experimental bomb.

² *Bulletin* No. 123 (March, 1917), 416.

It appears that there must be two kinds of erosion in the bore of the gun, abrasive wear due to the travel of the projectile mass, and scorification or wire-drawing of the gases, not considering, for the moment, the destructive effect of great variations of temperature and pressure on the lining; and I recall that some years back the Ordnance department issued us projectiles with extra-wide rotating bands, having the immediate effect of increasing the M. V. and, we believe, accuracy, undoubtedly due to more efficient obturation. I believe that had we a spectroscopic screen or other suitable apparatus, we might have observed, and even measured, a darker color corona in the muzzle-blast, indicating more effective gas-checking of the broader banded projectiles.

Endeavoring to secure the highest polish possible in the small rifle bore, we have used several means, including lapping, and have also fired the first few rounds after dipping the bullets in deflocculated graphite, with highly satisfactory results. The resultant silvery-black surface seemed highly resistant to wear and erosion, was entirely unaffected by our hot-soda solution, and the subsequent lack of rusting indicated that the minute surface pores and scratches may have been covered or filled up by the graphite. I am wondering if this form of carbon did not have some case-hardening effect on the gun steel under the high temperature present.

As to alloy steel, the state of the art hardly yet justifies our use of other than a mild plain carbon steel, commercially basic open-hearth, of about 0.45 to 0.55 carbon, with manganese running from 1.0 to 1.28, silicon not over 0.30, phosphorus not over 0.06 and sulphur not over 0.06, so heat-treated as to give us a desired elastic limit running from 80,000 to 90,000 lb., with better than 20 per cent. elongation and 50 per cent. contraction, showing a satisfactory fracture, and free from seams, piping or segregation of any kind. Does Prof. Fay feel that the introduction of some alloy, molybdenum for instance, might give us equal or better machinability and ballistic characteristics, and at the same time decrease the erosion?

J. W. RICHARDS, South Bethlehem, Pa.—If the heat conductivity of the metal has anything to do with the forming of these surface cracks by the rapid heating and subsequent rapid cooling, I think the question could be investigated by Mr. Fay's method of taking the test plugs which he puts into the breech, putting on them a thin sheet of metal to be tested, and back of it some heat insulation so that the metal might become heated. By putting different amounts of heat insulation back of it, it could be heated and chilled more or less rapidly by cutting out the cooling influence of the metal back of the surface, so that the effect of the more or less rapid cooling by the metal back of the surface might be investigated.

Mr. Maxim has spoken of the fusion of the surface, because of the

high temperature, I think he used the term vaporization also, and I wish to speak a little on that point. The metals have vapor tensions at lower temperatures than is usually supposed. We know what the vapor tensions of some of the metals are at ordinary temperatures, and the tension goes up in an ascending logarithmic curve. We often speak of their boiling point, meaning their normal boiling point under one atmosphere pressure, but they have vapor pressures all the way down to ordinary temperatures. The vapor pressure of iron at its melting point is about 1 mm. of mercury, or $\frac{1}{460}$ atmosphere; but if you assume that at the temperature of the explosion the gases in the breech of the gun become saturated with the vapor of iron, you will find a quantitative correspondence between the loss of weight on the inside of the gun and the amount of iron vapor that could be contained in that volume of gases. Mr. Maxim has said that a large gun will gain about 0.001 in. in diameter in one shot; that 0.001 in. would correspond to a loss, on 1 cm. length of the gun, of about 0.6 gram of iron; whereas, if the gases in the gun were saturated with iron vapor, it would correspond to about 1 gram. The quantities are, therefore, of the same order of magnitude and the vaporization of the iron may account for the loss of weight, or the increase in caliber. This, if true, points to using means to reduce the vapor tension of the iron. We know that alloys of metals of higher vapor tensions have less vapor tension than the original metal; or, to put it in another way, that you can decrease the vapor tension of a metal by alloying it with a metal of higher boiling point. Now iron has a high normal boiling point, about 2,450° C., and there are not many metals with a higher normal boiling point, or a less vapor tension at a given temperature. The metals that can be named as having a higher boiling point than iron are particularly boron, titanium, tungsten and molybdenum, and the question to be solved, as a field for experiment, is whether the use of alloys of iron with these constituents would not decrease the vapor tension of the iron and possibly decrease the loss by vaporization in the firing of the gun.

ARTHUR L. WALKER, New York, N. Y.—I would like to say a few words in regard to the theory mentioned by Mr. Addicks, namely, the effect of a very thin surface coating of oxide of copper—the results he found were questioned. Sometime ago my attention was drawn to a case of serious defects in copper rolled from cakes cast in the regular manner. This copper cracked when subjected to the test imposed. The test was a very severe one. The copper cakes were rolled into sheets $\frac{1}{2}$ in. thick and cut into blanks. The blanks were cupped, and from the cupped blanks rings were cut to be shrunk on shells, the copper ring taking the rifle of the gun. A test piece cut from the rings was bent cold in the "back-bending" test—in other words bent 180° back on itself. From

the same charge, certain samples cracked while others did not. A microscopic examination showed that in every case one side of the ring had more suboxide of copper than the other, which would be the result naturally expected. When a cake is cast the surface is necessarily exposed to the atmosphere for a short time and a film of suboxide of copper is formed on top of the cake. In rolling, this film is rolled into the cake so that naturally one side of the sheet has for its surface a thin film of suboxide of copper. When the test piece from the cupped blank was bent so that the original surface of the cake was on the outside it was fractured, sometimes all the way through. When the test piece was bent so that the original surface of the cake was on the inside the sheets did not crack at all. This illustrates in a striking manner the effect of this thin film of suboxide of copper on the sheets. It also bears out Mr. Addick's theory as to the results which might be obtained by the removal of this oxide film. In the case mentioned, apparently the brittle eutectic of $\text{Cu}_2\text{O} + \text{Cu}$ cracked when subjected to stress and started a fracture which extends into the copper itself. I had no personal experience with the tests just mentioned; the results were simply brought to my attention and I have samples in my collection which illustrate them.

HENRY FAY.—I would like to say a word in regard to Major Wilson's amorphous metal or allotropic metal, as he stated it; I hesitated to use the term amorphous because of the many interpretations that might be put upon it. It seems to me that it is an important term, because we have a metal which is in a markedly different condition from that in which we ordinarily consider it; *i.e.*, as crystalline metal. We all know that if a metal is cold worked at the surface, it will corrode more readily at the point where it has been put in the amorphous or cold-worked condition, that is, it has a different solution pressure from the crystalline metal. I also used it because it seemed to me that if the metal is amorphous, having a greater solution pressure than the crystalline metal, we might also expect that it would have a greater solubility effect on iron carbide. In other words, amorphous metal might promote the solution of iron carbide with a stronger tendency to hold it in the form of solid solution, than the metal which was in the crystalline condition.

In regard to Major Wilson's other question, as to motion: It seems to me that that might be explained by the fact, as Mr. Maxim has said, that the interior of the gun increased in diameter 0.001 in. on firing, and having this hard skin, it might easily, on cooling, contract and crush a small amount of metal and make a ridge.

RALPH EARLE, Washington, D. C. (communication to the Secretary*). —Prof. Fay treats of the phenomena attendant upon the firing of large caliber guns for many rounds, that is, the fact that the surface of the

*Received Feb. 20, 1917.

metal becomes first, hard and brittle; second, cracks; and third, wears away. These conditions are termed by the Navy: cementation, heat cracks, and erosion.

A very unusual set of conditions exists in the firing of a gun. A large quantity of powder is suddenly converted into gas at a very high temperature, and the pressure of the gas rises quickly to several thousand pounds on the square inch. The effects produced by this action are several.

If steel is heated to a high temperature, and suddenly cooled, it will be hardened, and this hardening will be more pronounced with high carbon steels. A thin film of the bore of the gun is heated at each discharge of the gun and suddenly cooled, the rate of cooling being accelerated by the cold metal of the tube. It is found that after a few rounds the bore becomes quite hard. The bores of some guns, returned from proof after five rounds, are found so hard as to resist the ordinary tools used in manufacture.

Aside from the hardening due to heating and rapid cooling, steel can also be hardened by working it at a fairly low temperature. The friction of the products of combustion and the rotating band may assist in the hardening of the bore aside from the effect of rapid cooling. If the metal of the bore absorbed carbon, the additional carbon would assist in making the bore harder.

After a number of rounds have been fired from a gun, there is found to be a network of fine cracks covering the chamber in the neighborhood of the compression slope and extending down the bore. These cracks are heat cracks and are found in closed bombs as well as in guns. Various explanations have been offered to account for them. The hardening of the metal alone sets up surface strains, as well as making the metal more brittle, and in this condition it is liable to crack when subjected to high pressures. If the metal has been superficially carburized by the action of the gas these conditions are exaggerated. The strains which exist in the metal of the bore are caused by a thin layer of heated metal expanding against metal relatively colder.

The physical change in the metal of the bore assists in generating heat cracks. The physical characteristics of the steel over a thin film are changed by firing, the metal becoming much harder, with consequent reduction in ductility, and elongation, and increase of brittleness.

In the hard state, steel will rupture quickly where there are any sharp changes of direction in its surface. Hence heat cracks are most pronounced at the angles at the bottom of the rifling grooves, or on the driving edge, as Prof. Fay states it.

One explanation of the cause of heat cracks is this: At each discharge of the gun, the metal at the surface of the bore is heated to a temperature that would cause it, if free, to expand much more than it can stretch

within the elastic limit. (The coefficient of expansion of steel is 0.000075° F., and it is evident that the expansion due to 250° F. is about equal to the extension of gun steel at its elastic limit.) The free expansion being prevented by the cold outer metal, the metal at the bore is crushed. When it cools it is held in a state of tension by the outer metal. The surface more and more assumes a state of circumferential tension, and finally yields by developing cracks (heat cracks) for the most part longitudinal.

Inspection of rings cut from the bore of guns shows that the heat cracks are of little importance at the muzzle of the gun, and in general are most pronounced somewhat beyond the origin of rifling. It is likely that the greater erosion at the origin prevents the growth of the cracks to the extent to which they are found farther forward.

It is evident that heat cracks weaken the tube, but the extent of this weakness is not known. Thus far they have *not* been considered the cause of serious weakness, especially in guns of two or more layers. Investigations are still being made as to the maximum depth of cracks, their extent, and the increase of these cracks with repeated firings.

It is evident that heat cracks are a source of weakness in permitting ruptures to occur, but all guns that have been fired to any extent show heat cracks, and on the whole it cannot be said that their presence is a serious menace to the strength of the gun, especially in guns of two or more layers throughout.

Thus, the hardening of the bore is considered a cause of heat cracks rather than an important cause of erosion. Occurring as they do, principally in the grooves, they can hardly be called a serious cause of erosion in which the metal of the bore is worn away fastest on the lands.

Prof. Fay calls attention to the fact that this surface hardening consists of troostite and martensite. It is known that such structure can be caused by quenching and cold-working. For instance, the tires on locomotive driving wheels are often found pitted with martensitic structure as a result of the wheel sliding on the track due to an application of the brakes.

In the case of locomotive wheels, this structure causes hair cracks to proceed into the tire, oftentimes resulting in the cracking of such tire. The Navy Department has never heard of a case where this structure was worn away faster than the remaining surface. It does not show as pit marks, but is only noticed after microscopic examination, and shows up best after being etched. The bore of a gun is subjected to somewhat the working of a locomotive tire, and therefore the similar structure can easily be explained as cold working.

What is the reason that this hardened martensitic surface is, or should be, worn away faster than other parts of a gun? If the carbon is segregated close to the surface it would reduce the melting temperature, and

therefore permit a great amount of molten metal to be scoured away by the rushing of the gases. It is doubtful if such segregation of carbon occurs, the only excess of carbon being absorbed from the excess of carbon in the powder gas. According to analyses of powder gases, there is very little excess carbon, and certainly not sufficient to account for this theory.

As a result of erosion tests covering a number of years, it has been most noticeable that the greater amount of alloy a metal contains the greater the erosive effect of the gases upon it. Metals have been tested that contain as high as 43 per cent. of nickel and 14 per cent. of tungsten, and such metals are the worst that have been tested. Wrought iron or mild steel (0.100 carbon) have given the best results. These results might confirm the fact that martensitic structure or hardening of the surface is the cause of erosion, for it is practically impossible to turn wrought iron or mild steel into troostite or martensite.

A series of experiments is now being carried on with erosion plugs that are not quenched but are annealed above the A_{c_2} point. It is believed that such specimens being absolutely free from troostite or martensite will have less erosion than specimens that have been quenched. This point was brought forth by the fact that certain quenched specimens that had been drawn at exceedingly high temperatures showed up somewhat better than similar specimens drawn at a much lower temperature.

It appears, therefore, that the heat cracks are not of very great importance in relation to their effect upon the erosion of guns, and they have not progressed deeply enough into a forging to be considered serious as far as the strength of the gun is concerned. Before this point is reached, the erosion has generally made the gun useless.

The Navy considers erosion in guns as being of two kinds—that caused by the escape of gas around the projectile, and that due to the rush of gas behind the projectile. As both kinds manifest themselves by a fairly uniform wear of the bore, it is not possible to state their relative value.

The necessary conditions for producing erosion are the heating of a thin film of metal together with the rush of the powder products, over this heated surface. In a given time greatest erosion will occur in the gun where these two factors have their greatest erosive relation. Erosion will occur with high temperature of metal and relatively low velocity of gas. It likewise occurs with high velocity of gas and low temperature of metal, as is shown by the erosion at the muzzle; but in all cases to produce any appreciable erosion a thin film of the metal of the bore must be heated and the products of combustion must pass over this film at considerable velocity. Friction is necessary to produce erosion—there is no erosion in closed bombs in which both the pressure and the temperature are the same as in guns—there must be motion of the products of combustion. There must be a friction, or cutting, or washing away of the weakened film of metal. In addition to the friction of the products of

combustion there is the friction of the rotating band, and this is a considerable part of the total friction. The time element is of great importance, for the quantity of metal washed away increases with the time of action.

The greatest erosion occurs at the origin of rifling, where the relation between the time, temperature of the bore, and the velocity of the gas (aided by the friction of engraving the band) seems to be at a maximum for producing erosion.

The time taken by the projectile to travel one caliber down the bore is about 50 per cent. of the total time of travel from origin of rifling to the muzzle. It is evident, therefore, that as the bore near the origin is very much longer exposed to the destructive effects of the products of combustion greatest erosion is to be expected in this part of the gun.

Erosion is serious because it gradually wears away the rifling and enlarges the bore near the origin. This results in a reduction of both pressure and velocity with the ultimate result that the projectiles do not get proper rotation and hence give inaccurate flight.

Having been recognized for more than 30 years, a large number of investigations and experiments have been made in an attempt to prevent the erosion. The methods utilized and the results and conclusions reached thereby are beyond the scope of this paper, which is intended solely as a discussion of Prof. Fay's paper.

It is sufficient to remark that while the evil effects of erosion have been nullified to a great extent by the Navy, no method to prevent its occurrence has yet been developed. The design of the gun in particular reference to its powder chamber dimensions has much to do with the rapidity of erosion. One of the recent high-powered navy guns of 14 in. in caliber has already been fired one and six-tenths times the number of rounds that its recognized life permits, and does not show appreciable loss in velocity or accuracy.

In conclusion, it is to be hoped that Prof. Fay will continue his researches and eventually reach a conclusion as to the relation between the hardening of the surface and the erosion of the guns, and point out a new direction of research for the Navy to pursue in its endeavor to lessen this phenomenon.

Dry-Hot versus Cold-Wet Blast-Furnace Gas Cleaning and Some Suggestions Regarding Construction of Hot-Blast Stoves

A discussion of the papers of LINN BRADLEY, H. D. EGBERT, and W. W. STRONG, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, pp. 209 to 228.

F. H. WILLCOX, Pittsburgh, Pa. (communication to the Secretary*).
—We must keep in mind, in balancing the savings—to be anticipated by the most efficient combustion of gas, the best heat absorption by stoves or boilers, and by heat conservation of the thermal equivalent carried in blast-furnace gas—against the cost in operation, maintenance, depreciation and interest charges of the equipment required to realize the savings, that in boilers and hot-blast stoves we are basing our economies upon two different combustion phenomena.

In a boiler, evaporating water at a heat efficiency of 70 per cent., approximately 47,550 B.t.u. is consumed per boiler horsepower. Taking the blast-furnace gas cited in the above paper, we are using about $38\frac{3}{4}$ lb. of cold-washed gas per boiler horsepower. Using the hot-cleaned gas, the same equivalent heat is furnished by 34 lb. of the gas. This leaves of the $38\frac{3}{4}$ lb. required in the case of cold-washed gas, $4\frac{3}{4}$ lb. or 6,650 B.t.u. available for raising more steam when dry-hot gas is used, which means that an equivalent amount of coal firing can be dispensed with. Per ton of iron, this figure $\frac{2}{3} \times \frac{12,000}{38.75} \times 4.75 = 980$ lb. gas or 1,372,000 B.t.u. or 100 lb. coal. At \$1.50 per ton of coal, the net saving per ton of pig iron produced by the use of dry-hot over wet-cold cleaning will, therefore, be about \$0.06 $\frac{2}{3}$.

At a stove heating 8,500 lb. of air per ton iron from 150° F., one-quarter of the dry-hot cleaned gas, or 3,000 lb., would heat the air to 1,450° F.

$$3,000 \times 1,400 \times 0.70 \div (8,500 \times 0.255) = 1,300.$$

The cold-wet gas would heat the same amount of air to 1,280° F.

$$3,000 \times 1,225 \times 0.70 \div (8,500 \times 0.255) = 1,130.$$

Assuming that in each case 150° is kept as reserve for times at which the stove is not used to best advantage, we have a temperature 1,300° for hot-dry against 1,130° for wet-cold cleaning. The difference, 170°, represents $8,500 \times 0.255 \times 170 = 368,500$ B.t.u. delivered to the furnace per ton iron in excess of what would have been delivered using cold-cleaned gas. The mechanism of the saving this effects is interesting,

* Received Feb. 20, 1917.

for whereas in the case of the boiler every 13,500 B.t.u. saved and delivered in the gas to the boiler can be computed as equivalent to 13,500 B.t.u. in a pound of coal, 13,500 B.t.u. conserved and delivered in the gas to the stove delivers 70 per cent. of this to the hearth, or 9,450 B.t.u.

Instead of this saving $\frac{9,450}{13,500}$ lb. of coke, as at the boilers, we save almost four times as much. Assuming 90 per cent. fixed carbon and 82 per cent. of the carbon burned at the tuyères to carbon monoxide, then the amount of heat generated at the tuyères per pound of coke is $4,500 \times 0.90 \times 0.82 = 3,320$ B.t.u. So the 368,500 B.t.u. saved is equivalent to about 110 lb. of coke per ton iron rather than about 27 lb. coal, had the heat been used at the boiler. At \$4 per ton coke, this saving is \$0.22 per ton of iron.

I believe that the authors may have erred in assuming the dry, clean top gas to be at a temperature of 700° for the basis of their calculations, and think that 350° F. would be a fairer figure. The gas at boilers and stoves would correspond more closely to this lower temperature at the great majority of furnace plants in the country. This basis would roughly cut the above approximate savings in half, or we might expect a saving at the boilers of, say, 3 c. and at the stoves of 11 c. per ton of iron, by the use of dry-hot instead of cold-wet gas.

Under these circumstances, I question whether either cold-wet or dry-hot cleaning at boilers has any marked advantage over the use of hot uncleaned gas. At the stoves, we have an undisputed big saving which in reality and in practice may be greater than the estimate made above on account of the time and heat saved by not taking stoves off for cleaning, the elimination of the progressive decrease in heat efficiency due to bridging of the checker tops and dirt accumulations on the walls of the heating surface, and on account of the saving in labor to clean the well bottom. Further, in a stove there is an exceptionally long combustion chamber in which combustion of the gas can be effected even when the gas is at a low temperature, whereas in a boiler combustion must be completed in a relatively short length of travel, before the gas and air mixture reaches the tubes, if high efficiency is to be expected. Finally, in a stove there is no good reason to anticipate any marked excess of air because the usual stove tender is able to regulate his air inlet to give a good stack analysis, and with a modern type of stove, the stack temperature will not exceed 400° F. and may be as low as 200°. My observations lead me to believe that all good Northern and some Southern practice, and the trend of most other practice, places the temperature and moisture content of gas on the side, where, from the point of flame temperature, stack temperature at the stove and flame control, the cold-wet method of cleaning will give the best results. There are practices where magnetites comprise the burden that will undoubtedly indicate

hot-dry cleaning of gas for the stoves, but they are likely to be a small proportion of the total operations.

Cold-wet cleaning can be carried to an extreme. During the month, I have been at a plant where the wet washer delivered gas so cold and wet that the gas would not burn in the stoves. I have been at another plant where the auxiliaries of the wet washer froze up. Elaboration of the washer can be carried to an extreme, increasing interest, maintenance and operating changes to a high figure and multiplying the operating difficulties both at the washer and stoves. I believe there has been tendency toward overestimation of the difficulties of washing gas for stoves, that is, in the ability to obtain clean and dry gas from a wet washer. Efficient wet cleaning can be accomplished at moderate expense and without intricate layout and I believe wet cleaning for stoves would be the choice of the majority of furnace men, their judgment being confirmed by paragraph three of Conclusions Derived from Sensible Heat Data. Another item to be considered in the use of dry-hot gas is the deterioration of the brickwork at the base of the combustion chamber by an exceptionally high flame temperature. With the use of a good mixing burner for the cold-wet method, wet cleaning gives excellent stove practice and no deterioration of brickwork. It is to be borne in mind that should furnace practice give a top heat of 600°, the application of dry-hot cleaning will give better stove heat, lower coke and eventually lower top temperature, so that eventually conditions may be reached when wet cleaning is indicated according to the tabulations of the paper.

At the boilers, we have a slight margin of savings due to the increased heat brought to the boiler in the gas, with the additional advantage that the gas is clean. There is no question in my mind but that dry-hot is more advantageous than wet-cold cleaning for boilers. In tests I helped run some years ago, we found the utmost difficulty in burning cold-wet washed gas in the boilers without an excessive amount of air. Hot gas burns more quickly and has a shorter flame; flame control is easier and we found that the increase in boiler efficiency due to clean gas was but little better than that of uncleaned gas, not enough increase in efficiency, in fact, to offset the loss in sensible heat and cost of washing. I am sure that with hot or cold gas, some type of aspirating burner is imperative, to obtain highest efficiency, so this cost goes for any system of gas burning in boilers for uncleaned, dry-hot or wet-cold gas. The question resolves down to relative pounds of steam evaporated by equivalent amounts of uncleaned and dry-hot gas.

We know that there are blast-furnace boiler plants operating with uncleaned hot gas at 68 to 70 per cent. efficiency and up to 130 per cent. overload. It is perfectly feasible to keep boiler tubes free from flue-dust deposition and to keep the flue temperature to about 600° F. with very little air dilution. Can the dry-hot cleaning better this work at an in-

stallation and operating cost that will do better than break even with the savings due to the better work obtained?

At plants where the dust evil is actually an insurmountable handicap on account of depositions on the tubes when it cuts the iron work of the burners and flues or destroys the brick, the dry-hot cleaning should be of more marked benefit and obviously superior to wet-cold cleaning. If the cost of operation and installation of the dry-hot method is less than that of mechanical or hand soot and dust blowers, and of the deterioration of boiler and settings due to flue dust, then it is bound to be the ultimate means of handling gas preliminary to use under boilers.

The suggestions in regard to stoves are timely and are apparently in line with the developments in hot-blast stove-design practice.

THE CHAIRMAN (JOSEPH W. RICHARDS, South Bethlehem, Pa.).—The flame temperatures, as calculated by the method of Kuzell and Wigton, are not determined upon the right principle. They are probably accurate enough for relative purposes, but would not be correct absolutely. The principle on which they were calculated was inherently a faulty principle.

The amount of heat which is saved as sensible heat in the hot gases, is probably larger than is usually estimated, and a rough calculation which I have made shows it to be equivalent approximately to the useful effect of 100 to 200 lb. of coke per ton of iron. The loss depends upon the temperature of the issuing gases. Taking the temperature at 600° F., the loss represents a considerable portion of all the heat generated in the furnace. The ultimate reason of that is that the gases are 4 or 5 tons in weight per ton of iron produced, and therefore what appears like a small amount of sensible heat in the gases amounts to a relatively large proportion of the heat generated in the furnace. Another point, the fuel seldom generates in the furnace more than 50 per cent. of its calorific power, so that if you estimate it in percentage of the heat generated in the furnace instead of in percentage of the calorific power of the coke used, then it figures up nearly double what it otherwise would. It thus makes a very respectable percentage of the total heat generated in the furnace, and if a considerable fraction can be returned into the furnace, in the place where it needs it most, around the tuyères, in the form of hot blast, the saving is even intensified. I would ask Mr. Bradley whether Fig. 1 should not have some explanation on it to show the difference between the curves *A* and *A'* and *B* and *B'*.

LINN BRADLEY.—That will have to be corrected. When the paper was prepared, these curves were very fully prepared and a great deal of information given, but that has been censored and left out and it needs explanation; it is not clear just how those curves were prepared.

CHAIRMAN RICHARDS.—I agree with Mr. Willcox that 60 per cent excess of air in the stacks from the stoves is too liberal an allowance.

LINN BRADLEY.—That is too liberal on the principle of practice, but there are some furnaces where the laborers really need to be shown in a graphical way how important it is to cut down.

CHAIRMAN RICHARDS.—It is poor practice; it should not be so.

C. P. PERIN, New York, N. Y.—At one of the furnace plants with which I am familiar, we recently installed eight deep cast-iron gas washers, located in front of the gas valves of each stove.

At this furnace the ores decrepitate more or less, as does also the dolomite, and this dust has given more or less trouble in the stoves. These washers cost in the neighborhood of \$400 each, exclusive of the piping connections, and use the waste water from the stoves. They cause a drop of about 100° in the temperature of the air coming from the stove. They have not been in operation long enough to prove how valuable they are. The reduction in temperature obtainable, however, shows itself very plainly.

My present state of mind is one of extreme doubt as to the value of the elimination of that amount of dust in the stoves, if it results in cold, wet gas and consequent lower heats. A prolonged period of use may show that the diminution of loss of time on the score of dust may offset the disadvantages due to lower heats.

F. G. BREYER, Palmerton, Pa.—I heartily agree with all Mr. Bradley has said with regard to the value of hot-dry gas versus cold-wet gas. There is one point, however, on which I do not agree with the speaker, namely, that "the electrical process is unique and that it is the only known method that operates very efficiently for removing small suspended particles." I am connected with the New Jersey Zinc Co. and we have two blast furnaces making spiegeleisen from the residue from the zinc-oxide furnaces. In the residue there is always considerable zinc left behind, so that we get a very dirty gas which is partially settled out in an elaborate system of dry dust catchers. Formerly we burned this gas after passing through the dry dust-catcher system, directly under return-tubular boilers. We recently installed modern water-tube boilers and it became desirable to clean the gas more thoroughly from this standpoint as well as for the zinc recovered in the dust. The question of a proper method of cleaning came up and we decided on the wet method and finally adopted Theisen washers. At the present time I believe we are, perhaps, the only plant in the country that is scrubbing with Theisen washers, not only the gas that goes to the boilers but also that which goes to the hot-blast stoves.

I mention this because I believe zinc oxide certainly comes under Mr. Bradley's heading of "small suspended particles." The fineness of zinc fume is of a different order of magnitude from that of fine ore particles or coke dust, yet our Theisen washers are now removing that without

any preliminary scrubbing whatever. We run the gas through a large dry cooling tower, then into the Theisen washers at a temperature between 150° and 200° C., where it is scrubbed down to below 0.05 grain of dust per cubic foot; the average is 0.02 grain. We of course have no trouble at all with boilers and none with our stoves on such gas.

We have had the difficulty mentioned by one of the gentlemen present, that a gas after a wet washing gives more trouble at first than is met when burning hot gas. When we changed over from using the dry-hot gas to the wet-scrubbed gas, our stove men reported that they could not get sufficient temperature in the stoves. We found that was largely due to the fact that the stove men were judging their combustion entirely by the appearance of the flame in the lower part of the stove, and that the cold-wet gas from the scrubbers at this point looked as if it were giving a cold flame, *i.e.*, as if there were not enough gas going in. The furnace men kept crowding more gas in, and less air, in an effort to get the lower part of the combustion chamber hot; whereas, as a matter of fact, they already had a large excess of gas. As soon as we began putting the gas machine behind the stoves, admitting more air and regulating the CO₂ content of the gas, we soon got back to our proper temperatures and have had no trouble since. The stove men now have learned that it is not necessary to have the combustion chamber hot right where the gas goes in. They used to have it very hot there, through the higher initial flame temperatures and greater velocity of combustion of the hotter gas. Now the combustion is slower, the flame longer and the hottest zone well up in the combustion chamber.

To come back again to paragraph 7 of Mr. Bradley's report: Perhaps we are in a better position than is the average furnace man to make some comparison of this dry-hot versus cold-wet cleaning, on account of the fact that we have abundant experience with dry filters. We have a pretty good idea of the cost and efficiency of filtering gas through bags as compared with the cost of taking out the suspended particles with Theisen washers, and I take exception to Mr. Bradley's statement that the electrical method is the only efficient method, because, with bag houses, we get practically 100 per cent. efficiency. The only loss is a very small one on shaking, certainly less than 1 per cent., and I venture to say less than 0.5 per cent. Similarly with the Theisen washers working on a very fine fume such as ours, the efficiency is as good as any published results for electrical precipitation. From our experience with the electrical method, we have never been able to get as good recovery of fume or dust of any sort as we do with the bag house.

There have been a good many unsatisfactory experiments made, in Germany, especially, cleaning blast-furnace gas by the dry method, but I do not believe that the automatic bag house had been improved up to its present state when these experiments were made. Leaving aside

the question of explosion, which I think can be taken care of just as readily as we can take care of it on a wet-washing system, I believe that the method of cleaning blast-furnace gas dry and hot with the automatic bag house is going to be the final solution. We may not be able to keep our gas up as high as 450° or 500° F., as by the electrical method, but we certainly can get them up to 250° or possibly 300° F. We operate houses now where the gas is filtered regularly at 400 and possibly goes up as high as 500, but that would not be a regular operation because the bag consumption would be excessive.

L. E. RIDDLE.—I would like to ask Mr. Bradley if there has been any successful electrical precipitation in connection with the iron blast furnace? I understood that they have been experimenting with direct blast-furnace gas, and that moisture, precipitating with the dust, caused the precipitate to clog up very rapidly. I am much interested to know if that has been overcome, because we have furnaces in which the gas temperature is 1,000° and I would like to save that if I could.

LINN BRADLEY.—In a test of the hot process on the cleaning of blast-furnace gas, about 1,000 to 2,000 cu. ft. of gas were cleaned continuously over a period of several hours, under furnace operating conditions. The temperatures varied from 100° to 500° F., depending on the care taken for conserving the sensible heat. If the temperature gets low, the moisture will precipitate and if there is a large amount of moisture compared with the amount of dust, you get a sticky precipitate. To overcome clogging the temperature must be kept up so that the moisture will not be condensed. It is merely a question of temperature and dust, even in the presence of sulphuric acid.

CHAIRMAN RICHARDS.—The statement of Mr. Breyer that in the modern bag-house gases can be filtered up to 400° F. is extremely interesting and points to the possibility of fireproofing or heatproofing the bag materials, or using some sort of material which is a little more resistant to heat than wool, so as to filter gases at the ordinary temperatures of blast-furnace gas.

F. G. BREYER.—We have used asbestos bags successfully; the question is their relative economy as compared to the ordinary muslin bags. I think we have some asbestos bags that have lasted about 8 months to a year, may be longer than that. Still, their initial cost is very high and we do not as yet feel satisfied that we would be justified in putting them in. If the gases that we are filtering with those asbestos bags were subsequently used to heat a boiler or for some such purpose where the heat was of value to us and it was desirable to run these bag houses at a high temperature, the heat saved by running at the higher temperatures would undoubtedly justify the use of asbestos bags and I do not doubt

that asbestos bags will eventually come into use in an ordinary bag house.

C. P. PERIN.—Frazier and Chalmers, in England, are building gas washers with a coarse washing bag. It is expected that a cotton bag, treated with some fireproof preparation, will bring about an extension of the use of this type of washer. So far, they have installed a number of this type of dry gas-cleaning machinery. It is thought in England that this is the coming method for gas cleaning.

I should like to ask whether anybody here has ever tried with a blast furnace, the experiment which is now being carried on in the case of heating furnaces. In Chicago there are two continuous-heating furnaces equipped with something resembling the old-fashioned cast-iron pipe stove. The air intake passes through these "U" tubes and is admitted with the gas at the point of combustion. It has been suggested that a similar arrangement might be installed in the chimney flue back of hot-blast stoves.

CHAIRMAN RICHARDS.—A basic principle which is often sinned against in the operation of hot-blast stoves is to consider that the waste gases must be hot enough to operate the chimney. If a chimney works by the heat of the waste gases, enough heat must be supplied to operate the chimney, but that is an extremely wasteful way of drawing air through the stoves. The chimney, as a heat engine, has an efficiency of not more than 0.5 per cent. of the heat which it consumes. Therefore, if by any means you can save the heat in the gases and substitute mechanical fan draft or suction for the operation of the stoves, you may save many times as much energy as it takes to run the fan or blower. If the fan is run by a steam engine from steam raised by the boilers, the mechanical power required to run the fan, or the heat equivalent required to run it, is only a small fraction of the heat which might be saved from the gases. Therefore, a stove should be designed to take as much heat as possible from the gases. If you could make the gases go to the chimney at ordinary temperature and get the heat that is in them back into the hot blast by some means of methodic operation, you would stand to save nearly all the heat which otherwise goes up the chimney, and, besides that, you would not need a chimney.

C. P. PERIN.—We had a most interesting problem in India, in operating open-hearth furnaces. The draft was seriously affected by the climatic conditions. The period of low temperature is at 8:00 a. m., and the period of high temperature between 2:00 and 2:30; the temperature in the shade frequently rises to 120°. The air, being diathermous to the direct rays of the sun, seems to be very quickly heated and pushed up by the reflected rays, so that by 10:00 o'clock the draft stack had very much less effect than at 8:00 o'clock, and stacks that were ample in this country

required an addition of, in certain instances, 50 ft. to make them effective. In several instances, blowers were added to increase the efficiency.

In reply to Prof. Richards, I thought the question of the mechanical efficiency of a blower inducing draft had been worked out adversely, and that many engineers were of the opinion that a fan was not the cheapest way to induce draft.

CHAIRMAN RICHARDS.—My impression was the other way. We build a chimney and use an immense amount of heat to operate it; whereas, if we have mechanical power at hand, we can devise means of saving the heat which is in the gas and usefully putting it back into the furnace. There is a margin of saving which is well worth looking at.

LINN BRADLEY.—Is it a question of saving heat? Is it not a question of getting heat where you want it and at the temperature you want it, rather than that you can utilize high blast? It seems to me that is the way to work it so that you get a high blast, high temperature, as well as conserve heat.

F. G. BREYER.—The question of utilizing further the waste heat of the products of combustion and the advisability of putting in a fan, instead of using a stack and gases hot enough to give the draft, has been rather forcibly brought to our attention since we put up the Theisen washers to clean our gas for the stoves. We have installed a new type of stove in which the checkerwork and heat-absorbing surface are considerably greater than formerly. We made this change because we could afford to do it with our clean gas. We could cut down our flues to whatever we thought a reasonable sized opening and operate these stoves at the same temperature, that is at 1,200° F., as before. We find that they are so much more efficient than our old stoves that our exit flue temperatures are down as low as 200° C. We only have one stove out of nine built that way, after we get all nine of them changed and if the temperature continues around 200° C., we may be forced to use a fan to pull the gases out of our stoves. Those stoves, I might say, not only have smaller checkers, but they are also insulated with insulating brick around the outside, conforming on the whole somewhat closely to Mr. Bradley's ideal stove construction.

CHAIRMAN RICHARDS.—Perhaps I did not make clear my conception of what should be done. Suppose you have a stove with two passes that allows the gas to go to the chimney at 300° C., and you can put in an extra set of checkerwork that will reduce the temperature to 150° C., which is not enough temperature to pull the gases through the stove. Assuming that the 150° C. which you are saving goes back into the blast and increases the temperature of the blast by about the same amount, it seems to me

many times the saving may be effected by using a fan to help bring the gases out of the stoves, than the power necessary to drive the fan. If the temperature of the gases going into the stack could be reduced to the outside temperature, 60° F., a fan or suction blower would have to be used to pull the gases out of the stoves; but if one-half or three-fourths of that heat is sent back into the blast, a large saving will certainly be made.

R. J. WYSON, So. Bethlehem, Pa. (communication to the Secretary*). —In addition to the printed discussion† (of Huessener's paper) which was prepared a year ago, the following remarks are offered in discussion of the papers by Messrs. Bradley, Egbert and Strong.

The authors have made an interesting theoretical comparison of the relative merits of the dry and wet methods as applied to the cleaning of blast-furnace gas. Their suggestions, in general, are in line with actual practice and endeavor in the construction of modern hot-blast stoves.¹ Preheating the combustion air, as suggested by the authors, is in advance of present practice; for three-pass stoves this would be a more difficult proposition than for two- or four-pass units.

In a separate paper² I have stated that in this plant almost perfect cleaning of raw gas leaving the dust catcher was accomplished in a small experimental Cottrell treater. It is reasonable to suppose that time and experience will be required to develop the electrical-precipitation method on a full-size scale for blast-furnace conditions. However, there seem to be no insurmountable difficulties in the way of such development. Certainly much time and money have been expended in developing wet-washing processes, and there is still room for improvement.

In the event of the ultimate installation of large electric precipitators in connection with new blast furnaces, in plants where gas fuel is at a premium, the logical plan would be to insulate all mains, etc., leading from the furnace to boilers and stoves and from the latter back to the furnace; this would include downcomers, dust catcher, precipitators, connecting mains, boilers, stoves and hot-blast system.

From a practical operating standpoint, however, the difference in B.t.u. saving or maintenance of a slightly higher flame temperature as effected by one system of gas cleaning over the other is of less importance than the relative cleanliness of gas produced. Especially is this true in hot-blast stove practice. The glazing or destruction of brickwork, or the blocking of checker openings, due to unwashed or imperfectly washed gas, is responsible for a far greater loss in stove efficiency than is a slight difference in thermal value of the fuel gas. The maintenance of brick-

* Received Feb. 18, 1917.

† *Bulletin* No. 123 (March, 1917), 405.

¹ A. J. BOYNTON: Progress in Hot-Blast Stove Design. *Yearbook of the American Iron and Steel Institute* (1914), 76.

² R. J. WYSON: Potash as a By-product from the Blast Furnace. *Bulletin* No. 121 (January, 1917), 1.

work in stoves for regenerative purposes is destined to receive much more attention in the future than in the past. Furthermore, in some blast-furnace plants, chiefly isolated merchant furnaces, there is an excess of fuel gas available, and hence the thermal value within ordinary ranges is of little moment.

Whether the wet or dry system of blast-furnace gas cleaning for the average plant finally predominates, of course, depends upon various factors, such as first cost, operating cost, space required for units, dependability and safety of operation and cleanliness of gas produced. Other conditions being about equal, the superior system will be the one which cleans gas to an appreciably lower average dust content than the other. If there is no appreciable difference, other conditions being about equal, the balance of favor will be with the dry process.

K. HUESSENER, Pittsburgh, Pa. (communication to the Secretary*).—I agree with the authors on the main point that as far as economy is concerned dry-hot cleaning is always preferable to cold-wet cleaning. One strong point in favor of this argument has apparently, however, been overlooked; namely, that the sensible heat is all velvet to the stoves and boilers. The sensible heat is always completely utilized, because it has not to be developed by combustion as in the case of the latent heat. In other words, hot-blast furnace gas of a latent heat of 100 B.t.u. per cubic foot and a sensible heat of 5 B.t.u. per cubic foot is more valuable than cold gas of a latent heat of 105 B.t.u., without any sensible heat. In order to develop the additional 5 B.t.u. latent heat, a corresponding amount of air has to be admitted and the volume and weight of the flue gas will be correspondingly increased, so that for the same stack temperatures the heat losses in the case of the cold gas are of necessity larger than in the case of the hot gas where the weight of the flue gas is not affected by the sensible heat. If both gases are burned with an efficiency of 70 per cent. calculated on the latent heat, 75 B.t.u. are utilized in the case of the hot gas as against 73.5 B.t.u. in the case of the richer cold gas. If 75 B.t.u. per cubic foot are to be utilized under a 70 per cent. efficiency, while using cold gas, the latent heat of such cold gas would have to be 107 B.t.u. This difference increases inversely to the efficiency. At a 60 per cent. efficiency the 5 B.t.u. sensible heat are equivalent to 8 B.t.u. latent heat. If the temperature of the gas entering the washer is 500° F. and is cooled down in the washer to 100° F., approximately 8½ B.t.u. per cubic foot are lost in sensible heat. At an efficiency of 60 per cent. these 8½ B.t.u. sensible heat will correspond to over 14 B.t.u. latent heat. This [is in my opinion the strongest argument against wet cleaning.

On the other hand, I think that a comparison of hot-clean gas at

* Received Feb. 28, 1917.

temperatures of 600° F. with wet-cold gas at 60° F. favors the dry-hot cleaning and is not quite fair to the cold-wet cleaned proposition. In the first instance top temperatures of 600° F. are now always the exception rather than the rule. The average blast-furnace operator always attempts to keep the top temperatures low and if he cannot do it otherwise he will use water on top of the charge. 350° F. for hot gas appears to me to be a fairer average. As far as the cold-wet cleaned gas is concerned, very few plants are in a position to get temperatures of 60° F. even if they try to and I believe I have conclusively shown in my paper on the combustion of blast-furnace gas that the most economical temperature for cold-wet washed gas is 100° F. The comparison should, therefore, be made between hot gas of 350° F. and cold gas of 100° F. The loss in sensible heat is in that case about 5½ B.t.u. per cubic foot of average gas.

I also differ from the authors in regard to what they say about the importance of higher flame temperatures in hot-blast stoves; at any rate in so far as it is claimed that a higher initial temperature will *de facto* result in a higher blast temperature. Experience has abundantly shown that the final blast temperature is only very little dependent on the temperatures developed in the combustion chamber. These blast temperatures depend nearly exclusively on the heat storage capacity of the stove. Assuming that the checkerwork is clean, the heat storage capacity in its turn depends on the number of square feet of heating surface available per cubic foot of blast blown per minute. It has been shown time and time again that where wet-washed stove gas is used the same temperatures are continuously available, which were obtained with hot unclean gas as long as the stoves were clean.

An excellent illustration of the argument of heat-storage capacity as against combustion temperatures is given by the Larimer four-pass stove which is now in operation at the Illinois Steel Co., Joliet, Ill. In this plant cold-wet washed gas is utilized of a calorific value of not more than 94 B.t.u. The combustion temperatures rarely exceed 2,200° F. and yet blast temperatures of 1,850° F. are obtained, which is more than any furnace could probably utilize under present circumstances. A heat-balance sheet of the Larimer stove shows that only 6 per cent. of the total heat is absorbed in the combustion chamber as against 64 per cent. in the second pass. The average temperature of the combustion chamber was 2,150° F. and the average temperature in the second pass 1,445° F. which in my opinion shows conclusively that an increase in the combustion chamber temperatures of 100° F. or 150° F. could hardly, if at all, have affected the blast temperature. The difference between the combustion-chamber temperatures with hot-gas temperatures at, say, 350° F. and the cold-dry gas at 100° F. would be only a little over 100° F.

I am also of the opinion that it might have been preferable to deal

with stoves and boilers separately, as the two are really different propositions. In the case of the boilers, the combustion temperatures are the most important factor, as stack temperatures usually fall and rise inversely to the combustion temperatures. As far as boilers are concerned, the moisture in the products of combustion will seldom, if ever, reach such an amount that condensation in the exit gases could take place. Not so in the case of stoves. The Larimer stove, for which most detailed figures are available, shows that the temperature of the exit gas is usually in the neighborhood of the compression temperature of the blast. While this temperature when leaving the blowing engines will usually be in excess of 140°F. , it was shown at the Joliet plant that in a number of cases it was between 110°F. and 130°F. when reaching the stoves. Temperature charts showed that the exit gas very frequently did not reach a temperature of 140°F. until after the stove had been on gas for $\frac{3}{4}$ hr. If dry-hot washed gas was used in such a case, it would surely result in sweating taking place in the last pass if the dew point of the top gas is in the neighborhood of 130°F. This is, of course, a condition that cannot possibly be countenanced in stove practice, as it would invariably ruin the fire-brick work in the shortest possible time.

As far as the cleaning of boiler gas is concerned, it must be kept in mind that boilers do not appreciably drop in efficiency on account of the dust in the boiler gas. S. K. Varnes, Experimental Engineer of the Pennsylvania Steel Co. calculated the drop in efficiency over 6 weeks from the date of starting on the clean boiler to the time of laying off the boiler for cleaning and found it an average of $1\frac{1}{4}$ per cent. owing to increase in the temperature of the exit gas. In the case of a 500-ton furnace making $3\frac{1}{2}$ tons of steam per ton of coke and assuming the value of steam at 20 c. per ton, this drop of $1\frac{1}{4}$ per cent. in efficiency only represents \$1,600 per year to which one would have to add the wages of two men, say \$2,000, who have to blow the boilers. There is, of course, also a drop in efficiency while the boilers are being blown, but as this is not more than 1 hr. per day, the total loss is only 4 per cent. of the total operation time, so that even if the boiler efficiency dropped by 25 per cent. during blowing, it would mean a loss of only 1 per cent. on the total operation, which causes a yearly loss of \$1,280. This will show that it would require a very cheap cleaning arrangement to make it pay to wash boiler gas.

Potash as a Byproduct from the Blast Furnace

Discussion of the paper of R. J. WYSOR, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 1 to 32.

J. S. UNGER, Pittsburgh, Pa.—On page 22 reference is made to 36 carloads of dust shipped. Did the material in that last sample come from a furnace running on pig iron, ferro alloys, or spiegel?

R. J. WYSOR.—Pig iron; we have not made any ferro alloys, or spiegel at Bethlehem in the last 3 years.

J. S. UNGER.—Do you use domestic or foreign ores?

R. J. WYSOR.—In regular practice, largely foreign ores.

J. S. UNGER.—How much per unit does it cost to produce that?

R. J. WYSOR.—You might say it does not cost anything; we do not change our blast-furnace practice at all to obtain the dust. Although it has been a profitable byproduct, I would rather not have it unless we could reclaim it on a large scale, as we would have better working stoves.

J. S. UNGER.—Does the trade take it as readily as natural salts?

W. J. WYSOR.—The fertilizer dealers have been anxious to get it. This did not happen, however, until the war started.

J. S. UNGER.—They have been willing to take anything since the war started.

R. J. WYSOR.—Yes. However, we could have sold the material before the war; in fact, about 4 years ago we had a contract ready to sign, but we did not think it would pay us to bother with it then.

W. H. ROSS, Washington, D. C.—We of the Fertilizer Division of the Bureau of Soils have been particularly interested in Mr. Wysor's investigations on the possibilities of recovering potash from the blast furnace, and we look upon the results he obtained as among the most important that have yet appeared on the subject of finding new sources of American potash. About 5 years ago we undertook a corresponding investigation on the recovery of potash in the cement industry. Representative samples of raw mix and ground clinker were collected from the different cement plants in this country with a view to analyzing each sample for potash. With the data thus obtained and knowing the ratio between the raw mix and the cement produced and the output of the latter, we thought it possible to calculate approximately for each plant the quantity of potash that escapes daily from the kilns. After partly completing the work, however, it had to be abandoned for a time. It was again taken

up last fall, and has now been finally completed. We find that the potash that escapes from the kilns varies in different plants from 1 to upward of 5 lb. per barrel of cement produced, and that the total potash that escapes from all the cement plants in this country is in the neighborhood of 100,000 tons annually. The proportion of the total potash that is volatilized in different plants varies from about 25 to 85 per cent. If cement-kiln practice were standardized so that in all plants, say, 85 per cent. of the total potash were volatilized, the potash produced would approach 200,000 tons. It has been demonstrated that of the potash that escapes from the kilns at least 85 per cent. is recoverable by electrical precipitation and of that which is recovered perhaps as much as 95 per cent. may be considered as available. In at least one cement plant in this country the potash that escapes from the kilns is worth at present prices as much as, if not more, than the cement produced. The potash in the dust of some cement plants is for the most part water-soluble, but in others as much as half of the potash, as determined by the official methods, may be insoluble. In these plants part of the potash seems to undergo a recombination during the burning, with the silicates in the dust. Mr. Wysor shows that part of the potash in blast-furnace dust is also insoluble. In the case of cement dust we find that the recombined potash may be readily rendered soluble by digesting with water under somewhat increased pressure and by this procedure a more concentrated solution may be obtained in one treatment than is practical to secure directly by leaching with water in the ordinary way.

About 6 months ago samples of dust collected in the stoves and boilers of a number of blast furnaces were sent us by the Research Corporation. On analysis, the potash content of the samples was found to vary from nothing to upward of 20 per cent. It might be expected at first thought that the potash in the samples might have some relation to the total potash escaping from the furnaces. This, however, is not the case. In cement plants using the rotary kilns the potash that escapes from the kilns passes off more or less uniformly, but Mr. Wysor has shown that in the case of the blast furnace the escape of the potash takes place irregularly; consequently, the potash content of a grab sample of flue dust cannot be depended on as representing the average composition of the dust. In coöperation with the Bureau of Mines we are now undertaking an investigation designed to show as nearly as possible the average amount of potash volatilized annually at the blast furnaces of the country, and in this we are simply extending in detail the investigations which Mr. Wysor has so successfully carried out at the plant of the Bethlehem Steel Co. .

L. E. RIDDLE.—I operate a furnace on ferromanganese and ferro-silicon, and, as Mr. Wysor's paper shows, the manganese ores contain

about twice as much water-soluble potash as the other ores. I thought we could probably get a much larger percentage of potash, but we have not found more than 2.3 water-soluble potash in our ferromanganese spiegel dust. As a matter of interest, the dust from the checkers on our stoves, this is on manganese, furnishes 1.79; the dust from the combustion chambers, furnishes 1.9; the dust taken from the burners on the stoves furnishes 2.2; the dust taken from the goose necks on the stoves furnishes 1.9, and the average analysis of dust taken from the dust catcher, would show slightly under 2 per cent. and as low as 0.8 per cent.

THE CHAIRMAN (JOSEPH W. RICHARDS, South Bethlehem, Pa.).—May it not be possible that the insoluble potash may be converted into soluble potash by the methods suggested?

L. E. RIDDLE.—That might be so; we have not gone into that.

CHAIRMAN RICHARDS.—I would suggest that possibly the insoluble potash in the dust product of a furnace running at a high temperature is due to the coincident volatilization of silica, which then forms potassium silicate. That may explain the large predominance of insoluble potash in the dust of the ferrosilicon furnaces.

F. G. BREYER, Palmerton, Pa.—The American potash industry, if there is going to be one, I think owes Mr. Wysor a big debt for telling us once for all what becomes of the potash put into the blast furnace. In the course of some work at Palmerton, I had occasion to investigate some slates and shales from the hard coal region around Hazleton, Scranton and Wilkes-Barre, and I found something that I could have readily found without any analytical determination, namely, that those shales and slate carried anywhere from 2 to 4 per cent. of potash. If you will look through the Geological Survey bulletins, you will find that $3\frac{1}{2}$ per cent. is about the average potash content of shales and slates over the whole United States. Now, as a source of potash material, that looks like a very unpromising start. All the work so far on potash recovery has had to do with much richer ores than 2 or 3 per cent. Commonest among the processes that seemed promising are those involving the use of feldspar, in a blast furnace of some sort. There are plenty of feldspar samples that may run as high as 10 and 12 per cent. potash, but if a large tonnage is required the average will probably not be above 6 per cent. If this feldspar is put into the furnace together with enough limestone and coke to fuse the residue to a slag and volatilize the potash, the potash content of the charge will not be above 4 per cent. So after all, the above mentioned slate is not such a poor start as it might at first seem, particularly if one bears in mind that the slate not only has potash in it, but has enough carbon in it to fuse the whole mass down and flux the slag out of the bottom of the furnace. The only addition required is a small amount of

limestone. Three or four patents have been issued to various parties for the manufacture of potash from feldspar or its ores in which the potash is volatilized with coke. Before I make that statement, I might say that nowhere, that I know of, are the coke and the feldspar or potash-rich ores in close proximity, so either the feldspar or the coke must be hauled. For this reason potash, from any American ore we have so far developed in which coke is depended upon for volatilizing the potash, is an impossibility unless potash remains at \$500 a ton.

It was in considering the possibility of volatilizing the K_2O from this slate that both my good friend Mr. Wysor and myself found absolutely no data were available on the question of the behavior of potash in a blast furnace. Now that Mr. Wysor has published these excellent potash balances it is possible to predict with certainty what will happen to potash when charged into a blast furnace under at least several sets of very definite conditions.

The raw material I speak of is something that I think is well worthy the consideration of the furnace man when the iron trade is dull again. Here is a material that will carry 40 lb. of potash per ton. We know that an iron furnace will volatilize more at higher temperatures than lower temperatures. With a furnace working primarily for heat, and not to make iron, and without any reduction of iron ore in the neighborhood of the tuyères at all, the temperature we should maintain in such a furnace would be the limit attainable with the combustion of carbon. I feel that under those conditions a volatilization of 75 per cent. of the potash from that material is almost certain. Now, if you can charge into a 1,000-ton burden furnace in the form of slate, 40 lb. of potash per ton, on a recovery of 75 per cent. you produce 15 tons of potash a day.

One of the difficulties in the recovery of potash from furnace gases has been the matter of collecting; as Mr. Wysor's paper brings out, only 2 per cent. of the potash is actually deposited in his stoves and places where he can collect it; by far the greater percentage is caught in the first rough dry dust catcher, and a considerable percentage goes beyond. Up to the present time, the electrical precipitation seems to have offered the greatest chance of successfully recovering this very light potash fume. On the other hand, I am a firm believer that the one sure way of catching 100 per cent. of it is to use a bag house, preferably an automatic one.

A furnace as outlined above would be highly profitable at the present time, product 15 tons K_2O per day at \$300 per ton, \$4,500. What it would do after the war is a question that must be determined by experiment. For the benefit of those who are not familiar with our coal regions, I might add that there are millions of tons of this slate there and there will be millions and millions more. A blast furnace built near to some large colliery to treat this material seems to offer attractions. Blowing engines would be operated by the gas produced. Gas and slag would

be byproducts. Prof. Richards suggested that the slag might make excellent material for pillars in the mines.

We have just taken out a patent on the use of this process, but at present have no means of carrying it out to a finish. It looks well on paper, and if we can persuade somebody to try it for a couple of months, it will certainly develop some information that is possibly of great value to the whole country, because if there is anything in the proposal, there is enough potash on those dumps and enough carbon to burn them, to supply this country with all the potash it needs.

The crux of the whole thing is simply in the fact that the raw material costs nothing and both potash and fuel are present in one raw material.

CHAIRMAN RICHARDS.—I would like to ask Mr. Breyer if that were a bituminous shale, whether there would not be other valuable byproducts from the gas in the furnace?

F. G. BREYER.—There is that possibility in the bituminous shales. On the other hand, they would offer some difficulty in a dry dust-catching system. I would like to have the opinion of some of the blast-furnace men as to what they think of charging 1,000 tons of this material into a blast furnace that would produce roughly 700 tons of slag.

H. A. HUSTON, New York, N. Y.—I was impressed with the statement that the raw material would not cost anything. We have had that same expression about the cost in making alcohol out of sawdust, but somehow or other, when we went at the sawdust, it did cost something. I have grave doubts about any kind of ore being of any commercial importance which carries less than 10 per cent., or 8 per cent. perhaps, of water-soluble potash in its original condition, because of the freight and handling charges. All proposals for the fusing of feldspars or other material to make commercial potash must provide for writing off the entire plant before the shipment of foreign potash begins. These war prices are exceedingly interesting and attractive, but unfortunately there is no considerable market for it at war prices. One cargo would break the market without any trouble. We must take that into consideration. If you could write this entire plant off, as the plant at Hoffman has been written off, then the thing would be very profitable, but it takes a long time to pay for a building and plant. We have been wrangling about producing American potash since 1909, and the Hoffman plant is the only one that has done very much along that line in a profitable manner. That produces a 28 per cent. potash mixture which is profitable at the present time, although it is alkaline. The trade wants a neutral potash. I was also interested in the paper on the recovery at cement plants. The standardization of cement plants would operate them in such a way as to volatilize a greater percentage of potash. It

is interesting from a chemical engineering standpoint, and one of my friends undertook to look into that and visited one of these plants that was volatilizing so much and found they were getting a high potash recovery, but some of the cement was coming back. They had destroyed their cement in trying to make potash. He decided that so far as his company was concerned, they would not put that on their plant.

F. G. BREYER.—With regard to the value of this product, it has been lying there ever since the mining industry began. There is absolutely no use for it at the present time, as it is a decided disadvantage to the coal miner, due to the fact that it gives him a heavy overburden when he tries to mine his coal directly beneath those piles. He has to leave a good deal more coal under there than otherwise. Eventually those lands will become farming lands, then what is going to become of those piles? I know that it can be bought for less than 5 c., but I think, as a matter of fact, that the cost of taking it from the head frame out on the dump would be gladly turned over to any company that wanted to take it. The only question would be the disposal of slag, since there would be almost as much slag as original tailings, but, as Prof. Richards suggested, there is a very good use for that in the mines. And take such a case as the fire in the Mammoth vein at Lansford, where a big barrier was made out of cement at tremendous cost. That fire is still going on; they are attempting to put it out in a number of different ways. There is a possibility that it could be choked off with a slag barrier. But leaving the question of slag value out of it entirely, the process is independent of anything else but potash values, on the assumption that you are going to have enough gas to operate your blowing engine, which is perfectly logical, and that you will have some gas to sell to the neighboring boiler houses which is likewise self-evident.

H. A. HUSTON.—Do you suppose you can produce potash for 40 c. a unit by that process?

F. G. BREYER.—That will depend entirely on the way the furnace works on this burden. The physical nature of the material charged has a great deal to do with it, and that is in our favor, because the material on those dumps is largely in good-sized pieces so that it at least ought not to choke up the furnace. There is only one way to go about it and that is to haul some of it to the nearest blast furnace and run it for a couple of months and find out how much potash is left in the slag. If 75 per cent. is driven out, it can be caught; we have caught it from blast furnaces even after it has been burned in the stoves. We caught a material from an iron furnace in the Lehigh Valley that averaged 14 per cent. of water-soluble potash, and we caught 100 per cent. of it economically from the gas leaving the stoves.

W. H. ROSS.—With regard to Mr. Huston's reference to cement manufacture, I might say that the manager of one of the cement plants which has introduced the Cottrell process for precipitating the dust, told us that since the precipitator was installed, they are able to manufacture their cement more efficiently than they did before, instead of less efficiently.

CHAIRMAN RICHARDS.—As far as the possibility of smelting that material in the blast furnace is concerned, it is only a question as to its chemical composition. If it contains enough carbon to reduce by smelting, and is not too high in magnesia, it can be fluxed. There is no doubt, from the blast-furnace end, that it can be smelted; it is nothing much more than a slagging gas producer.

G. A. ROUSH.—This paper interested me from the statistical standpoint particularly. Mr. Wysor's balance sheet shows 22.4 lb. of potash entering the furnace per ton of charge. At the present rate of making pig iron in this country, that is equivalent to almost 400,000 tons of potash per year, assuming that the charge was uniform for the entire country, which, of course, it is not, not to mention something over 600,000 tons of soda, both in the form of oxide. Of this only a small proportion was actually caught. On the basis of the small proportion caught, in this one plant in the course of a few months, the recovery would amount to something like 15,000 tons for the entire blast-furnace capacity of the country; and this evidence, in conjunction with the manifest possibility of increasing the recovery, makes a rather promising commercial proposition out of the operation. Even accepting Mr. Wysor's caution that this charge at Bethlehem carries potash above the average for the country, there is still left a considerable amount of potash to draw on. The imports of potash from Germany in the year ending July 1, 1914, were approximately 250,000 tons, and in the year following that, only a little over 200,000 tons. The latter figure was considerably reduced due to war conditions, and later importations are much smaller still. This leaves us then with a normal annual importation of about 250,000 tons as a figure for comparison with the possible savings from the blast-furnace operation, providing means can be provided for the securing of a greater saving of the potash carried over from the furnace.

Then, Mr. Wysor's second caution as to the cost of extraction he seemingly puts in as a caution to not take things as coming too easy, but does not point out some of the advantages on the other side. He says it is going to cost more in order to get a greater recovery, which is true; but in spite of this extra cost we have the promise of methods for producing greater recoveries, particularly dry cleaning, and this sort of a process is not only going to give you a better yield of recovered product, but, as pointed out by Mr. Bradley's earlier paper at this session, it is going to

give you the advantage in the operation of the blast furnace of the hot-dry blast, which can be counted as an additional advantage in favor of the collection of the product.

H. A. HUSTON.—The maximum importation to this country in any year was never more than 260,000 short tons of actual potash. About 1,125,000 tons of potash salts are imported. I think a good many of our people have been misled in this country about the consumption of potash by some curious statistics that appear which are all very high. The chemical consumption in this country is not over 11,000 tons of actual potash.

F. G. BREYER.—I do not think the question of past consumption concerns us so much as the question of future consumption, and 200,000 tons year before last is not the question; the question is what we are going to import 10 years from now. If we could produce it at any sort of reasonable figure, 200,000 tons would not be more than a fifth of the use, because our farmers are not using the potash they should.

WALTER A. SCHMIDT, Los Angeles, Cal. (written discussion).—Mr. Wysor brings out the interesting consideration that a heavy or dense fume interferes with the proper combustion of blast-furnace gases, particularly when used under the boilers. He makes the suggestion that this results from the breaking up of the continuity of the gases causing a large increase in the distance over which the flame must be propagated. He also says that an exceedingly large number of finely divided fume particles may prevent proper contact of the gas molecules and thus prevent proper flame propagation, but it would hardly seem likely that such a fine state of division of solids could be reached as to interfere with molecular contact.

I wish to suggest that which to me seems an important factor in the phenomenon discussed by Mr. Wysor; namely, the effect of screened radiation. Any solid will absorb a portion of the radiant energy impinging upon its surface, dark bodies absorbing more than light ones. If a particle absorbs heat waves, it then itself becomes a center of radiation, but being of a lower temperature than the source of the primary radiation, it establishes proportionately a much feebler radiation in any particular direction. An incandescent body or a flame will have its free radiation greatly checked or retarded if surrounded by a screen of such solid particles. This can easily be demonstrated by simply inserting a wire-mesh screen between an intensely hot flame and one's face, this instantly causing a cessation of the burning sensation. I have often used this well-known means of protection when observing operations within cement kilns and certain metallurgical furnaces. Comparatively coarse wire cloth, for example 10-mesh, is quite sufficient to screen the intense radiation yet at the same time permit of ready observation through the

openings of the fabric. This is not a phenomenon of heat absorption by the metal of the wire cloth, as the shielding action continues indefinitely.

My thoughts in connection with Mr. Wysor's disclosure is that the particles composing the fine fume shield the radiation of the flame and prevent its propagation to any great distance from the center of ignition. Under certain conditions this action could probably be sufficiently pronounced to extinguish the flame altogether. Mr. Wysor's statement that the phenomenon under discussion is less noticeable in the stoves than under the boilers is significant, as the incandescent checkerwork in a stove undoubtedly decreases the importance of radiation from the center of ignition as a means for maintaining proper flame propagation.

I offer this suggestion for what it may be worth, being in total ignorance of first-hand information on iron blast-furnace operation.

F. G. BREYER.—I cannot agree with that explanation, since it does not seem to correspond to the facts in a similar case. We can burn zinc vapor readily to a very dense fume containing 10 to 20 times as many solid microscopic particles per unit of volume as blast-furnace gas, and therefore with that much greater screening action than offered by the particles in the case of the dirty blast-furnace gas. My opinion as to the cause of it is that it is inherent in the potash itself. The fume when it comes into the boiler is relatively cool and the potash is there as a solid. If you suddenly generate heat and vaporize the potassium chloride, the latent heat of vaporization is way above that merely required to heat up solid particles to the temperature of the products of combustion, which heat absorption may sufficiently retard the flame propagation to cause the effect noted.

J. S. UNGER, Pittsburgh, Pa.—Mr. Wysor has presented a paper on a comparatively new subject, which is not only very interesting, but of great importance as to the whole country as well as the iron and steel industry, as it furnishes a new source for potash.

The shortage in the potash supply has led to the investigation of a great many materials as a possible source of potash, and it is a question to which we have given considerable attention, our investigations covering various waste products from six of our plants with a view to recovering the potash or using the waste product itself as a fertilizer.

Our studies have covered the determination of the potash in blast-furnace slag, water from slag-granulating pits, blast-furnace flue dust, deposits in blast-furnace boiler combustion chambers, in flues leading from the boilers to stack and deposits in the stack, deposits in the hot-blast stove combustion chambers and the dust accumulations in the last passes of stoves or flues near stack, open-hearth furnace slag and the deposits in the regenerator chambers of open-hearth furnaces.

Our blast furnaces operate principally on a burden of Lake Superior

ores, Ohio and Pennsylvania limestone and Connellsville coke, and the amount of potash in our various waste products would naturally differ somewhat from that found at Eastern blast furnaces which are run on an entirely different burden.

We found that our blast-furnace flue dust carried only a few tenths of a per cent. of potash; the blast-furnace slag, about $\frac{1}{2}$ per cent.; water from slag-granulating pits, traces; deposits in blast-furnace boiler combustion chambers, about $\frac{1}{2}$ per cent.; and deposits from the last passes of hot-blast stoves, from 1 to $1\frac{1}{2}$ per cent. In a general way, the potash we found in the various materials was either insoluble or, in the most favorable cases, about one-third water-soluble.

Considering our results as a whole and the amount of the various products produced, a sufficiently high percentage of potash was not found to make it desirable to attempt to recover the potash from such materials.

Sometimes the potash is fixed, passing through the furnace, and is found in the slag, at other times it appears to be easily volatilized and is found in the largest quantities in such portions of the furnace as are comparatively cool, such as the gas flues, regenerator chambers or stack. Occasionally we have found in locations quite distant from the furnace a few ounces of a grayish or reddish fume very high in potash, but the quantity is too small to be considered. As an example, we found 58 per cent. of potash in a fume adhering to a blast-furnace stove chimney valve, 21 per cent. on the brick near the stack in a blast-furnace boiler, 20 per cent. in a fume in a boiler stack, 43 per cent. in a deposit on the piston of a gas engine, and 7 per cent. on the vanes of a Theisen gas washer. These figures indicate that the potash is carried for quite a distance to a cool spot before most of it is deposited.

From Mr. Wysor's work, it appears that there would probably be a larger yield of potash from furnaces working on certain alloys, and we are at present making determinations of the amount of potash in various deposits around furnaces working on ferrosilicon and ferromanganese.

R. J. WYSOR.—I think I can account for the difference in practice as found by Mr. Unger and Mr. Riddle. As noted in my paper, the western blast-furnace burdens are lower in alkali content than those at Bethlehem, ours being an exception, and in the second place, their hot-blast and top temperatures are lower than ours. These temperatures largely determine the proportion of alkalis discharged from the cinder notch, and from the top of the furnace. I believe, Dr. Unger, you found more potash in your slags, than we did at Bethlehem.

J. S. UNGER.—Half of 1 per cent.

R. J. WYSOR.—We found less than 0.4 per cent., as an average in the slag, with our burden richer in potash. Furthermore, if your investigation of the potash collected at the various points mentioned was made on

dry gas, I can state positively that most of the alkalis escaping from the furnace passed out through the stove and boiler stacks. If you investigated washed gas, then most of the alkalis were carried out by the wash water in the primary washers. The more perfect the performance of the primary washers, the less potash-bearing dust will be recovered from stove and boiler passes.

CHAIRMAN RICHARDS.—Enough attention probably has not been paid to the function of the vapor tension in the volatilization of these constituents. We know very little about the vapor tension of these salts, but the question as to whether they go out at the top or come out at the slag notch is largely a question of the temperature at the top; the vapor of the salt is carried as vapor, at a very small partial tension, up the furnace, and if the top temperature is low, you carry out proportionately less; you cannot carry more than a very small amount.

In the combustion of the gas in the stoves, is it not possible that the enormous number of small particles which Mr. Wysor speaks about in his paper act as such efficient radiators of heat that they check the development of a high temperature in the gas. They may radiate the heat so efficiently and with such enormous radiating surface that they check the rise in temperature and keep down the actual temperature of the gas.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 111, March, 1918, pp. 627 to 650.

THE CHAIRMAN (JOSEPH W. RICHARDS, South Bethlehem, Pa.).—The record of the 26 furnaces given by Mr. Howland shows conclusively that there is nearly a constant amount of carbon used up in direct reduction above the tuyères of the blast furnace. It cannot be accidental that the amount of carbon consumed above the tuyères in those 26 furnaces varies very little, from the furnace using the most coke to the furnace using the least. As far as these furnaces are concerned, Grüner's ideal working does not apply. There is a constant amount of carbon burned *above* the tuyères, and then all the rest of the carbon which is gasified is burned *at* the tuyères, because that is the only place it can be burned. As far as American furnaces are concerned, I think this conclusion simplifies considerably the discussion of the reduction of iron ore in the furnace.

Notes on Flotation—1916

Discussion of the paper of J. M. CALLOW, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 245 to 275.

H. A. MEGRAW, New York, N. Y.—One interesting thing that might be brought out in this connection is the cost of making sulphide gas. I had a communication the other day which suggested that, in view of the high cost of oil, sulphur and sodium sulphide, we might make use of a well-known chemical reaction to form this sulphide gas by the use of aluminum sulphide. It is difficult to see any advantage in this, however, since aluminum has been affected just as much as any other commodity by the rise in prices, due to war conditions.

RUDOLF GAHL, Miami, Ariz. (communication to the Secretary*).—Mr. Callow refers to my paper, *History of Flotation at Inspiration*, which was presented at the Arizona meeting of the Institute. These notes of Mr. Callow's contain much valuable information on flotation practice, as is to be expected from an article written by an author of Mr. Callow's experience in this branch of metallurgy. Nevertheless, I cannot help contradicting a number of the statements made in the references to my paper.

In passing, I will also say a word about the reference in Mr. Callow's paper to the article by Messrs. Laist and Wiggin entitled "Flotation Concentration at Anaconda, Mont." Mr. Callow calls attention to the fact that evidently Callow flotation machines are not very sensitive, inasmuch as one man per shift handles four 800-ton sections of Callow machines or 3,200 tons per day at the Inspiration plant. While it has been the experience of the Inspiration company that the four sections of Callow machines in the Inspiration concentrator are easily handled by one American operator per shift, it is only fair to state that each man operating four sections is assisted by two Mexican helpers who constantly clean the bottoms of the machines. The tonnage per section, by the way, whether equipped with Callow, Inspiration or Minerals Separation machines, is considerably in excess of 800 tons. Mr. Callow further says that local opinion at Inspiration is that two 800-ton sections consisting of two 10-compartment and four 6-compartment Minerals Separation machines (these machines are of the Hebbard type) is the limit that one man can properly attend. I do not hesitate to say that I consider it perfectly feasible to have four such sections operated by one man, perhaps assisted by one helper. The Minerals Separation machines of this type have undoubtedly an advantage as far as easiness of operation is concerned,

* Received Feb. 19, 1917.

because no porous bottoms have to be taken care of. If Mr. Callow would point out that with neutral pulp the blanket cells require less repair and require fewer shutdowns, I would readily agree with him.

Mr. Callow mentions that when his machines were installed the tailings obtained in the Inspiration test mill contained 0.5 to 0.6 per cent. copper, the concentrates from 23 to 25 per cent. copper and that in the course of some 8 or 9 months of continuous competition his results showed a general tailing of from 0.25 to 0.35 copper and a concentrate of 30 per cent. and better.

I have discussed the difficulties that arose when the attempt was made to install the flotation process at Inspiration on a quasi-working scale, and how they were overcome. Besides the introduction of more suitable machinery, that is, flotation machines utilizing injected air, several other changes had to be made which contributed very heavily toward the improvement obtained.

Mr. Callow further says that a summary of 2 months' results, November and December, 1914, showed a recovery of 2 lb. copper per ton in excess of the best results of the other competitors. It may be fair to state here who the other competitors were. They were the Standard Style Minerals Separation machine in November, which was intentionally overloaded to establish the point that a tonnage representing the rated capacity is an overload and a Minerals Separation machine designated as "Subaeration machine" in December which, as I pointed out in my paper, was soon abandoned and replaced by a simpler and better type of machine. No differences of the magnitude quoted by Mr. Callow resulted when improved machines had taken the place of the two Minerals Separation machines just mentioned, neither one way nor the other. In the later contests the differences in the results obtained by machines of different designs were very small indeed. I will not in any way deny that when Mr. Callow, upon the invitation of the Inspiration company installed a number of his cells in the Inspiration test plant and showed that they effected a much better recovery than the Standard Minerals Separation machine, he rendered a very important service which in my opinion certainly justified the company in buying the rights for the use of his apparatus for the treatment of a limited tonnage of ore.

Mr. Callow claims that the blankets of the Inspiration flotation machine require more attention and more frequent renewals than the blankets of the Callow machines. In reply I beg to state that the same number of men are employed per Callow section as per Inspiration section for the purpose of keeping the blankets in good condition and that there are no figures showing that there is any difference in favor of the Callow machine as far as renewals are concerned as long as the comparison is made under the same conditions. I wish to point out that the conditions under which the Callow blankets in the Callow sections at Inspira-

tion work are somewhat different from those met in the Inspiration machines. In the Inspiration sections the whole feed passes through one machine, sand and slime mixed. In the Callow sections there are primary cells receiving the regular mill feed and secondary cells treating slime only. Only the primary cells of the Callow sections treat the same feed as the Inspiration sections. In the experience of the company there is no difference in the wearing of the blankets in the primary Callow machines as compared to the blankets in the Inspiration machines. The blankets in the secondary Callow machines treating only slime feed, as mentioned before, wear somewhat better. The item of the blanket upkeep, is however, so small in both types of machines that it deserves no serious consideration.

Another statement of Mr. Callow's is that with a feed containing a larger percentage of sand than the Inspiration feed now contains, Inspiration machines would be unworkable. I beg to reply that very often in daily operation the Inspiration machines have received pulp containing more oversize than they had originally been designed for. Frequently the average of the whole mill has been 4 per cent. on a 48-mesh, which means that some of the sections carried 5 and 6 per cent. on an average of a whole day, and a great deal more for shorter periods. I want to say, furthermore, that retreatment of table middlings in flotation machines of the Inspiration type has been adopted, that is, of a feed containing sand alone and no slime, or at least very little of it, and that no difficulties whatever are found in treating such a feed. We have, furthermore, considered the retreatment of all of our table tails, that is a product which is practically all sand, on flotation machines of the Inspiration type and have, in order to satisfy ourselves that we would not find mechanical difficulties in treating a product of this nature, made tests regarding this point. We have found none.

Another of Mr. Callow's statements is that the larger unit machine, that is, the Inspiration machine, is a positive disadvantage, since when removing blankets 400 tons of capacity has to be lost or diverted, whereas with a small subdivision of units as in the Callow cells only 50 tons or less has to be diverted. I wish to reply to Mr. Callow that when Inspiration machines were installed, care was taken of this point and an air box designed which could be removed from the flotation machine while the same was in operation, so that Mr. Callow's statement that 400 tons' capacity has to be lost or diverted is incorrect. As I pointed out in my paper, however, we soon found that our flotation machines stand overloading very well and that it was not a great detriment to temporarily switch the load from one flotation machine to another. Since that time we have made it a practice to divert the feed when repairs are made. It is a little easier to make repairs on an empty machine than it is with a machine running. I want to assure Mr. Callow, however, that when-

ever we want to we can make arrangements to change the flotation blankets without diverting any feed, and I am inclined to think, therefore, that the disadvantage which Mr. Callow attributes to the larger machine does not exist, but that the machine presents the advantage of not requiring any diversion of feed, should any one using it desire to change bottoms in this manner. It has the additional advantage of having a solid bottom, thereby obviating leaks, and of not requiring calking when changing air chambers. Metallurgically speaking, our experience is that the Inspiration machines work just as well as the Callow machines, if not better. I consider it a decided constructional advantage that it requires materially less mill height than any other air-bottom flotation machine that I know of, for instance, the Callow machine. Mr. Callow admits the advantage of reduced mill height in his reply to Mr. Cole contained in the article under discussion, although he modifies his statement by maintaining that this consideration is quite secondary to recovery attendance and power.

To Mr. Callow's further assertion that "inasmuch as all the essential points of the Inspiration flotation machine were anticipated and, moreover, reduced to practice, prior to the installation of any flotation at Inspiration, it was a Callow machine in every essential," I want to reply that to decide this question it is necessary to establish the characteristics of the Callow machine. If the Callow machine were the original pneumatic flotation machine, I would admit the validity of this argument, but as far as I know it is not. As I pointed out in my paper, the application of a porous bottom for flotation work had been patented in England to Minerals Separation Ltd. much earlier, and the use of this principle could not, therefore, constitute a peculiar characteristic of the Callow machine. The Callow machines that were demonstrated to the Inspiration Consolidated Copper Co., in addition to the characteristic of a porous bottom, contained the further characteristic that the bottom was inclined. This naturally cannot be considered as a discovery, as all the surfaces over which ore passes in any concentrating mill are inclined. Furthermore, the cells contained a float valve each for the regulation of the pulp level in the cells. I presume that neither this nor the combination of these two characteristics will be considered a discovery by Mr. Callow. From my viewpoint, Mr. Callow, contemporaneously with some others, rendered a very important service to American metallurgy, the importance of which cannot easily be exaggerated, when they showed that the Minerals Separation Co. had neglected the development of one type of their machines at the expense of another. To call the type of a flotation machine which they originated, but neglected to put into practical operation in America and even to patent in America, a Callow machine is, I believe, not justified.

Concerning Mr. Callow's priority statement, it is my impression that

when the first flotation machine was installed at Inspiration or the first pneumatic flotation machine was patented, there was no such a thing as a Callow cell.

The Inspiration flotation machine makes use of the principle that ore pulp somewhat of the character of the Inspiration pulp, that is pulp containing some oversize on 48-mesh, can be passed over a horizontal or nearly horizontal bottom by proper arrangements of baffles. This may not be much of a discovery, still it seems to me that it contains the elements of invention to a certain extent, inasmuch as everybody at the first glance is of the opinion that baffles would have the effect of retarding the flow of the pulp. This is the only point that is claimed as new in the construction of the machine outside of the air chamber, which I think has points of originality and the advantage that it can be removed without diverting the flow of feed through the machine. I do not hesitate to say that the machine utilizes the principle patented in England to Hoover and the Minerals Separation Co. As far as I can see, it does not embody any invention of Mr. Callow's. We use, however, the same kind of a porous medium, that is, canvas.

Mr. Callow also calls attention to the use of hydrogen sulphide to assist in the recovery of copper carbonates and silicates as practiced by the Magma Copper Co. and offers this process as a partial answer to the question of treating such ores, which I discussed in my paper. Mr. Callow deserves great credit for describing this application of the flotation process which, as his figures show, gives encouraging results. As he knows himself, it is only a partial answer and the Inspiration company has been fortunate enough to receive several partial answers lately. The company has before it the dilemma, however, of deciding which partial answer is the best, and is holding out in the meantime for a final answer.

G. D. VAN ARSDALE, New York, N. Y.—If we divide the practical operation of flotation into three steps:

1. Oil mixing,
2. Aeration,
3. Separation of froth and pulp,

then ideally the machine arrangement should be such that in step 1 no particle of mineral can escape coming into contact with oil, and in step 2 that each particle of mineral, presumably coated with oil, shall have sufficient opportunity to come into contact with an air bubble. Recoveries will accordingly be lowered by a lessening of efficiency of either of these operations, and costs will be raised if the apparatus used for either purpose is less efficient mechanically than it should be.

It seems sufficiently obvious, as stated by Mr. Callow, that blowing finely divided air through a pulp is mechanically a much better way of

mixing air and pulp than agitation with a paddle. It would not seem impossible, however, to devise a machine in which the same contact of finely divided air and pulp could be effected without the necessity of blowing the air through such a depth of pulp, which of course means power consumption.

Mr. Callow says, in comparing the so-called "Inspiration machine" with his: "The larger unit machine is a positive disadvantage since when renewing blankets 400 tons has to be lost or diverted, whereas on a smaller subdivision of units only 50 tons or less has to be diverted." This disadvantage does not seem so obvious to me.

If the diversion or loss is expressed in terms of cost per ton treated, and if we have two machines in parallel, one of 50 tons and the other of 400 tons capacity, and if we assume that the blankets on both are renewed at the same time, then it is true that during the time of shutdown eight times more pulp has to be diverted from one machine than from the other, but during the operation period the larger machine will have treated eight times as much pulp. Therefore, the loss, if expressed per ton treated, will be exactly the same in both cases. In other words, there will be a disadvantage of the larger machine only if the tonnage cost per blanket renewal is greater for it than for the smaller machine.

If the cost of renewal of a blanket in the larger unit is eight times the cost of renewal in the smaller, then the disadvantage will be proportional to the greater frequency of renewal, but if the cost of renewal is less than eight times the larger blanket can still be renewed more frequently at an equal or even a less cost per ton.

Mr. Callow's note on the advantage from the use of copper sulphate is important. It has been known that very minute amounts of certain addition agents may exercise either a deleterious or favorable action on flotation. Practically it has been found that a substance whose action is very favorable on the flotation of some ores may exactly reverse this on others. For example, sodium sulphide is in use in some mills and has the result of increasing the recovery. The advantage from its use seemed so obvious that a quantity was ordered for use in a southwestern mill. Very unexpectedly it was found that the addition of even very small amounts to the flotation feed at this mill had the effect of producing a white froth carrying no mineral whatever. This result is the more curious because at another mill it has been found that considerable benefit is derived from the use of small amounts of caustic soda.

I believe therefore that we must divide ores for flotation into three classes:

1. Those requiring acidity of pulp.
2. Those requiring alkalinity of pulp.
3. Those requiring neutrality of pulp.

There seems no doubt that this varying action is to be referred mainly to the influence of the constituents of the gangue.

We have been investigating this question for some time, using for the purpose artificial ores made up of mixtures of as nearly pure minerals as possible, with various gangue minerals. When this work is finished and is supplemented by tests on actual ores of varying gangue constituents we can then correlate extractions with gangue constituents and possibly find a way to eliminate the influence of some of those that are found injurious. Definite results have already been obtained but are not as yet sufficiently complete for publication.

This brings up another very important practical point, which so far as I know has not been thoroughly investigated, namely the composition of mill waters and the effect of dissolved salts on flotation.

In cyanide mill practice we find in many cases greatly reduced efficiency from "foul" solutions due to the gradual accumulation of impurities by solution from the ores, and there seems no reason why the same thing may not occur in flotation practice. In our own work we have had unexplained differences between the results of test mill and operating mill practice, and an investigation is in progress to determine the possibility of this being the correct solution.

Mr. Callow's description of the Magma sulphidization work is very interesting. We have also found in our work that H_2S does not interfere with flotation except when it is present as free H_2S .

We have a simple way of eliminating this, and our flow sheet is different in some details from Mr. Callow's. We have obtained very good extractions on partly oxidized copper slime tailings but find it impossible to make a high-grade concentrate in one operation although we can do so by a recleaning cell.

Our method of generating H_2S contemplates the use of low-grade copper matte and acid, which for our conditions I believe will be cheaper than the Magma method.

Metallurgically, sulphidization by H_2S and flotation seems entirely practical, and apparently its use will be decided on the question of the advisability of working with large amounts of a very poisonous gas, together with the relative advantages and disadvantages of other alternative methods for partly oxidized ores.

To my mind the most important conclusion in Mr. Callow's paper is that pointed out on the last page. Flotation is the resultant of an extremely complicated set of conditions, all of which are important, and while gravity concentration may in many cases be safely left to the "millman," it will in most cases pay very well to have the flotation end of a mill under the direct control of a competent flotation metallurgist, whose time will be fully occupied.

Finally, the profession is always indebted to the man who will take

time to write a summary and review like the paper under discussion, and I think there is no doubt that flotation is still in such an unsettled state that it will be a wise prophet who can accurately forecast from our present knowledge either the final forms of method or apparatus, or even the final field or extent of its use.

E. R. RAMSEY, Denver, Colo. (written discussion).—Under the heading of "Handling Flotation Concentrates," Mr. Callow says that his experience has not been satisfactory with the continuous method of settling or dewatering the concentrates before filtration, and that he has for that reason adopted the intermittent system in his more recent installations.

The continuous system, to which reference is made, and which is in use in practically all of the large flotation plants, involves the use of the Dorr continuous thickener. The flotation froth with sufficient water to carry it through the launders is fed to the thickener in the usual way and the thickened concentrate drawn off at the bottom continuously into the filter, which is in most cases the revolving drum type.

Flotation concentrates are very difficult to settle, particularly copper concentrates, and require a very large settling area compared with that of the gangue material from which they have been separated. For copper sulphide concentrates the average of a number of plants would show about 40 sq. ft. of settling area required per ton of concentrate per 24 hr. In some instances this is considerably less and in others it will run over 50 sq. ft. For zinc concentrates this area is considerably less and will average in the neighborhood of 12 to 15 sq. ft. per ton. The character of the original ore itself, particularly of the colloids, as well as the quality and quantity of oils used, seems to have an important effect on the settlement of the concentrate, and it is usually found that the froth collecting on top of the thickener tank will carry an appreciably greater quantity of gangue than the average concentrate.

At a few plants some difficulty has been encountered in the handling of the froth which accumulates on the top of the thickeners. However, I do not feel that this difficulty can be attributed to the continuous system itself, but rather that it is due to crowding the thickener capacity. In every case that has come under my observation, where the thickener was being fed beyond its capacity with the resulting dirty overflow, difficulty was encountered with the froth building up and tending to work over the sides of the tank and into the overflow launder. As against this, where the tank is being fed at or below capacity and a clear overflow is obtained, very little, if any, difficulty is experienced with the froth, as it usually builds up to a certain extent, dries out on top and apparently breaks down from underneath as fast as it accumulates. An explanation for the froth accumulation when a dirty overflow was being ob-

tained, would be that the fine particles of concentrate rising to the top of the tank attach themselves to the bottom of the froth layer and gradually build up faster than they disintegrate.

At the Magma plant, to which reference is made, the Dorr thickeners were operated in series, which arrangement experience at other plants has conclusively shown is inefficient compared with parallel settling. A comparison of this method of thickener operation with the intermittent system as later installed, shows that the latter had a definite advantage in settling area, and consequently, capacity.

At some other plants the original thickener installation was insufficient for the tonnage of concentrate to be handled, but in a majority of cases the tonnage of flotation concentrate has been increased beyond the capacity of the tank without providing additional settling area. The tray thickener, as developed by the Dorr company in the past year, offers a most satisfactory means of obtaining a large settling area with a small amount of floor space. The tray is of proven merit in settling slime, giving the full 100 per cent. increase in capacity where properly installed, and makes a flexible and compact installation.

The first installation of a tray for handling flotation concentrate was recently made at the plant of the Old Dominion Copper Mining & Smelting Co., and the latest advices from this plant indicate that the thickener and tray are working at full capacity without any indications of overload. This installation consists of a 40-ft. tank with a single tray of the connected type. Settling tests indicated an area of 40 sq. ft. per ton and a discharge product of 57 per cent. solids. Present feed consists of about 608 tons per day at 8 per cent. solids. Approximately 63 per cent. solids is being obtained in the discharge. The overflows from both tank and tray are clear, and by the use of paddle arms and sprays on the surface of the tray compartment no trouble is experienced with froth accumulation. This method of handling the froth is said to be very effective.

A number of schemes for handling the flotation froth are being tried which promise satisfactory results. With the froth taken care of, the continuous system should offer the most satisfactory means for thickening concentrates in that the installation can be made with the minimum of floor space and head room, particularly where the tray is used; would require less attention than the intermittent system; less horsepower, although this is not of great importance; and would give as thick, if not thicker, discharge, with reasonable attention given to its regulation.

There has been considerable discussion as to whether the continuous settler would give as thick an average product as could be obtained from the intermittent tank. The Dorr company in the past year has made experiments on a large number of different slimes and concentrates, which show that stirring near the end of the thickening period will give a thicker

final product than can be obtained with undisturbed settling. No samples have failed to give this result and in some cases the difference is very marked. The same result has been shown in actual operation. In practice the Dorr thickener provides this stirring with the plows, in raking the thickened material to the central discharge, and should, therefore, with a proper regulation of the discharged product, give a greater density than in intermittent settlement.

At the plant of the Timber Butte Milling Co., the zinc flotation concentrate is settled in Dorr thickeners to approximately 70 per cent. solids, and in the same district the Butte & Superior and the Anaconda Copper Mining Co. are obtaining from 60 to 65 per cent. solids in their underflows on zinc concentrate. On copper flotation concentrates the density of discharge will average somewhat less, the Inspiration and Anaconda thickener plants giving around 60 per cent. solids. At the Arthur plant of the Utah Copper Co. one 75-ft. thickener has a capacity of over 200 tons of flotation concentrate per day, giving a clear overflow and a thickened product averaging about 60 per cent. solids. The concentrate thickener discharges at Chino and Ray also average about 60 per cent. solids. The area required for lead flotation concentrate is about the same as for zinc, namely, 12 to 15 sq. ft. per ton. The St. Joseph Lead Co. at three of its plants in Missouri obtains about 65 per cent. solids in the discharge from the thickeners on lead concentrate.

On flotation concentrate where a large settling area is required, as a rule very little depth is necessary for thickening, so that a number of trays could be installed in one tank without an excessive depth. In view of the marked economy which can be made in initial and operating costs and in space required, it is probable that future installations will be made along this line.

E. E. FREE, Baltimore, Md.—I am not competent to discuss at all the operating details in Mr. Callow's paper or indeed anything that has to do with the practical use of the flotation process, since I have had no experience whatever with that end of the matter. The thing which I do have to say I say very diffidently and tentatively. It has to do entirely with the theory of the flotation process. So far as it relates to Mr. Callow's paper at all, it is to the incidental observation of the assistance sometimes obtained by the use of copper sulphate in the solution.

In order to explain what I am about, let me assume that the essential thing in flotation is the attachment between the mineral and the oil. Other things of course are necessary—the froth formation, etc.—but the essential thing about the flotation process is that a given oil will attach to certain minerals in the ore mixture and not to the other minerals. This is commonly ascribed to what is called "interfacial tension,"

which means nothing whatever. All that we really mean is that the oil adheres to some minerals and does not adhere to other minerals.

Now there is a whole series of experimental results which are closely related to this sort of thing, namely, the experiments on what is called adsorption. At the present time there are two theories as to the cause of adsorption. First, there is a physical theory which imagines the adsorption as the same sort of condensation as would be produced by increase of pressure on a gas. For instance, if one imagines a gas condensed on the surfaces of charcoal particles by extreme pressure, one has the action of the charcoal in adsorbing the gas, which is actually observed. The second theory ascribes adsorption to a quasi-chemical union between the adsorbed substances and the material that is adsorbing it.

The adhesion which is the essence of flotation is closely related to adsorption and probably due to the same causes,¹ but that does not get us very far practically, and the thing I particularly want to speak of is not the adherence between the oil and the mineral but the effect which might be produced on this adherence by the previous adsorption on the mineral surfaces of films of some other substance. All mill waters carry small amounts of dissolved substances. The particles of ore and gangue are ground in, or mixed with, this water before coming into contact with the flotation oil. Undoubtedly they adsorb out of this mill water, dissolved substances which are then held in the form of adsorbed films on the surfaces of the solid particles. They might, for instance, acquire adsorbed films of sodium chloride from the mill water.

It is probable that this previous adsorption of dissolved substances on the surfaces of mineral particles will have important effects on the tendency to adherence between the mineral and the oil and therefore on the possibility of flotation. The adsorbed films may have detrimental effects, or may have beneficial effects, depending upon whether the different minerals are made more alike or more different in their tendency to adhere to the oil.

It is known that these adsorbed films are affected importantly by many different things, especially by the presence in the solution of small quantities of other dissolved substances, and it seems to me that in this fact there lies a possible explanation of the effects of copper sulphate and similar substances. If copper sulphate in the mill water has an effect in displacing or modifying in some way the films of adsorbed substances formed on the particles of ore or gangue, then the copper sulphate may affect importantly the subsequent tendency of the ore or gangue to adhere to the flotation oil. There are similar implications in connection with the small chemical differences between different flotation oils which differences seem to have such great practical effects. Perhaps the

¹ See Bancroft: *Metallurgical and Chemical Engineering* (June 1, 1916), 14, 631-635.

effects of these small differences may depend upon an action of some sort on the previously formed adsorbed films.

Essentially, all that I have to suggest is that the adsorption of substances on the particles of the ore or gangue prior to the application of the flotation oil may have much importance to the working of the process, perhaps even more importance than the properties of the mineral and the oil themselves. It seems not improbable that some of the surprising anomalies which, I understand, are encountered in working the flotation process may be related to something of this kind.

J. W. BELL, Montreal, Que.—I have been very much interested in the remarks of the last speaker, and would like to ask him if he thinks that adsorption may have an important bearing on the fact that at times the use of an acid seems essential to secure good flotation, or in fact to get any flotation. Of a number of ores on which we have made laboratory experiments, in every case the use of sulphuric acid was necessary either to get any froth or to get a clean froth. The information we have just received suggests the possibility that when the acid is added to an otherwise non- or difficultly floatable ore, the adsorbed material may be dissolved and thus permit the adherence of the oil to the mineral particles.

E. E. FREE.—It is very difficult to give a categorical answer to this question. There are no generalizations with regard to adsorption which I regard as sufficiently well established to be willing to state them. So far as one can be sure at present, every case of adsorption must be considered individually. However, it appears to be true that the concentration of the hydrogen ion usually has an important effect upon all of the adsorption phenomena occurring in the solution. Accordingly, one would expect that the presence of sulphuric acid would disturb importantly the adsorptive behavior of all substances present. I do not believe that it is possible to predict in advance whether addition of sulphuric acid would be beneficial or detrimental in any special case.

Incidentally, I do not believe that it is quite correct to speak of sulphuric acid as "dissolving" an adsorbed film. Whatever adsorption may be it is not the same thing as, for instance, the coating of a particle of oxide with a sulphide film. Such coating probably involves a chemical change which penetrates a distance of some molecules into the particle. If that film is removed by acid an appreciable portion of the particle is removed. The ordinary case of adsorption does not seem to be analogous to this and it is better, it seems to me, to speak of sulphuric acid or other substances as "displacing" an adsorption film rather than as "dissolving" it. According to Langmuir's theory² the adsorption film consists of a layer one molecule thick of one substance held on another substance by a

² Irving Langmuir: *Journal of the American Chemical Society* (November, 1916), 38, 2221-2295.

quasi-chemical attraction. A third substance, for instance sulphuric acid, might displace this film and replace it with a layer of hydrogen ions or of sulphuric acid molecules or of SO_4 ions or of something else. Of course, all of this is very tentative and theoretical. As I said in the beginning, I do not think it possible to make any generalizations from which one could predict, without experiment, the behavior of any particular case.

H. W. DuBois, Philadelphia, Pa.—I have been at several of these flotation law suits and I think it is one of the most unreasonable things to ask judges, even the most expert in the interpretation of patent law, to pass intelligently upon such subjects without a technical advisor, such as is provided in English Courts. The subject is so highly technical, that the mere use of obscure terminology is used with evident success in clouding the subject in such a way that I do not see how it is possible for us to look to our Courts, constituted as they are, for a really fair interpretation of the patent situation. The legal situation is quite similar to trying to play a game of baseball with an umpire who had never seen the game. You might instruct a man effectively in baseball umpire law during the progress of a game but I doubt if the ordinary baseball fan would like to be compelled to accept his decisions; I doubt if it is possible for the judges of even our highest courts to sufficiently inform themselves upon such a complicated subject, so that they can intelligently pass upon it.

There is one point I would like information upon and that is, is it universally true that the best recoveries are only obtained when less than 1 per cent. of oil is employed? In my own experience I have found that much more than 1 per cent. of oil can be used with certain ores, without affecting the recovery. It is doubtless true that some classes of ores give better results with less than 1 per cent. oil, employing a certain prescribed method of treatment, but other methods of treatment may allow variations in quantities of oil that can be employed, in order to obtain the same recovery. The so-called critical amounts of oil below 1 per cent. in amount, in which such effective results are claimed, is not true with all classes of ores, as far as my own experience goes.

My problem of concentration has been very unusual, owing to high transportation costs, in which I have been required to concentrate a 5 per cent. copper ore which is mostly bornite. As is well known, bornite usually has an oxidation of the surface which in some cases seems to interfere with an effective coating of the oil, thus diminishing the flotation effect. This tarnish probably acts like the non-metallic surface of the ordinary gangue minerals. We tried a good many experiments and found that no matter how we varied the character or amount of oil, it did not seem to give any satisfactory recovery. Fortunately the

gangue was calcite and small additions of sulphuric acid immediately produced sufficient CO_2 bubbles to float effectively the particles which were not raised by the air bubbles.

G. D. VAN ARSDALE.—In the work which I have done I have not been able to find that a "critical" point exists at 1 per cent. of oil, and I do not see any theoretical reason why there should be a "critical" point at this, or in fact at any, percentage of oil.

The difficulties which Mr. DuBois speaks of in the use of more than 1 per cent. of oil in my opinion may be perhaps obviated by a different apparatus construction or by different details of manipulation so that with an amount of oil greater than 1 per cent. under certain conditions it will be possible to get fully as good or even better results than with less than 1 per cent.

THE CHAIRMAN (E. P. MATHEWSON, Toronto, Ont., Can.).—One of these experts came to us during the experimental stage at Anaconda, and said that he could float our minerals. We put him in charge of the first commercially sized unit we had and for 3 weeks he labored with all his lore and experience, but his results were abominable. He was on the point of giving up when either he had an inspiration or some member of the staff suggested to him that he try a little sulphuric acid. Then the adsorption was removed and everything went smoothly.

Another thing, as to the amount of oil being exactly 1 per cent., I know that there were some very excellent concentrates produced when considerably over that amount of oil was used but, of course, the company that I was with, having made an arrangement with the patentees of the process, got on the economical side as quickly as possible and found how little oil we could get along with. We used considerably less than 1 per cent.

J. W. BELL.—The question raised by Mr. DuBois is at least of scientific interest and may be very important in its practical aspect. I recall one laboratory experiment in which nearly 2 per cent. of oil was used to secure a good result. In the concentration of an ore, consisting of calcite, barytes and chalcocite, by tabling, it was found that only about a 25 per cent. copper concentrate could be made. Practically all of the barytes was contained in the concentrate, and it was of interest to devise means for raising the grade of the concentrate by eliminating the heavy spar. By strongly heating the concentrate, the decrepitation of the barytes permitted removing much of it by screening but the separation could be made much more effectively by flotation. By using 38 lb. of wood tar oil, it was found that 94 per cent. of the copper in the original concentrate could be obtained in a flotation concentrate containing approximately 50 per cent. copper.

Mr. DuBois's last experiment reminds me of some recent experiments I made to see whether it would be possible to float the sulphides in the Porcupine ores. As is well known at the Hollinger mill, table concentration is used to separate the sulphides from the gangue and as the ore is ground to 200-mesh, before concentration, it seems reasonable to suppose that a large part of the mineral must go into the tailings. Just as a matter of scientific interest, therefore, a few tests were made on ore similar to the Hollinger ore to see if a good separation could be made by flotation. It was found to be impossible to float the sulphides in a neutral solution, but it so happened that before the oil was put in, the acid was added and it was observed then that a splendid flotation was made. Very clean concentrate was floated to the surface of the spitzkasten, but of course the froth was extremely weak and to have removed the concentrate it would have been necessary to use quite a heavy overflow.

The test described merely illustrates the flotation of sulphides by CO_2 gas as in the Potter-Delprat process, but it is possible that the effectiveness of the use of acid in an oil flotation process is partly due to the liberation of CO_2 , which seems to be an extraordinarily effective gas in the flotation of mineral particles.

H. W. DuBois.—Japan (I am informed) has passed a law by which no flotation patents will be allowed, solely in the interests of the best conservation of their natural resources. That is interesting, if true. I also am informed (and this, I think, is reliable information) that applications, with the identical wording of the fundamental flotation patents which were granted to the Minerals Separation company in the United States, were submitted previous to the U. S. applications, to the German government. After an investigation, the German patent office decided that there was nothing new in these claims and therefore patents were not allowed.

It is a matter of undoubtedly a good deal more importance to us as mining men to know, if these flotations patents are going to be sustained by our highest courts, whether the practice in the past of demanding excessive royalties is to be maintained. In some countries, if a man has a patent and wants royalties which the industry considers unreasonable, those effected can go to the government and get an adjudication as to some reasonable rate of royalties. That is not true in our country; but I think we should take, as mining men generally, a little more interest in the subject to prevent statements which are not true from being quoted as truth before our highest courts, and apparently accepted by them with such confidence that decisions are being rendered along such lines as are going to greatly burden the mining industry, on account of the unreasonable royalties demanded by the parties having their patents sustained by the courts.

Anthracite Stripping

Discussion of the paper of J. B. WARRINER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 33 to 60.

S. A. TAYLOR, Pittsburgh, Pa.—What was the distance of haul?

J. B. WARRINER.—There is practically no limit to the haul. The length is determined entirely by the feasible location for a dump. I have known of hauls up to 2 miles in length.

R. V. NORRIS, Wilkes-Barre, Pa.—This paper was originally presented to the Anthracite Section as the result of the appointment of Mr. Warriner as the Chairman of the committee to study this matter and present this paper. As he has stated, he obtained data very freely from all the stripping operations in the region, and it is a type of paper of which we should have more in the *Transactions*. Two or three things in it strike me as particularly valuable.

The two tables, 1 and 2, showing the coal remaining, have been the result of a great amount of study, as it is not an easy matter to obtain data as to the amount of coal left in mining. As an example, one of the anthracite mines was mined and robbed out in 1880, so the maps show. That mine is today producing 250,000 tons of coal, over 50 per cent. from the bed which on the maps is marked as exhausted. What actually happened is that the seam was on a light pitch inconvenient to mine. There was ample thickness of coal and the chambers were driven at a convenient pitch through this 30-ft. bed, leaving top and bottom, taking out about 10 ft. The pillars were removed and the working caved and marked as gone.

That is the type of area that is now being stripped, which explains the reason for the enormous amount of coal left in, and the fact that Mr. Warriner has worked out the actual results from a large number of stripings is of great interest to the operators in any region handling thick coal which has been mined over in the past generation. The modern mining, of course, is different. We do not have all the coal we want in big veins, and we do not mine it that way.

Another interesting feature of the paper to me was the study of losses. It is not uncommon to assume that after you have stripped off the overburden all the coal goes to market. Mr. Warriner has shown that there is a loss ranging up to 10 or 15 per cent. that does not go, so that estimates made on stripping, figuring on all the coal, are subject to this very material correction.

The use of planes has not been given the weight in the paper that the history of stripping in the anthracite region would seem to warrant. The great majority of the stripings have been in the very narrow, sharp

basins of the Hazleton district, where it was practically impossible to get out graded tracks, and the greater part of the stripping has been done by planing. Overburden is removed by shovels and then pulled up on wire rope planes. The plane has a further advantage in that the coal is loaded directly into the mine car and taken into the original car to the breaker, so that there is no dumping between loading and final delivery. Every time anthracite coal is dropped the resulting breakage means more small sizes and therefore a less average price for the shipped material.

The tendency, of course, is to larger machinery and larger operations. It may be seen that Mr. Warriner has not given the cost per ton of coal. That is a figure which has but little value, and is generally unobtainable, at any rate until a stripping is completed, because the cost of moving overburden is confused with the total cost until the stripping is absolutely completed and there is no coal left. Further, the amount of overburden removed and the cost per ton is dependent upon the price received for the coal, because stripping will be continued as long as profitable and no longer, except in the few cases indicated, where stripping is a matter of safety for the mine or for the prevention of getting rock or waste into the mine.

GEORGE S. RICE, Washington, D. C.—I notice that special reference has been made to the stripping operations in the district where recovery of pillars in old mines has been attempted. I notice, however, that the author speaks of virgin fields, and in one or two illustrations, as for instance the last one on page 9, shows some illustrations of apparently unmined beds. I was wondering whether or not the factor of drainage of surface waters would afterward make the open excavations a detriment of any serious nature to future underground operations, so that perhaps it might be wise not to begin the stripping until after underground mining has ceased. My reason for that inquiry is because I have been in certain bituminous districts with pitching beds where the stripping of the outcrop forming canal-like excavations had later proved to be detrimental in subsequent underground mining operations down the dip. The water seeping through pillars and running down and into the mines caused continuous pumping operations.

J. B. WARRINER.—That point, though a very important one, is not dwelt on particularly in the paper for the reason that in the anthracite regions early mining operations failed to take drainage into consideration at all. The early openings were entirely water-level openings and the workings were pushed through to the surface and water allowed to run in and find its own way out. Most all our strippings are of areas so worked over in earlier days, and, therefore, the stripping does not increase the drainage area materially. Even where the stripping is of virgin veins, there are usually other veins in the vicinity that have been worked through to the

surface sufficiently to form an inlet for all drainage water, so that even such strippings do not materially increase the amount of water to be handled. It is regrettable that in the early mining operations such methods were used, as a chain pillar left under the surface, or even all the water-level coal left in place, would have greatly reduced the difficulties of present-day operation.

HARRISON SOUDER, Cornwall, Pa.—This paper is extremely interesting to me, as our company is doing some work in the same line. Our business is mining iron ore but we have a problem of moving something like 3,000,000 yd. of stripping, practically all rock. I think the figures given here are quite valuable. I would like to ask Mr. Warriner about the operating hours. He gives the average monthly yardage here. Does that mean working one 10-hr. shift daily, or day and night?

J. B. WARRINER.—On contract stripping work, one shift only of 9 hr. is worked daily.

H. SOUDER.—Name some tentative figures.

J. B. WARRINER.—The rate of 45 c. mentioned is higher than the usual rate for rock on a classified basis. By a classified basis I refer to a contract that specifies one rate for clay excavation and another for rock excavation. The percentage of rock to clay in each stripping operation determines the average rate, or the unclassified rate, per yard.

H. SOUDER.—That puts the price pretty low, it seems to me.

J. B. WARRINER.—Those are the rates that prevail in the anthracite region. We have stated the limits to be between 35 and 40 c. for rock, but these are only the average limits. There are some few exceptions; both higher and lower, that occur.

H. SOUDER.—Your suggestion of heavy equipment, I think, is in line with present practice not only up in the Mesabi country but in our own operations. We originally started with small, 65-ton shovels and now are putting in 100-ton shovels; started with a 5-yd. dump car and now have 20-yd. air dump cars. I notice you mention the height of the benches. That is a very important item in shovel work and we started originally with 16-ft. benches and are gradually working them up and now have adopted 40-ft. benches as the proper working height in rock.

J. B. WARRINER.—The difficulty of that is blasting the rock so as to break it small enough to be handled by an 80-ton shovel.

H. SOUDER.—That is where the large equipment comes in.

J. B. WARRINER.—In a 40-ft. bench the holes required for blasting

must be nearly the same depth; naturally, therefore, only the lower part of the bench is broken up at all finely. The surface rock will be found in immense boulders after the blast and a great deal of labor and powder must be expended in redrilling and shooting these boulders. For that reason our contractors have come to consider the lower bench as more nearly ideal.

H. SOUDER.—We find that to be the case sometimes. Since the introduction of the jackhammer and larger equipment we have increased the output of a shovel I think something like 20 per cent.

J. B. WARRINER.—Our contractors also use jackhammers.

H. SOUDER.—Our high bench makes for fewer moveups and bigger loads.

J. B. WARRINER.—That is the usual argument in favor of the higher bench.

H. SOUDER.—Those are things which have been tried out in our operations at Cornwall and which have proven to be correct. Of course the tonnage moved at a fair cost is the thing that counts, in the long run.

ROBERT PEELE, New York, N. Y.—When Mr. Norris spoke of the relation between overlying and wholly or partly worked-out veins and subsequent stripping operations, my thoughts reverted to a case in the bituminous regions that I saw some years ago, and although not quite pertinent to this subject I thought it might elicit some further discussion of the paper.

A seam lying horizontally and $8\frac{1}{2}$ to 9 ft. thick was being worked. It was overlaid by five other seams, included in a total height of 100 or 120 ft., and ranging in thickness from about $3\frac{1}{2}$ to 5 ft. The mine was on the borders of the Pocahontas field, lying close to the line between West Virginia and Virginia. In working the main seam, the roof was allowed to cave, with the result that all the other seams were probably ruined so far as any future recovery of coal was concerned.

This case occurred to me also in connection with the remark made by Mr. Warriner himself, regarding the stripping of a series of seams, the upper ones of which had already been wholly or in part exhausted. It would seem as though the large tonnage of coal included in these thinner seams would be absolutely lost in the subsidence and breaking up of the overlying strata, in the caving following the working out of the main 9-ft. seam.

J. B. WARRINER.—The point brought up by Mr. Peele is not a matter of particular importance in those sections of the anthracite region where stripping is mainly resorted to. The veins in those sections are highly inclined and can be mined to a large extent without disturbing the over-

lying veins, at least to the extent that they would be disturbed in flatter measures particularly in the bituminous region. In certain sections of the anthracite region, however, it is very important that the overlying veins should be worked off first before the underlying veins are mined at all and even in the regions of highly inclined strata some importance must also be attached to this point.

THE CHAIRMAN (R. M. CATLIN, Franklin Furnace, N. J.).—Increase of water due to the stripping has been touched on but I did not understand that it was considered much of a factor from the discussion here. Is that correct?

J. B. WARRINER.—That is correct, because the watershed area is not increased appreciably by stripping for the reason that the previous, or early-day mining operations have already breached the surface to such an extent that stripping an area does not add much to the watershed.

CHAIRMAN CATLIN.—I could imagine that in some cases where the surface had settled you might be able to divert a good deal of the surface water falling on that watershed.

J. B. WARRINER.—That is true, and that is done in some cases, but it has not been an important point with us. The quantities of water handled in anthracite mining are enormous and I know of few instances where stripping in any area has greatly increased the total quantity to be handled by the mine.

CHAIRMAN CATLIN.—Its chief danger is due to sudden water falls, for instance a very severe thunderstorm will crowd an enormous amount of water there temporarily.

J. B. WARRINER.—Considerable money has been spent in constructing ditches outside the outcrops of our veins to prevent surface water in flood times from flowing into the crop falls or surface breaches. The same is also done with stripping areas where feasible.

T. M. DODSON, Bethlehem, Pa.—I have had some experience in stripping a large virgin vein—average thickness 65 to 75 ft. The depression caused by mining this vein would be so great that its effect, so far as to render impossible the drainage of the area, would be equivalent to the stripping process. In both instances the surface water would have to go into the mines. Of course, wherever possible, we ditched the periphery of the stripping and endeavored to prevent any outside water flowing into the pit.

H. M. CRANKSHAW, Hazleton, Pa.—How far can the large shovels which are being used in the soft coal region be applied to mining and stripping in the anthracite regions? Some of these shovels now have 90-ft. booms, a 5-yd. dipper and can make a pile of refuse over 60 ft.

high. It seems to me if these shovels go on increasing in size the anthracite stripping of the future might not have nearly so much transportation to contend with as it has now.

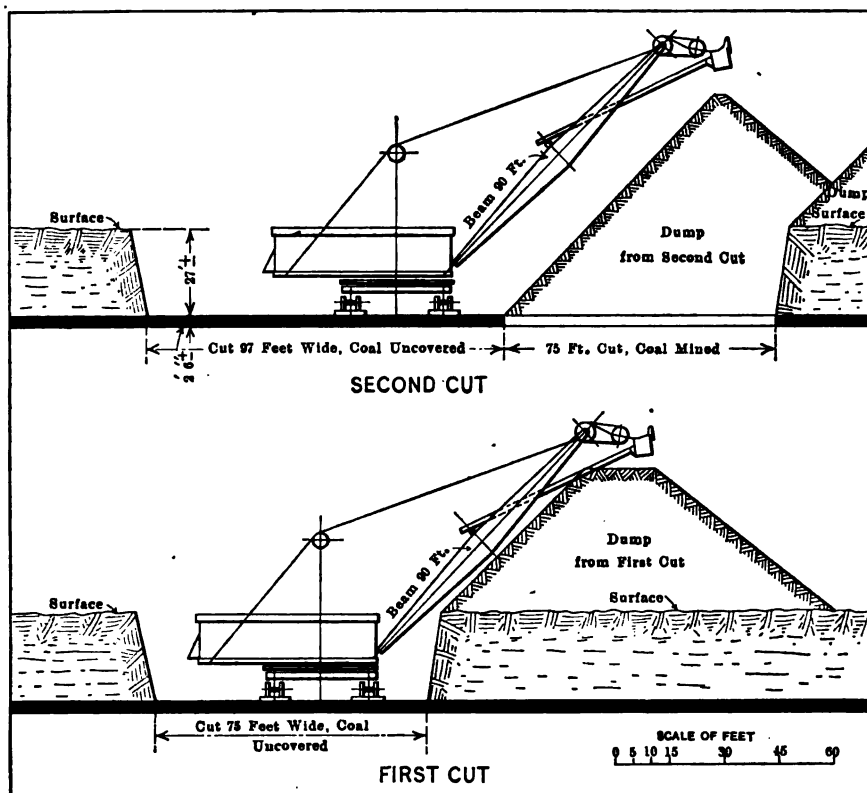
J. B. WARRINER.—If you will turn to Fig. 9 you will see that that is what is being done where the 250-ton drag-line excavator that I speak of is being used. The drag line in that instance was able to make its dump outside of the stripping limits, as is shown very plainly in the illustration. The width of the basin being stripped is such that a drag-line excavator of this size can handle it in two cuts, each cut extending from one outside limit to the center and one bank being thrown off on one side and the other on the other side. When the stripping area is wider than is shown it can readily be seen that this cannot be done unless the coal is removed first, as is done in certain cases, I understand, in the Kansas field. If the coal is removed from an area that has already been stripped, then the spoil or the refuse material from the next cuts can be dumped into the area from which the coal has been removed, and in that way, by mining directly behind the stripping, it has been possible to strip very large areas in Middle Western practice, at a low cost.

T. M. CHANCE, Philadelphia, Pa.—The Pittsburgh region in Kansas, to which Mr. Warriner has just referred, was the first place, I understand, where they used really large self-stacking shovels. The machines in use about four years ago were not drag-line machines; they were simply shovels with a 5-yard dipper carried on a boom some 90 ft. long and stripping a cut 70 to 100 ft. wide. The coal, at the locality which I have in mind, was about 2 ft. 6 in. thick and the over-burden around 27 ft. in depth. This over-burden was composed of partially decomposed sedimentary shales and sandstone, in no way similar to the hard sandstones and slates encountered in anthracite stripping.

The method of operating the stripping was to take a preliminary cut of any desired length and from 70 to 90 ft. in width, placing the spoil from this at the side of the pit and necessarily covering some coal that could not be recovered. The coal uncovered by this preliminary cut was then loaded up and a space left for the spoil from the next cut, approximately as shown in the accompanying sketch. The shovel was operated on a pair of double tracks, about 29 ft. gage from outside rail to outside rail, this track being laid on top of the coal. By operating the shovel in this way the difficulty of an unstable track foundation in times of wet weather was largely obviated, as a good floor was always furnished to carry the load of the shovel and its equipment.

The costs of this method of stripping are exceedingly low, as no extensive transportation is required for the spoil resulting in removing the over-burden. On a basis of 27 ft. of stripping to recover 2 ft. 6 in. of coal I understand a cost as low as 5 c. was easily reached and could be

considered conservative. The principal difficulty encountered in this field was in loading the coal after it had been stripped by the shovel. If this work were done by hand it should not have cost much over 35 c. a ton to load, but a cost such as this would hardly be comparable to the low unit cost reached in the stripping of the over-burden. I understand



that small power shovels were tried for lowering this loading cost but certain difficulties were encountered in their operation simultaneously with that of the large stripping shovel, and I believe that a form of drag-line loader was eventually worked out that largely eliminated these difficulties and greatly lessened the final cost of handling the coal itself.

J. B. WARRINER.—The point I have tried to make is that drag-line machines cannot be used except under favorable conditions such as the one described in the paper or possibly where the stripping is of large enough area to make it profitable to remove the first cut only by this method. Such a machine, however, can be equipped with a boom and used as a shovel. That has not been tried out at all, but I believe it is contemplated, and I make the point that probably it will be tried out and

probably it will be successful. It must be remembered that in our strip-pings the depth is variable and successive vertical cuts must be made. Most stripping pits have been excavated to a depth of 60 ft., and some even to 150 ft., from the original surface to the top of the coal.

THE CHAIRMAN.—Where would the spoil go in such cases?

J. B. WARRINER.—The spoil would have to be loaded onto cars and hauled out.

T. M. CHANCE.—Then you lose the advantage of a 90-ft. boom.

J. B. WARRINER.—Yes, you lose that advantage, but I think the big dipper would be the chief advantage. You would still have the transportation problem to contend with in any anthracite stripping where those machines are used, except in a few cases where they can be used as the one described is being used. The machine described in this paper is carried on wooden rollers.

A. O. IHLSENG, Joplin, Mo.—Reference has been made to the large 5-yd. shovels used in the Pittsburgh, Kan., field. The Marion Steam Shovel Co. has a 7-yd. and the Bucyrus Co. has an 8-yd. dipper operating in this field making a 40-ft. cut. I think they have been bringing the cost of mining the 3 ft. of coal underlying it down to a very small figure. With the large over-burden of 13 yd. of strip. per ton of coal mined, the stripping cost must be less than 5 c. a yard.

T. M. CHANCE.—Five cents a yard is a fair, safe estimate, with a 5-yd. dipper.

J. B. WARRINER.—The Geological Survey Statistician gives 10 yd. of cover to 1 ton of coal as the maximum result for 1915.

A. O. IHLSENG.—This 8-yd. dipper was put in operation in March, 1916.

J. B. WARRINER.—Even 10 yd. to 1 ton is more than double the results obtained in the anthracite region.

H. M. CHANCE, Philadelphia, Pa.—It is hardly fair discussion to compare methods of work done under conditions that are quite unlike those in the anthracite regions.

Conditions in the Kansas field, where more of this kind of work has been done than in any other district, are peculiar to that field and to some small areas in the Western or Central Western coal fields, in that the formation overlying coal is usually a rather soft slate or shale with interbedded sandstones. The sandstones are true sandstones but are generally more or less disintegrated. I have seen one of these Marion shovels cut right through what looked like 2 ft. of solid sandstone. No shovel could do that in sandstones such as we have in the anthracite region. The cost figures are not applicable to conditions such as are present in

the anthracite field. In these Western strippings it is customary at times to shake up the stuff with a few holes when the country rock is relatively hard but in many cases they take off the whole country rock covering without shooting.

In Kansas the coal is relatively thin, $2\frac{1}{2}$ or $3\frac{1}{2}$ ft. thick—very rarely 4 ft. thick—and that coal is worth, as a crude proposition, we will say 80 or 90 c. or \$1 a ton at the outside. In the anthracite region the coal may be worth about \$3 per ton in its crude form, so that in the anthracite region the material to be mined is worth perhaps three times as much as Kansas coal and it requires perhaps three, four or five times the kinetic energy to break up that rock and remove it as a stripping proposition.

It seems to me that it would seldom be possible to adopt the suggestion of Prof. Peele to attack bituminous coals by this method with the idea of working several coal beds that lie one above the other. I assume Prof. Peele had in mind the idea of stripping right from the top of a hill and taking vein after vein, thus winning the coal from several veins. That proposition has been suggested before but there are very few places where it could successfully be used for the reason that Nature has not generally placed our coal beds close enough together, the workable beds usually being separated by intervals of 30 to 60 or more ft.

If we adopt 10 cu. yd. as the maximum quantity of waste material that can be moved for each ton of coal recovered, that would require a coal bed 3 or more ft. thick to every 30 ft. of rock and there are few places where coal beds of this thickness are so close together with the top bed close to the surface.

W. S. AYRES, Hazleton, Pa.—I quite agree with Mr. Chance that we cannot, from the data given, get at the relative merits of the different systems of stripping referred to in this discussion. We have not taken into account all of the elements which enter into the problem. Not only the cost per cubic yard, but every possible phase that enters into the cost should be set forth. The nature of the material, ranging from soft drift material not requiring any blasting, to that which is very hard, such as sandstone or conglomerate rock, and which requires the most extensive use of powder, should be clearly and definitely stated. The length and system of haulage, the system of dumping, and whether the material when dumped does or does not adhere more or less to the bottom and sides of the car, should be stated. A very important feature is the thickness and the quality of the vein of coal, and also the market value of the coal recovered. Without all of the facts before us we cannot get at an intelligent comparison, nor can we select from the systems described the one that will suit our particular problem best.

Years ago—15 or more—we had at Hazleton, Pa., a stripping which was considered at that time a very large one. The cross-section of it was almost identical with the cross-section of the Panama Canal at Gold

Hill, and it was so deep—300 ft., the coal being 80 ft. thick at the bottom—that it was possible to remove a part of the rock by the milling system. We ran one section of the stripping by the steam-shovel system, using seven shovels, and the other by the milling system. The cost per yard by the milling system was 5 c. less than by the steam-shovel system. The milling system cost 17 c. and the steam-shovel system 23 c. per yard. We used at that time the 1½- and the 2-yd. dippers, which seemed to be about as large as we could use to an advantage on so deep a stripping as this. A larger machine would have enabled us to handle the large blocks of sandstone rock with less subsequent blasting. There were serious disadvantages seen in the larger machine at that time, and it was not considered as adapted to the conditions.

J. B. WARRINER.—I recognize very clearly the truth of the remarks made by the two previous speakers. My idea is not that we can ever equal the Kansas results, but that our results at the present time when we are only doing about a third or a quarter as well as they do, are not highly creditable. If we better these results to do even one-half as well as they, we will have accomplished a great deal for the industry. This can be done only by the use of large-sized equipment, and that means that the equipment all the way through the stripping must be changed; wider gages and bigger dump cars must be used as well as larger shovels. The usual gage for stripping tracks at the present time is 36 in. and in certain cases 42 in., and the dump cars are of about 5 yd. capacity or even less. To use a shovel handling a 5- or 8-yd. dipper with small equipment of this kind would be out of the question. To get the full economic benefit out of the cost of such a machine it would be necessary to use 20-yd. dump cars at least and standard-gage tracks.

CHAIRMAN CATLIN.—This matter of costs, I think, unless it is reduced to common denominator, is largely a state of mind. I believe comparisons between two localities widely separated are unsatisfactory, as conditions are probably very different.

D. B. REGER, Morgantown, West Va.—How does coal mined by stripping compare in quality with that mined by the ordinary mining method?

J. B. WARRINER.—There is practically no difference in the quality of the coal itself. The quantity of refuse that must be excavated with the coal, however, varies. Naturally, all the stratified refuse of the vein must be excavated with the coal; also nearly all strippings are of areas that have been previously worked over and the old breasts and chutes distributed through this area are filled with rock which has fallen from the roof. This rock, or refuse, is mixed with coal which has sloughed off the ribs of the opening, or perhaps a bottom fender, or a top bench of

coal may have been left in. In the gradation zone between the virgin pillar and the old openings it is of course impossible to excavate with the shovel, or even by the chute method, and prevent contamination of the coal with refuse. The practice, therefore, is to make as clean a separation as possible and if cars contain too high a percentage of refuse to warrant sending through the breaker, to dump them elsewhere and to pick the coal out by hand. Where there is as high as 50 per cent. of coal in a car it is usually sent through the breaker.

B. F. TILLSON, Franklin Furnace, N. J.—In extenuation of the Chairman's remarks, I might suggest it would be of much value, I think, to the members of the Institute if cost data could be placed on a basis of units of labor and units of material used in operation. I think it is of extreme importance to have our discussion on a basis of labor and material in order that we can compare it to our own practices.

CHAIRMAN CATLIN.—That eliminates only part of the trouble. We still have the differences of local conditions to contend with and whether you take it in terms of dollars or terms of hours of labor, it is still the difference in local conditions that makes a wide divergence.

Portable Miners' Lamps

Discussion of the paper of EDWIN M. CHANCE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 235 to 243.

HERBERT M. WILSON, Pittsburgh, Pa. (written discussion).—Permit me to endorse the author's conclusions and their form of presentation as being, in my judgment, the last word on the subject of lights for the illumination of coal mines. I also feel that the open-flame oil lamp has outlived its usefulness, and that it should go, in the interests not only of safety but of efficiency.

Where an open-flame lamp can be used with safety in non-gaseous mines, the acetylene light adds materially to efficiency by its better illumination, and to safety for the numerous reasons given by Mr. Chance. The latter, however, lays chief stress on the value of the acetylene because of its non-fouling and non-obscuring effect on the atmosphere. Another important advantage over any other form of illumination is the clear, bright light projected at the point where the workman may look for danger. The better illumination materially safeguards the worker against obstacles over which he may stumble when traveling, and against dangers from falls of roof or of coal, weakness in timbering, and similar conditions. The acetylene lamp, where it may be used with safety, has advantages even over the present electric cap lamp, in that it gives a higher luminosity, and when the miner comes to learn its signals, as he will in time, just as he learns those given by the open-flame or the safety lamp, he will find that the acetylene lamp is a fair detector not only of methane but of carbon dioxide, as clearly shown by Mr. Chance.

The only serious objection raised against the use of carbide lamps has been recently in Pennsylvania, due to the belief of the State Mine Inspection Department that it endangered miners working on pillar work, or other places where carbon dioxide might accumulate, or where there is lack of sufficient ventilation, and because it was believed that the indications of the carbide lamp would not give the miner sufficient warning. On this point Mr. Chance has clearly shown the fallacies of the objections. I may add that for those unable to detect carbon dioxide with the acetylene lamp, the maximum of protection would be afforded by using an open oil lamp or a flame safety lamp in such places; or, because the miner might not watch his oil light, a better protection still would be offered by a sufficient number of daily inspections of such working places by a fire boss or other official, with flame safety lamp.

I agree with Mr. Chance that in gaseous mines the electric cap lamp as now perfected is superior to any other form of illumination. The

workers using it must, however, be protected against the accumulations of explosive quantities of gas by sufficiently frequent testing with a flame safety lamp by a mine official. With this protection, the electric cap lamp gives not only greater efficiency and greater protection, because of its more clear illumination, than the flame safety lamp, but it also renders the user far more safe from the possibility of ignition of gas. I have knowledge of a number of gas flares or explosions occurring within the last few years, when the only possible source of ignition appears to have been a defective flame safety lamp. Time and again I have had brought to my attention defective safety lamps used by men unskilled in their use, which lamps were, in some cases, in the presence of moderate quantities of gas, and which were in such a condition that the flame would have ignited an explosive mixture of gas. This could not happen with an electric cap lamp.

The States of Kentucky, Kansas and Colorado are in advance of some others in respect to the use of open-flame oil lamps, which are prohibited in the former State. I hope that similar prohibition may ere long be made in Pennsylvania, Illinois, Indiana and other States. Like Mr. Chance, however, I agree that the effects of legislation in respect to mine lights has not always been successful, and that the safety measures and the favorable premium rates based thereon adopted by casualty insurance companies under Workmen's Compensation, have been most effective in improving the illumination in mines in the last year or two. A recent ruling by the Industrial Board of Indiana provides, under the Indiana Workmen's Compensation Act, that "No compensation shall be allowed for an injury or death due to the employee's wilful failure or refusal to use a safety appliance, or performing a duty required by statute," and the Board holds that refusal to use electric safety lamps comes under the provisions of that section of the Act.

M. D. COOPER, Curtisville, Pa. (written discussion).—The interesting paper contributed by Mr. Chance on Portable Miners' Lamps brings out several valuable points. More particularly, his clear explanation of the comparative values of oil and acetylene lamps as regards the indication of black damp ought to go far in the direction of removing a rather widely held notion prejudicial to the acetylene lamp. Mr. Chance clearly demonstrates the safety features of the acetylene lamp.

The author's discussion of the work of the Bureau of Mines in bringing about a remarkable similarity in construction of the American types of electric cap lamps is well made. In fact, some of us have come to believe that there is less importance attached to the particular type selected from among the several well-known makes than to the proper care and handling of the lamps after they are placed in service. Careful study of the causes of lamp failure has revealed the fact that most of

these may be traced not to defects in the lamp, but to carelessness in handling and use.

There are two general causes of the failures of electric cap lamps. For one, the lamp-house attendant is responsible; for the other, the miners are to blame. Care in manufacture may be speedily offset by lack of attention or ignorance on the part of the lamp man. Therefore, it is essential that a competent and careful person be placed in charge of the lamp-house. I know of a case where the results obtained in the use of electric cap lamps at two adjacent mines operated by the same company and similarly equipped, were satisfactory in one, while there were constant failures at the other. The difference was due entirely to the lack of a skilled and careful attendant at the mine where the failures occurred.

The miner himself, or the user of the lamp, is largely to blame for the loss of time resulting from the failure of the lamp to give an adequate light throughout the shift. At best, mine conditions offer a severe test of the construction of the lamp. But the miner has, for instance, a popular habit of removing his battery from his belt and carrying the apparatus by means of the cable. Such practices as this must be stopped before entirely satisfactory results can be expected. But with proper treatment both in the lamp-house and in the mine, it has been found that very good service may be obtained from the electric cap lamp.

ROBERT P. BURROWS, Cleveland, Ohio (written discussion).—Mr. Chance has given a very interesting review of the portable lighting situation. To one not directly connected with the daily operation and application of the various types of portable lamps, Mr. Chance's data seem to be very practical and are undoubtedly of considerable value in considering the question of such lighting equipment. The opinions of the miner, together with practice data relating to the operation of the apparatus, will always be the largest factor in the final decision. It is true, however, that the more fundamentally scientific analysis of this problem may contribute much.

Mr. Chance has given much practical information, but has not attempted to give complete technical data nor comparative costs of operation. This information would be of considerable value, it seems to me, and worth much further work. The data Mr. Chance has given, of course, do not completely specify the illuminating power of the lamps. His method of test must assume the limiting angles of useful field and the relative value of the light at various angles in that field. Unless we come to some standard method of comparison, involving both the question of light output and cost of operation, I am afraid that many of us will be confused. For instance, the total light emitted from one type of cap lamp may probably be higher than the total light emitted from another

of different light distribution. If, however, the two are compared on the basis of a working plane, the efficiency of operation might show to the disadvantage of the one giving the greatest total light. This may be due to the variance of the distribution of light from the two different units.

In view of the importance of this class of lighting and the various types of equipment, perhaps it would be possible, through coöperation of the large mining interests, together with manufacturers of portable mine lamps, and laboratories, where all the necessary equipment is available, to obtain data that will give such final results as cost per unit of light on some standard basis, or the cost per total unit of light emitted from the various equipments, including such items as upkeep and depreciation, interest, etc., which are not now considered.

I am sure there are manufacturers or groups of manufacturers who are sufficiently interested in this subject to be willing to furnish such photometric and comparative candlepower information as they could, and to obtain from large mining interests such average costs of operation as could be obtained from large installations. These data when carefully analyzed would undoubtedly be of considerable value to everyone concerned.

H. H. CLARK, Washington, D. C. (written discussion).—The table of relative candlepowers and costs of operation is especially interesting. The electric cap lamp has been in general service for so short a time that accurate data with respect to the cost of operation are not available. I believe, however, that some of the lamps can be operated at a cost of less than 2 c. per shift. I will just mention in passing that operating costs may be computed upon so many different bases that a statement of cost operation has no absolute value unless accompanied by data with reference to the basis of computation, and so far as I know there is no standard basis of computation.

I have always been interested in statements of safety lamp candlepower and have found that they do not always agree with one another. The candlepower will vary with the flame height, the fuel, and the basis of computing the candlepower; *i.e.*, whether it is mean horizontal, maximum in one direction, or average over the entire illuminated zone. In tests that I have been interested in we have found considerable difficulty in checking our own results and in checking the results of other investigators. We have not yet done any very extensive work with respect to the candlepower of flame safety lamps, but some time we hope to, and then we plan to measure the total light produced by the lamps after standardizing very carefully the height of the flame and the quality of the fuel.

Some time ago I made some measurements of the average candlepower over the light stream of Wolf-type lamps, using as a fuel naphtha

having a specific gravity of about 0.70, and trimming the flame to a height of 1 in. The average intensity of the light stream under these conditions was determined as between 0.4 and 0.5 cp.

More recently, some tests have been made with the Ackroyd & Best lamp, burning mineral seal 300° oil with the flame trimmed to 1 in. in height. Under these conditions the average candlepower over the stream of light was determined as between 0.5 and 0.6 cp. It is hardly likely that the discrepancy between these results and those given by Mr. Chance is entirely due to personal equation or to inaccuracies of observation, but rather to conditions under which the tests were made.

It seems to me that the best way to compare the light-producing capacity of flame safety lamps is to compare the total amount of light that such lamps give, and this is most easily done by comparing the total light flux expressed in lumens. While some may balk at the new term when candlepower has been used for so long, I believe that it is the proper term to use, and, moreover, believe it will be used more and more as time goes on. Makers of incandescent electric lamp bulbs are urging the adoption of the lumen unit for the comparison of the product, and for this reason I believe that in a few years we shall be comparing no longer by candlepower but by lumen ratings.

E. M. CHANCE.— Mr. Clark's figure of 0.4 cp. for the Wolf type is interesting, and I think I know where the discrepancy, if any, exists. In the first place I am decidedly skeptical as to whether the integrating sphere is applicable for use in photometering flame safety lamps due to the fact that a very slight fouling of the air from lack of ventilation causes an enormous drop in candlepower. I am not quite sure whether I am quoting correctly, but my impression is that it has been shown with ordinary lights the candlepower falls off $3\frac{1}{2}$ per cent. for each 0.1 per cent. decrease in oxygen.

Mr. Clark's data are exact, precise and of unquestioned accuracy, as is the case with all the work that he has done, as I can bear witness. The figures that I quoted were based on approximations. This paper is in a sense non-technical and the average mine superintendent or mine foreman or mine engineer, for whom I wrote that paper, is not interested in the point Mr. Burrows brought up, that one electric lamp will give 10 per cent. more light than the other. My figures on the Wolf type are practically maximum; that is, the lamps were driven at as great a light intensity as it was possible for me to secure from the lamps. They are undoubtedly far in excess of what is obtained in practical work. I do not doubt but that the ordinary Wolf-type lamps so widely used in bituminous regions of Pennsylvania, under working conditions, will give not to exceed 0.3 cp. The Davy lamp will probably not average, in the hands of the miner, above 0.05 cp.

The mining industry has been so educated by advertising literature

to look for excessive candlepowers in safety lamps that it was desirable, to my mind, to bring the matter down to a basis more closely allied to fact. With reference to the question that Mr. Clark raised regarding cost, I do not think we can quote costs. We can give instances of what certain lamps cost in certain mines. It is certain that with the electric miners' lamp we are in ignorance of what costs will be and will remain in ignorance for a matter of 18 months or 2 years.

G. S. RICE, Washington, D. C.—Is your test made with a round-wick Wolf?

E. M. CHANCE.—Yes.

G. S. RICE.—And the flame is perhaps more than an inch in height?

E. M. CHANCE.—Yes. The miners ordinarily carry their flame as high as they can, especially with a Davy lamp or a lamp burning a compound fat oil. They hang the lamp up and the flame gradually decreases in height, and if they start with an enormously high flame, they do not have to attend to it so often. It is the height of inefficiency and smokes the lamp badly, but that is the way they do it.

The fact should be borne in mind that a very slight difference in the height of the flame of a safety lamp when being photometered will make a great difference in the values found. This probably explains the discrepancies between Mr. Clark's values and mine. He has carefully measured the candlepowers given when the flame was exactly 1 in. high, while I measured the candle powers given when the flames were burned at the height that my experience had taught me is usual in practice.

Shot-Firing in Bituminous Mines

Discussion of the paper of M. D. COOPER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 123 to 133.

LUCIEN EATON, Ishpeming, Mich.—It is not the custom in the Lake Superior region, as far as I know, to employ shot-firers. Each man, or pair of men (most of the contracts are given to two men) do their own firing; each contractor keeps his powder separately. The general custom is to distribute powder (dynamite only is used) to the mines 2 days a week. The boxes, marked with the contract number and the level, are taken underground and the men take them to their working places. They are kept in boxes, which must not be nearer together than 50 ft. and are usually away from the timber roads or traveling roads. When holes have been drilled, one man usually takes down the machine, and the other man makes up the charge. He cuts the fuse, puts in the caps, etc., at a distance from the powder box. He then carries the powder charges to the working place where both men charge the hole. They do their own firing and warn others coming from any direction that the holes are being fired.

The customary manner of warning is to call "fire" and give the number of holes; when the holes have all been shot, they return to their working places.

We have discussed having special powder men and shot-firers. In fact, a good many years ago, when nitroglycerin was used, they had special men very often for charging. That requires usually the keeping of a large amount of powder in one place and we found it much safer to keep our powder supplies separate rather than in a magazine underground or one on the surface near the mines. We get our powder directly, except in outlying districts, from the powder companies' store houses. They deliver it right to the mines.

T. M. CHANCE, Philadelphia, Pa.—A few years ago, Mr. Vallat, of the Newport Mining Co., showed me a system of firing shaft rounds which they were using at that time in the sinking of the Palms shaft. This system employed fuse firing, but the fuses were lighted by the use of a battery together with a detonator, the ends of the fuses being led into a cardboard box which was partially filled with black powder covered with a small amount of rifle powder; the powder was fired by the detonator, which lighted the fuses. I understood they used from one to two boxes in the shaft to fire the round. The great advantage in the method was that it combined the advantage of timed shots secured through fuse-firing, with that of firing the round from the surface by means of a battery. The shaft was somewhat wet, but the boxes were paraffined and seemed to work very well.

A few months after I saw these boxes in use at Palms shaft, a contractor who was employed in sinking a shaft for us here in the East endeavored to make use of the same method, but without very much success, although I believe that the trouble his men encountered could have been overcome. I also understand that the Cleveland-Cliffs company used this same system of shaft-firing in sinking the Athens shaft, and I would like very much to have Mr. Eaton tell us what results they secured in that work with this method of shot-firing.

LUCIEN EATON.—We used that for a very short time. This firing method which Mr. Vallat worked out consists of a box about a foot long and probably 4 or 5 in. square in cross-section, with small holes about $\frac{1}{4}$ in., bored in it from the sides. The fuses were cut and inserted in these holes, the fuses being cut in different lengths for the order of rotation. Then inside this box was placed about a tablespoonful of coated blasting powder, also a little package of fine rifle powder that ignited with an electric squib.

This blasting powder on firing made a long flame, igniting all the fuses so they would go off all at once. That made it possible to do the blasting by electricity, merely by that ignition. We tried that method of blasting in high raises. Instead of using an electric squib, although we did use that

in some places, we merely inserted a fuse in the end of this box which contained the powder. For convenience, the blasting powder for each firing was made up and kept in a sealed envelope. It was then merely necessary to cut a hole in the envelope, put in the fuse and when everything was ready, cover over the raise (we were supposed to have the end of this ignition fuse hanging down); the last man would light the fuse and go down. He left a long fuse so he could go down the shaft. It made quite a saving and was a great convenience. Recently we have been sinking some other shafts. We used what are known as electric fuse igniters. These consist of an electric squib to which is attached a short piece of fuse. The fuses come in lengths of 2, 4, 6 and, I think, 8 in. We used only the 2, 4 and 6. The igniters are made up by crimping on a blasting cap (we used number eights on the lower ends of the short piece of fuse); then wrapping the whole thing, squib, fuse and cap, with oakenite tape, to make it waterproof, and then dipping it in diluted paraffin paint. One man makes up all the powder and fixes the priming sticks. The shots were fired with the battery in series and the results have been very satisfactory.

We have had considerable water in the shaft but as long as the water does not get above the joint made in the leading wires we do not have any difficulty. We gave up the use of the blasting box such as Mr. Vallat used in Newport, except in dry places. I think these electric fuse igniters will take its place entirely.

H. M. CRANKSHAW, Hazleton, Pa.—It seems to me that the main point about shot-firing in bituminous mines has been almost lost sight of in this paper. The main thing in firing shots in bituminous mines is to fire shots in such a way that there shall be no possibility of an explosion occurring. Explosion after explosion in soft-coal mines has been traced to blown out shots and the whole system of shot-firing must be so organized that if a shot does blow out it cannot cause a dust or gas explosion and so make a great deal of havoc.

In the first place, a bituminous mine should be zoned, that is to say, either by watering or by rock dust protection, it should be so arranged that if an explosion occurs locally it cannot spread through the mine. This paper simply goes into the details of actually firing shots and does not touch the general subject of shot-firing in bituminous mines.

B. F. TILSON.—If it is not astray from the subject, the question which Mr. Eaton raised in regard to the use of delayed action and electric exploders in one that is possibly of great interest to us. Our own experience at Franklin where we have been using them off and on for the past 6 years, has shown us that the use of the delayed-action exploder is now pretty satisfactory so far as the exploder is concerned but that the manipulation of the hand-operated blasting machine is something which is rather tricky, that is, unless the machine is level and the operator pushes the plunger down with an accelerated force, it is quite probable that all

the exploders will not be ignited; and many times we have suffered loss in the failure of a round being properly fired (some of the holes not being ignited) owing undoubtedly to the improper operation of a blasting machine.

This is particularly true where rounds include from 20 to 30 holes. We have used as many as five delays and the instantaneous electric exploders. Of course, what has seemed far superior, where you can employ it, is to use electricity from a direct motor generator outfit, but that is particularly applicable possibly in shaft-sinking or limited operations and a little more difficult to perform where illumination of an electrical nature is not in general practice.

THE CHAIRMAN (R. M. CATLIN, Franklin Furnace, N. J.).—A good deal that has been said relates not especially to bituminous mines. I think we are nearly all interested in the subject of shot-firing and what has been said, I think is more or less applicable to all mines.

LUCIEN EATON.—We formerly used delay-action electric fuses, which are different from the fuse igniters and had unsatisfactory results. Probably for the reasons just given, some of the holes would not go and some would go too soon. We tried it both with the battery and with the electric current. We found by test that there was a certain definite limit to the amount of current to which the exploders should be subjected to give the best results in blasting with the direct current but we found the best results to be with large hand batteries. We use the No. 5 DuPont battery which is good for 100 holes and we shoot as high as 150 holes with them; if the men are careful in making good connections all the holes will be blown, but if they are not, the resistance in the connections seems to be enough to cut down the capacity of the battery. If a battery is good for 100 holes, if they are all clean, it may not be good for more than 35 or 40 if they are carelessly done. We found that by having a battery of much larger capacity than theoretically needed, we get very satisfactory results.

MR. TILLSON.—My remarks applied to fuse igniters as well as delay-action electric detonators.

CHAIRMAN CATLIN.—This is in connection with this subject, but not peculiarly applicable to the bituminous mines—particularly the question of the care and distribution of the explosive. I think it would be interesting if somebody could devise a way in which it would be possible to control the distribution of the explosive. Particularly in these times it is desirable not to have deadly material very widely distributed. The paper touches on that matter in relating to the caring for the explosive and carrying it in boxes. The difficulty of control is considerable, particularly where there are open quarries in the vicinity.

Powder is used indiscriminately and often gets into the hands of those

that should not have it. Is there any way by which the illicit use or the illicit possession of powder can be avoided?

E. T. LEDNUM, Joplin, Mo.—Under the heading “materials used,” Mr. Cooper reports that permissible explosives were used exclusively in blasting coal, while on pages 124 and 126 he reports that straight nitro-glycerine powders were used for blasting rock from falls in the entry, and that the explosive charges were simply placed on the boulders and covered with wet clay. A short hole drilled into a boulder with a very much smaller charge of permissible explosives, say a part of a cartridge, would not only give better results, but at the same time reduce the vibrations with possible subsequent falls of roof, and also make a safer condition in the mine, especially in the presence of gas.

As to blasting batteries, dry cells are uncertain and liable to fail without warning. The small blasting machine, such as made by the Du-Pont company, is safer and more efficient. It is practically indestructible, and will last indefinitely when properly handled. The wires leading to the blasting circuit are held in place by thumb screws on the blasting machine.

On page 129, the method of charging the priming cartridge and the fact that the end containing the cap is placed in the drill hole last, may also result in misfires. In addition, the sharp bends in the wires from the half-hitch around the cartridge, may also result in broken wires or short-circuits. An improvement on this method to hold the electric blasting cap in place would be to punch a hole diagonally through the end of the cartridge, double the wires and force them downward through this hole, and pull the wires through until a loop is formed through which the cartridge could be placed. After this, punch a hole straight in in the end of the cartridge at the top, parallel to the longest axis of the cartridge. Then place the electric blasting cap in this last hole and pull the wires tight. This should give a safer primer, and at the same time prevent breaking of wires or short-circuits from sharp corners.

Under the heading of Misfires, defective or exhausted batteries is a fruitful source of trouble. As I have previously outlined, they become exhausted without warning, and the wires are also merely held in place at the terminals. Proper priming and charging methods will overcome misfires from broken wires or poor connections or short-circuits, etc.

Defective detonating caps may be the result of improper storage and rough handling. Caps should be stored in dry places, as they are sensitive to moisture.

Deterioration of explosives is also caused by improper storage.

Detonating caps may become detached from the cartridge, as a result of the method of priming as indicated by Mr. Cooper. I feel quite sure that this condition can be overcome by proper priming methods, and I

have found to be very satisfactory, the method which I have attempted to describe.

EDWIN M. CHANCE, Wilkes-Barre, Pa. (communication to the Secretary*).—The point has been raised by one of the speakers that the object of shot-firing in bituminous mines is to prevent the explosive from causing a gas or dust explosion that will spread to considerable portions of the mine workings. I would submit that this conception is fundamentally faulty. To me it seems that shot-firing has for its chief object the cheap and efficient breaking down of coal so that it may be loaded into cars, and transported to the surface.

New, to my mind, Mr. Cooper has shown us how such work may be carried out in detail and has thus rendered a valuable service to the mining industry by making this information a matter of permanent record. There is often a great temptation to be so carried away by the larger and more theoretical side of questions such as this of shot-firing that the writer loses his sense of perspective to such an extent as to forget to give the small and necessary details that in the aggregate form the basis of success. Mr. Cooper has painstakingly transferred these details from actual practice to the printed page, from which they may at any time again be transferred to practice. For this he deserves much credit.

J. J. RUTLEDGE, McAlester, Okla. (communication to the Secretary†).—In some mines in the Southwest, the miners' daily supply of permissible explosive is sent in to his working place in galvanized iron canisters of rectangular cross-section, in size about 9 by 6 by 3 in. These canisters contain only sufficient explosive for one day's use, and the canister and any unused explosive is sent from the mine before the close of the shift. In accordance with the mining law, the company handles the explosive both ways. This procedure prevents the leaving of any explosive in the mine over night, or for several days, where it would deteriorate. All shots in the part of the country above-mentioned are fired by special shot-firers after all other persons have left the mine workings. In some States the miner tamps the shots, in others the shot-firer tamps the shots as well as fires them. There has recently been brought to my notice a fiber canister for carrying permissible explosives which is said to be used in many Colorado mines. It is water- and weather-proof as well as a non-conductor of electricity, and for this reason very desirable in mines where there are electric wires. It can be purchased for from 60 to 70 c. It is 9 by 6 by 3 in. in size.

The author says that 40 per cent. straight nitroglycerine dynamite was employed for breaking up an occasional rock-fall in an entry. In my opinion, dynamite should not be employed in a coal mine. A per-

* Received Mar. 5, 1917.

† Received Feb. 5, 1917.

missible explosive can be obtained which would perform the work mentioned equally as well, or better than dynamite. The method of handling and distribution of the electric detonators is commendable, also the use of a safety-contact firing battery. Some idea of the difference between the conditions in Pennsylvania bituminous machine mines and in the solid shooting mines of the Southwest may be gained by a consideration of the relative number of shots fired by one shot-firer in the two districts in certain periods of time. In the Ellsworth mine, one shot-firer fired 40 to 50 shots in an 8-hr. shift, while in Kansas and Oklahoma one shot-firer frequently fires an equal number of shots in from 1 to 2, or 3 hr. time, but it is manifest that firing at this rate does not permit the proper inspection of preparation of the charges, where this is done by the shot-firer. In the former State, the miner prepares and tamps the charges but the shot-firer fires them; in the latter the shot-firer tamps and fires all the shots, the miner having prepared the charge.

The remarkably small number of misfires resulting from defective electric detonators as noted by Mr. Cooper certainly testifies to the reliability of the electrical method of firing shots.

The paper brings up for discussion the debatable subject, "Is it safer and more efficient to fire shots of permissible explosives, by means of an electric battery, by a special shot-firer, while all the miners and other employees are in the mine, or to fire them by special shot-firers after all other employees have left the mine?" The plan as outlined by Mr. Cooper could not be followed in Kansas or Oklahoma mines without coming into conflict with the mining laws of the States mentioned. Mr. Cooper's paper corroborates the experience of other fields, viz., that permissible explosives can be used successfully and with efficiency only when the shots are charged and fired by special shot-firers.

The suggestions in the first paragraph, under Precautions in Shot-Firing, are especially pertinent. Coal-mine officials should keep in close touch with their shot-firers and should caution them in regard to the dangerous practices to be avoided, and should give the shot-firers proper support when they take a stand against firing dangerous shots. It does seem sometimes as though the most reckless or ignorant man is employed as a shot-firer instead of the most careful and intelligent man. A mine owner or superintendent would not think of entrusting his property to the care of a drunken or sleepy night watchman, yet many operators and mine officials place their entire underground properties in the hands of incompetent or reckless shot-firers. They make the shot-firer the scapegoat and apparently quiet their consciences by the argument that, by the mining laws, the shot-firer is made the sole judge of conditions under which shot-firing is done; if he comes through the ordeal with his life, well and good, if he does not, then another man can be found who will be willing to take the risk for the high wages offered. The manage-

ment of the mine owes it to the owners of the property, to the general public, and to the shot-firer and those dependent upon him to make his work as safe as possible, and to that end he should be carefully chosen and his work regularly checked up to make sure that he is not following dangerous practices. The State Inspectors should also check up the work of shot-firers. There is one mine in the Southwest, in which during a period of 10 years, 15 shot-firers have been killed. Quite a number of mines have had four to six shot-firers killed at intervals varying from several months to several years. A mine superintendent should take as much interest in the work of his shot-firers as he does in the work of his fire bosses, since the safety of life and property depends on the carefulness of both.

GEORGE S. RICE, Washington, D. C. (communication to the Secretary*).—Mr. Cooper should be congratulated on the unusual data he presents. It is not often that we have from a trained engineer so interesting a record of daily performances in a special work like that of shot-firing.

The method employed by the Ellsworth Colliery Co. seems particularly good, and the example of the care taken in making examination of roof and gaseous conditions before firing a shot is worthy of emulation in the Southwestern coal field, where the shot-firers often fire as many shots in a couple of hours as are fired in a full shift at Ellsworth; but there are several points which seem to call for comment.

The evidence regarding fumes from permissible explosives given by Mr. Cooper is most interesting. It has been alleged by the miners' officials in some districts that the fumes from permissible explosives are bad. Mr. Cooper says that no special distress was given to the shot-firer, except in changing from one class of explosives to another, but the charge-limit ($1\frac{1}{2}$ lb.) of the explosive was not exceeded, which, of course, is important. Apparently he allowed a very brief time for the diffusion of the smoke and fumes, since he says "it is always advisable to wait a minute or two until the ventilating current has cleared away the smoke." It would seem best where a shot is in a heading or room where the air circulation may not be active, to wait 5 min.

For a mine that was regarded as gaseous, since locked safety lamps were used, and in a mining district in which the coal dust is of an explosive nature, it seems unwise to employ ordinary dynamite for breaking up falls of rock in adobe shots, as described, or even in "block shots." There are a number of permissible explosives that will do practically as efficient work in breaking up rock, and even if not exactly equivalent, the slight gain through the use of dynamite is not compensated for by the increased danger in a bituminous mine, potentially dusty or gaseous, in

* Received Feb. 2, 1917.

the case of neglect of some individual, through its use in blasting, or through accidental discharge in handling. But even when permissible explosive is used, every adobe shot should be covered with a good-sized heap of incombustible material, preferably with dry incombustible dust on top to form a dust cloud. Such covering greatly increases the efficiency of the explosive and lessens the danger of ignition of coal dust in the vicinity.

The diagram of the shot-firer's battery is not sufficiently detailed to make it clear about the position of the contact buttons. If, as it seems to show, these buttons project above the top of the dry-cell box, this does not seem so good a plan as to have them recessed, since an accidental blow or fall after the key is turned might cause a premature depression of both, whereas if recessed this danger would be much more remote.

The test made of the detonators by running a mine locomotive over them to determine whether the detonating compound, usually mercury fulminate, was in proper condition, and the conclusion that in two instances it was missing from the containers, is a method of testing open to question. The explosives engineer of the Bureau of Mines, S. P. Howell, made five informal trials of the method, by placing detonators on a rail and running a $2\frac{1}{4}$ -ton truck with flanged iron wheels over them. The end of the cap that contains the mercury fulminate was squeezed out in each case without explosion. The detonators had been so placed that the electric bridge ends were just beyond the edge of the rail and in three instances were not destroyed, so that on connecting the detonator wires with a firing battery the fulminate remaining in contact with the bridge was discharged. It would seem best for the testing of detonators that failed in a mine, to take them outside the mine and try with a stronger current.

The best way to destroy uncertain detonators, Mr. Howell advises, is to tie several in a bundle with a fresh, presumably good, detonator and fire them with a battery, after placing them behind a barricade to avoid flying particles of the copper container.

The shot-firer's report to the company is an excellent one for recording data, and it would be very interesting if the company would give a summary for a year or more of the results obtained not only by Mr. Cooper, but by the other shot-firers, giving a summary of the data collected through the agency of these shot-firers' reports, especially the cause of misfires and blown-out shots.

D. HARRINGTON, Butte, Mont. (communication to the Secretary*).
—To one not familiar with the coal field in which the system described is located, a few additional facts would be of value: What is the seam thickness? What is the nature of the coal as to friability, partings,

* Received Feb. 26, 1917.

etc.? What is the nature of the roof? Is the coal wet or dry? What is the pitch?

Mr. Cooper's paper brings out the fact that many meritorious practices are followed in this mine. In general mining practice, as well as in shot-firing, it is pleasant to learn that the coal is undercut and that cuttings are loaded into cars before shooting, as against the only too abundant practice of leaving cuttings not only on the floor, but banked in the cut in such a manner as to practically negative the value of cutting. The use of permissible explosives in amounts less than $1\frac{1}{2}$ lb. per hole is to be commended as against the use of black powder or dynamite, or against the use of permissible explosives in amounts up to 3, 4, or 5 lb. per hole and yet expecting the explosive to give no flame and not to shatter the coal. Handling of detonators by shot-firers exclusively is a most commendable practice and to my mind the carrying of explosive in insulated boxes in limited quantities into the mine by individual miners is by far a safer system than that of company men transporting into the mine large quantities of explosives to be distributed later on. The question of transportation of explosives underground and distribution to the working face is certainly one of importance. Carrying the daily supply by the individual miner is looked upon with disfavor by miners' organizations, as well as by many prominent mining men, yet in my opinion past records will prove that far fewer accidents are attributable to individual carrying than to the transporting of large quantities for later distribution.

The small two-cell battery is certainly worthy of extended use, in view of its small cubical content, cheapness and lasting qualities, as well as for the fact that closing the circuit is dependent on two separate and distinct connections, which practically eliminates the possibility of premature firing by accidental contact.

Removing drill cuttings from holes, refusing to fire holes of greater depth than that of the machine cut, use of wooden tamping rod, and of clay for tamping, are all commendable.

The matter of selection of shot-firer is of prime importance, and of scarcely less importance is the amount of authority to be delegated to him. While it is true that the shot-firer should be a man of judgment and discretion, local conditions, generally beyond control of the mine management, only too frequently result in the selection of men of only mediocre ability: in some localities wage-scale agreements are such that the earnings of miners, loaders, and others are so much greater than those of the shot-firer that the only men who will accept this work are the newcomers who do not understand conditions as to shooting and who soon demand more lucrative work, or leave; in other regions the risk of firing shots is so great that only dare-devils, frequently with little knowledge of explosives or of firing shots, will undertake the work, though as much as \$15

per shift is paid for 3 or 4 hr. work, the shot-firer practically betting the employer \$15 or more daily that an explosion will *not* occur even though such barbarous mining practices are in effect as "shooting off the solid" with black powder and fuse, using coal-dust tamping in gaseous places frequently covered with dry inflammable dust.

Assuming that the shot-firer is a competent man, his life is at stake every time he fires a shot and he should be allowed every available safeguard. Apparently the Pennsylvania Bituminous Law covers this at least in part by providing that the shot-firer may "*refuse to charge any holes not properly placed.*" In many localities where similar legal provisions apply, they are rendered inoperative by fines or other restrictions placed on shot-firers by miners' organizations, when the shot-firer refuses to charge certain holes deemed by him to be unsafe; in other regions mine officials force the firing of unsafe holes or compel the use of unsafe materials or methods. The shot-firer, if a competent man, should be able to dictate all details relative to drilling, loading and firing of shots as well as to safety considerations preliminary to shot-firing, such as sufficiency of propping, undercutting if done, condition of air, etc.; if he remains underground while firing shots, his life is endangered with every shot fired and he is entitled to be allowed to protect himself. As a matter of right and justice, the shot-firer should be independent of both miner and company, be paid an adequate wage by the State or Government, and be appointed preferably by a district judge upon recommendation of miners' organizations and company officials and be amenable to court only.

The main criticism of the system described by Mr. Cooper would be that shots are fired during the day while all the men are in the mine, as against firing them by night when ordinarily few, if any, men are working; the ideal system, however, would be an electrical one in which all shots would be fired from the surface after all men were out of the mine. While the use of permissible explosives, in small quantities per hole with electric battery firing, use of clay tamping, undercutting of coal and removal of cuttings before firing of shots, are all commendable precautions, yet firing of shots with men in the mine carries with it many dangers. The working faces as well as air currents have considerable quantities of fumes from explosives, which, while possibly not dangerous, are certainly disagreeable and at least temporarily render air conditions in working places such that clearness of vision is obscured and accidents possible. Again, where shots are fired with men in the mine, frequently the shot-firer neglects to warn before shooting, or he will give warning when firing the first shot, fail to warn as to other shots, or will shout "fire" and without lapse of time complete the circuit through the battery, leaving practically no time for an oncomer to retreat. The successive concussions due to firing shots are inclined to cause loose pieces of roof or rib to fall

even in remote parts of the mine, and these frequently cause accidents. All of these sources of danger are eliminated if shots are fired on the shift when miners are not at work; in some mines where as many as three, five, seven or more shots are fired in a room face, it is customary to fire one, or possibly two shots and allow this coal to be loaded out before firing the other shots. Under such conditions it is generally desirable to have shot-firers on hand during the working shift; however, in the mine described by Mr. Cooper, but two shots are fired in a place and presumably they may be fired simultaneously, or at least in quick succession, and the presence of shot-firers on day shift is not a necessity.

Mr. Cooper does not state whether the sprag placed against the face of the coal after undercutting is removed before firing shots; possibly the influence of leaving the sprag would be small, yet it appears that leaving the sprag would, to a certain extent operate as a bar to efficient work by the powder. The explosive used during the greater part of Mr. Cooper's observation is an ammonium nitrate explosive of comparatively slow rate of detonation. It is noticeable that the priming stick is entered last, with primer toward outside of hole, generally believed by authorities to be the proper method; however, this class of explosives is frequently somewhat insensitive to detonation and to assist in this matter frequently the paper ends of contiguous sticks of explosive are removed. It would be interesting to know whether this is done. It would also be interesting to learn whether the paper covering of sticks is slit lengthwise and whether the explosive is rammed to fill the full diameter of the hole. In wet holes, is the ammonium nitrate explosive used, and if so what precautions (if any) are taken toward waterproofing?

Mr. Cooper's paper is an admirable one, not only for the data it contains but also for the fact that it opens a broad field for discussion of a subject of vital interest to all those interested in the operation of coal mines.

Report of the Secretary of the Committee on Safety and Sanitation

Discussion of the paper of E. MALTBY SHIPP, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 87 to 122.

E. MALTBY SHIPP.—It was the intention of the Committee on Safety and Sanitation to collect information and useful data from the companies having well-organized safety departments and print this in pamphlet form for distribution among the small companies that have no way of employing safety engineers nor have at their disposal these data. Consequently, I took these rules and the general data, especially the rules, went over them a great many times and tried to cut out whatever duplications appeared.

Mr. Eavenson, the Chairman of the Committee, obtained a great deal of information from the coal companies. Mr. Charles W. Goodale sent in a great many booklets and rules and all issues of the *Anode* and other publications published in the Butte district, and this report is a boiled-down skeleton of all of the information at hand. Mr. Sidney Jennings called attention to a duplication as follows: "The principal duplication here might be stated to be between safety signs, and safety rules."

It was thought at the last meeting we had that a great many people might refer to the pages on safety signs to get an idea of the one they wanted to have printed. If they were making up a list of signs they would not be likely to turn to the rules, as they would think the signs had been covered, and *vice versa* in reference to the rules. If they were getting up a set of rules they would not turn to the signs to get out the data from there for rules, consequently in a great many cases the signs and the rules are practically the same for the same conditions and in making rules for general purposes I think that there is a possibility of cutting down, and not putting in the general rules anything that is covered in the special rules.

But again, it is hard to do that, and was not the object of this paper to do that, because we wanted to get the data before a large number of the committee, and also before the members of the Institute who had been doing this work in the field and might criticise from their practical experience.

Mr. Eavenson makes the following suggestions in his letter of comment:

"In the bottom paragraph on page 1, you instance the lack of knowledge of the actual cost of the safety work. So far as I know, the only actual costs that have been published are those of the United States Steel Corporation. These figures were exhibited at the Panama-Pacific Exposition in San Francisco, and have also been published in one of their industrial bulletins, and I can see no reason at all why they should not be repeated in your report as they are the actual figures and show a large money saving aside from every other consideration resulting from their expenditures for safety work.

"In addition to this, the Associated Companies, who are now insuring some of the mining companies in States having certain kinds of compensation laws, are experiencing this same saving, and in a paper read about a month ago before the Coal Mining Institute of Kentucky, some instances of saving were given, and I believe it would be advisable to incorporate these with your report.

"The advantages listed in the first paragraph of page 2, could be strengthened a little by the morale produced among the employees as the result of safety work, as there is no question that the thinking men

soon realize that their employer has their best interests at heart and appreciate the interest that is being shown in this way. We have experienced this in a number of cases in our own work here.

"In the list of Fundamental Principles on the bottom of page 2, I think it would be better to emphasize as fully as possible that an educational campaign is absolutely essential. In fact, I believe that the experience of everyone doing work of this kind is that three-quarters of the problem is an educational one and that the question of safeguards of various kinds, while important, is a comparatively small part of a safety campaign. This same point is emphasized further in your discussion of the human element on page 6, as there is no question that the ordinary man, as a general thing, does not realize many of the dangers daily encountered.

"Another important point which is not emphasized as thoroughly as it might be, under the head of The Organization, is the practice of having committees of workmen appointed at regular intervals to make an inspection of the mine or plant and make any recommendations necessary for safety measures. This point was brought out in my paper presented at the February, 1915, meeting and both ourselves and the H. C. Frick Coke Co. have found that the appointment of such committees results in considerable good.

"In Rule 37, page 28, the words 'or brakes' should be used after 'sprags' as in most of the up-to-date coal mines sprags are no longer used.

"Relative to the use of explosives, it would be advisable to insert that none but permissible explosives should be used in mines generating gas or in those using safety lamps.

"On page 30, Rules 70 and 74 conflict. In Rule 80, on the same page, it should be specified that the gage of augers should be examined frequently in order that the hole may be bored amply large for the cart-ridge. Rule 88, on page 31, is not in accordance with the best practice, as this is to prohibit everyone riding on trips excepting on the regular man trips, used for taking men into the mines.

"It would have been advisable, in my opinion, to have omitted the General Rules and Suggestions for Foremen and Superintendents, as per Jones' Interest in West Virginia Coal Mines, as most of this matter is included in the rules already written."

Now those Fundamental Principles that he spoke of were just a general review placed in the front of the paper and the educational feature is better emphasized on pages 3 and 4 but could be even more featured.

B. F. TILLSON, Franklin Furnace, N. J.—In discussing this report of the Committee on Safety and Sanitation, I am glad to express my high appreciation of the great amount of work that has been done in collecting

so much information from various sources. I believe that the Secretary is to be congratulated for his success in accomplishing this work.

I think it might be of value in addition to this report to state conditions that have since been changed, as apply to the method used by the New Jersey Zinc Co. for encouragement of interest in a safety campaign. On page 96 of this report the first paragraph mentions a bonus system which had been in vogue and which helped to greatly reduce the accident rating. That system has been still further elaborated as follows:

During the past year all shift bosses in the mine who have not had any accidents which caused a disability of a workman received a bonus of \$5 monthly; semiannually, all those shift bosses whose record for the 6 months showed that all the number of disabilities (a disability being an accident which required the loss of 1 or more days of time) would permit a rating under one and one-quarter disability per 10,000 hr. of labor worked, would receive a bonus of \$20.

Furthermore, at the end of the year, an added bonus was paid on a basis of the amount of time lost owing to accidents to men in a shift-boss's gang, and the rating used there was a matter of 0.4 per cent. of the amount of time which was worked by the laborer in each shift-boss's gang during the year. In other words, 4 days of time lost in a shift-boss's gang rated per 1,000 shifts of labor worked was a bonus record and all shift bosses who had an average less than this received an annual prize of \$20.

This system has worked out with considerable advantage to us. In May, 1913, we started in this bonus system which we have changed slightly from time to time in order to equalize the distribution of prizes so as to get the maximum from the bosses. The average for the first 4 months of the year was a rating of two and a quarter shifts of disabilities per 10,000 hr. of labor worked. This rating has been gradually dropping from year to year until we find that our average for this past year was three-quarters of a disability per 10,000 hr. of labor worked. It proves that a stimulation of the interest of the shift bosses underground in the prevention of accidents in order to fatten their payroll is probably the most efficacious means you can employ for gaining their sincere coöperation with all the workmen and their help in the latter's education. In our case we have found it most satisfactory.

H. N. EAVENSON, Gary, W. Va. (communication to the Secretary*).—The United States Coal and Coke Co. at Gary, W. Va., has, as a safety measure, installed electric lights in all of the working places in its mines. The voltage carried at the substations ranges from 275 to 290 volts, and at first carbon-filament lamps of this voltage were used. These gave good satisfaction, but in a short time after their general adoption, their

* Received Mar. 7, 1917.

manufacture was discontinued, and Mazda lamps were then installed. On account of the extreme delicacy of the filament, the life of these high-voltage lamps was very short, and the use of 100-volt lamps, three in series, was begun.

Along the headings lights were installed at intervals of about 100 ft., a lamp always being placed at each switch.

On headings these lamps are connected to the trolley wire, to each other, and the return, by No. 14 wire. In the rooms and other working places, No. 14 duplex rubber-covered wire is used, a length of wire sufficient for the projected length of the room being wound upon a reel which is hung upon a post near the face. At the mouth of the room, each end of this wire is connected to a pull-out plug, which is connected in turn to the trolley wire and the return.

The lamps are connected in series by weather-proof sockets to a special cable ahead of the reel and are carried forward and suspended on posts as the face advances, the reel being moved correspondingly; 40-watt lamps are used, the two outer ones being shaded by 30° metal reflectors, the center one being bare. Experience has shown that this amount of illumination is sufficient for a place as wide as 40 ft. Wires are carried on insulators on an inside row of posts. In all, over 10,000 lamps have been installed.

As usual, considerable objection was made to the installation of these lights, but after more than a year's trial, no one would willingly revert to the old conditions. The increased amount of light available increases the efficiency of the workman, as well as his safety.

The Influence of the Movement of Shales on the Area of Oil Production

Discussion of the paper of RICHARD A. CONKLING, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 119, November, 1916, pp. 1969 to 1972.

DORSEY HAGER, Tulsa, Okla.—Recently we have been doing a little work along the same lines Mr. Conkling has outlined in his paper. I am not prepared to say that Mr. Conkling is not correct but our results do not agree with his. I think it is a question of difference in correlation and interpretation of the well records. Our cross-section of the Cushing pool does not agree with that of Mr. Conkling and we find instead of the beds thickening in the syncline, there is a tendency to thicken on top of the anticline. We took the north dome of the Cushing field, and are now trying to get all the records. It will probably be another month before we have all the data complete but our results at present do not agree with those of Mr. Conkling at all.

Reservoir Gas and Oil in the Vicinity of Cleveland, Ohio

Discussion of the paper of FRANK R. VAN HORN, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 75 to 86.

DAVID T. DAY, Washington, D. C.—The statement was made that shale gas was drilled for in 1883. Wells were drilled and gas obtained from each sufficient to supply three families, in 1865. I worked on one myself in 1868. The gas smelled of gasoline at that time.

Problems Connected with the Recovery of Petroleum from Unconsolidated Sands

Discussion of the paper by WILLIAM H. KOBBE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 120, December, 1916, pp. 2253 to 2276.

THE CHAIRMAN (M. L. REQUA, San Francisco, Cal.).—We have had in California a great deal of trouble from the breaking off and collapsing of well casings from shifting sand, and it is quite true with us as indicated in the paper, that no casing has been found sufficiently strong to withstand that.

I know of wells in which casing that has not been in the hole over 2 or 3 days has collapsed and when that casing has been pulled out it has been just as flat as your hand. I am also familiar with swelling bed rock in the gold drift mining ground of California. It does swell and keeps on swelling and never seems to stop. Just as long as tunnels are kept open in that kind of material there has to be a gang going through the tunnel constantly cutting out and easing up. They have put in timbers 24 in. in diameter in an effort to hold that kind of ground and it has always filled. The only way to succeed is to keep a gang of men busy all the time cutting out and easing down as it squeezes in.

C. NARAMORE, Washington, D. C.—My experience has been very similar to yours in the same field in the matter of breaking off of casing. I had charge of one well, in which I put in reënforced casing, anchored it in place and got 1 day's production, 150 bbl., out of it. The following morning my lease foreman telephoned that it had broken off. I thought at that time we would have more evidence, so I drove out to the well without ordering a new shoe joint. It had been customary to merely start from town with a new shoe joint, pull the pipe, take off the crooked bottom joint, start the casing back and sometimes never touch any iron (lost casing) when redrilling the well. Where the flattened joints went

to, is more than I care to say, but in this case I thought we would hit that old joint. Many operators in the Coalinga field have had similar experience in redrilling wells in which the oil string had collapsed. At the Republic No. 1 well, there are four strings of casings side by side through the oil sands. They broke off as fast as we put them in. One of the lost strings is inserted pipe.

CHAIRMAN REQUA.—It is a very curious thing what becomes of those pipes that are broken off. They seem to have been pushed over sidewise. I have seen evidences myself of a very pronounced movement where the pipe has been bent as though there had been a slide or pushing over. Some people have ascribed it in part to gas and some are inclined to think it is the direct result of some movement in the sands themselves.

C. NARAMORE.—Many of these wells, where we have buried so much pipe, have made a great deal of sand. The casings there have been replaced and moved again and again, so that the total quantity of solids removed has something to do with the cavity below.

On another property I had a foreman who used to say "We've had another earthquake," as he expressed it, when two or three wells would go off at the same time. He firmly believed that the oil sands actually shifted over an area covering several well locations. This view is hardly acceptable as evidence for a discussion of this kind, but he was correct to the extent that three or four wells would be broken off during one 24-hr. period.

DORSEY HAGER, Tulsa, Okla.—I have measured roughly some of the sand heaps in California, and found as much as 100,000 to 200,000 cu. ft. of oil sand that had been ejected from the well. That is just as a matter of information as to quantity.

The Possibility of Deep Sand Oil and Gas in the Appalachian Geosyncline of West Virginia

A discussion of the paper of DAVID B. REGER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 117, September, 1916, pp. 1709 to 1724.

DAVID B. REGER.—I want to add that since this paper was prepared a well is now being drilled at the location recommended on the last page. It was not, however, done at my instance; it was done at the instance of Dr. White, the State Geologist of West Virginia, who has been trying for several years to induce the large oil and gas companies to make a test of some of these deep sands in West Virginia and a well is now just being started in Wood County, right at this immediate

locality, and doubtless in a year or so we will have a test in some of these deep sands probably to the Trenton limestone.

Well No. 33 in the table of well records in the back of the booklet, "The Benson Well in Barbour County", which encountered gas at a depth of somewhere near 4,100 ft. in what is possibly the Kane sand of Pennsylvania, had a rock pressure of 1,625 lb. Now, if you take 4,100 ft. and multiply that by 0.42, the coefficient for hydrostatic pressure, you will find it comes so close to 1,625 lb. that it surprises you. I do not mean to start a discussion about hydrostatic pressure and rock pressure and what gas pressure really is, but I simply furnish this as a very interesting fact that has recently come to light in connection with this deep-well discussion.

M. M. THOMPSON, Morsemere, N. J.—About a year and a half ago I did considerable detailed work in the region around Charleston, W. Va.

I was at first surprised to learn how little dependence the operators place in surface geology, in this general region where our present oil-producing industry practically had its birth.

The State Geologist's reports are very thorough indeed, having been worked out by counties with excellent maps indicating surface geology and the results of drilling. The wells are carefully indicated and numbered in accordance with brief tables giving the essential facts concerning them.

A brief study of these geological maps demonstrates the important fact that in the region south of the Little Kanawha River, the developed oil and gas pools cut right across structural contours apparently regardless of any relation between oil and geology. In the northern part of the State where the folding is more pronounced, they follow more closely the direction of the axes of the folds, but usually along a line offset some distance from such axis.

These marked discrepancies are very largely the result of convergence. The actual contour of the oil sand is not parallel to that of the surface beds. The irregular thickening to the southeast modifies and often outweighs the effect of the folds determined from surface observation. Much more information than mere surface structure must be utilized in successfully applying geology to oil-field development here.

Another unique feature of West Virginia is the fact that there are several important oil sands which do not carry any appreciable quantity of water. In regions of this character, where I have worked out the structural contours in developed pools from well records, I found that the oil pools are usually in the synclines. Not often in the trough of the syncline, however, because the porous portion of the sand frequently pinches or becomes "recemented" before it gets down to the lowest part of the trough. The pools seldom extend symmetrically onto both flanks of the fold.

The foregoing conditions are a few of the factors of uncertainty which must be considered by the geologist in West Virginia. Unfortunately, many important points connected with convergent series in fairly old formations cannot be determined very far in advance of drilling. Still, accurate intelligent geological investigations are a great help to the operator even in as difficult a region as this.

DAVID B. REGER.—Regarding the occurrence of oil and gas in West Virginia, in reference to structure, I might say that one thing that has doubtless confused Mr. Thompson is the fact that the sands of the Catskill red beds, in which a great many of our oil wells are found, are nearly all of them lenticular. You will very often find the sand not water-bearing, and lying well up against the slope. When you get the well records of the pool, you find that the sand pinched out completely going westward toward the syncline, so there could not be any oil in it. It got down to the bottom as far as it could, and then the oil pinched out in the shale. You normally find your oil directly along the syncline, because the sand is non-water-bearing and your oil goes down in there, and you must call it gravity because you cannot call it anything else. When you get up the slope in a sand that is regular and uniform you get gas.

The sand at the top of the Pocono is not always water-bearing; that is true of the basin immediately to the northwest of West Union, and there the oil occurs right along the axis of the syncline. You can trace it for miles and miles on the Doddridge County Geological Map and you will find the same thing true in a large proportion of the field maps.

We have a great deal more variable lithology in West Virginia than in Oklahoma; it must be that your sands are more regular there. Lithology has a bearing in West Virginia, no question about it, and we find it out after we drill these wells, but when we locate wildcat wells we usually put them on top of the anticline first to locate the gas and then go on down. If it is up in the Pocono or down in the Catskill we come down in the basin and hunt for it there without much fear of trouble. Of course we get a good many dry holes, but that is largely due to the lithological character of the sands. We figure that the anticlines and synclines are the great controlling feature of the oil fields of West Virginia and that these numerous exceptions are simply caused by the accompanying lithological conditions.

If we had absolutely regular lithology then we would have regular oil fields and we could predict them with great certainty, but lithological structure is something that we do not know ahead of time and it takes a lot of drilling, usually, to bring it out.

L. L. HUTCHISON, Tulsa, Okla.—Is the series there conformable from the lower producing member up to the top?

DAVID B. REGER.—It is conformable except in local places where the sands are lenticular. We have no great local unconformity at the base of the Pocono; of course, there is always a non-conformity at the bottom of the Pottsville where the soft red beds of the Mauch Chunk come in. I made this section for Marion and surrounding counties as typical of the West Virginia oil fields. The Pottsville measures are 300 ft. thick but gradually thicken southward until you get to Virginia where they are 4,000 ft. thick. The Mauch Chunk measures in some parts of West Virginia are not more than 50 ft. thick, composed of a few red shales and green sandstones but in southern West Virginia, in Mercer County, thicken up to 3,500 or 4,000 ft. The Greenbrier Limestone also thickens from 50 ft. in the north to nearly 2,000 in the southern end, and you therefore have that great divergence in those three series going southward across the State.

South of the Great Kanawha River which runs approximately through the center of the State, the Catskill measures thin out completely and these lower shales here, the Chemung, Portage and Hamilton, at the top of the Devonian, thin out also so that when you get down to the Kentucky line, you have the lower Devonian and Silurian measures right up close against the Pocono. All these 4,000 ft. of measures in here have thinned out so that makes it possible to reach these sands in Kentucky.

This great barrier of shales, 3,500 ft. or more in thickness, that we are positive exists between the producing sands of West Virginia and the producing sands of Ohio, has kept the operators from drilling to the Ohio deep sands in West Virginia. There have been some attempts; there is a well now in the State of Pennsylvania, number 42 in that booklet, that is more than 7,200 ft. deep and it is still drilling. That well has recently been cased and I do not know how far down they intend to drill it. It is away down now in the Salina beds. Whether they will ever get down to the Trenton beds or not, I do not know.

L. L. HUTCHISON.—I have always worked on the theory that the anticline, to control accumulation outside of the formations, contained the three liquids, oil, gas and water. The series must be conformable. If we have an unconformable series, the structure you might interpret at the top does not necessarily interpret the structure below the conformity. You have not sufficient non-conformity so that would exist, have you?

DAVID B. REGER.—We have the convergence of the Pottsville and Mauch Chunk measures, which sometimes overrides the surface structure. I had an instance of that last year down in one of the southern counties. I examined some territory that had an apparent terrace structure, which looked favorable for oil. When I obtained enough well

records to plot it on the top of the Big Lime I found, instead of a terrace structure, there was a syncline going right straight across the property and that spoiled our argument right at the start because it gave different conditions. Down in those counties south of the Great Kanawha, where this great thickening of the Pottsville has taken place, you frequently find the basin of oil sands is a mile or so away from the basin on top of the Coal or the anticline shown on the top of the Coal and that circumstance oftentimes explains the apparent shifting of the pools away from the axes.

The Evidence of the Oklahoma Oil Fields on the Anticlinal Theory

Discussion of the paper of DORSEY HAGER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 195 to 198.

DORSEY HAGER, Tulsa, Okla.—I have been asked why the Dexter region is dry. I would like to know myself. I drilled two dry wells on that same anticline which has production to the east and west and south and north. That same anticline has seven domes that have been drilled and only three have proven productive out of the seven. In other words, you have one well-defined anticline with seven domes along its length, all mappable, all checked carefully, and only three of those seven have produced. Now why the others did not produce, I do not know.

L. L. HUTCHISON, Tulsa, Okla.—There is no appreciable difference in the character or structure of the sands?

DORSEY HAGER.—No, I have never been able to see any appreciable difference in the sands where we got production and where we did not get it. I will say that the deepest wells we drilled, in the vicinity of Dexter, had no sands at the point where they got the production farther north.

I. N. KNAPP, Ardmore, Pa.—I first went to the Kansas-Oklahoma field in 1895 and occasionally thereafter until 1899, when I began drilling for oil. I started close to wells that were producing gas and which, when drilling, had made good oil showings in a sand above the gas. I thought these showings were good enough evidence of an oil pool for me without looking for any anticline and I made a paying development in what I was afterward told by geologists was probably a synclinal trough.

If I remember correctly Dr. Orton's report on the Iola gas pool (about 1898) he distinctly said that he found no evidence of anticlinal structure and concluded from the records of a large number of wells that there was a terraced structure of the productive gas sands, of which there were no surface indications.

As far as I could make out from my well records, the top of the oil sands was practically level in the region where I operated.

The lower sands were gas sands that seemed to me to be like sand dunes on the sea shore and the oil sands seemed like sedimentations forming sand bars in the shales and were not continuous sheets of sand rock. I imagined that conditions shown by the drill could best be explained in this way. I do not think there was any evidence of anticlinal conditions. I drilled in all 225 wells in Kansas by 1906 and came out in 1908.

Both the geologist and the operator undoubtedly know the Kansas-Oklahoma field far better than they did 8 years ago.

DORSEY HAGER.—In recent developments there has been a good deal of revival of activity in eastern Kansas in shallow districts, and anticlines have been mapped; the domes are not large, the reversals are from 10 to 20 ft. per mile, very low reversals. Indeed one might call them practically terraces. In some places the surface is such that if one has large shale bodies, it is practically impossible to get any outcroppings to work on. On the other hand, where one has good limestones and good sandstone formations, one can map them and define the structure without any difficulty.

C. NARAMORE, Washington, D. C.—I think it would be well for Mr. Hager to tell about mapping. I was very much interested a few months ago to hear of the methods used by Mr. Hager's men in the field. I have been told of one company in Kansas and Oklahoma that has 190 geologists in the field at the present time.

DORSEY HAGER.—Any well-defined structure in Kansas or Oklahoma that has produced oil can be seen with the eye. Before we send a man out to do any detail work, he is sent out to run over the country using his eye and hand level. East of Tulsa one will find big shale beds that show no structure, but when one gets in the limestone and sandstone areas one can see structure with the eyes.

We first make a reconnaissance instead of detailing to find structure. If we discover structure, we then go in and make the detail map and bring out the minor points of the structure. We detail everything using a detailed scale of 1,000 to 2,000 ft. per inch, allowing an error in closure of levels of 5 ft. in 5 miles as a limit; most of the work checks within a foot. We take the tops of limestones or sandstones as a working base; finer work than that is out of the question. The contour intervals used are generally 10-ft. intervals. Some work has been done on 5-ft. intervals but only when one gets very clean-cut exposures can one use 5 ft.

F. J. HIRSCHBERG, Choteau, Mont.—Our structures as compared to the Appalachian and Pacific Coast structures are very gentle. Throughout Oklahoma and Kansas, however, it is in our favor that any little

change in the topography that is at all noticeable is likely to be caused by structures. In other words, there is close coincidence between topography and structure. In reconnaissance work in those States, you must watch for the dip slopes. If the topography is not gentle to the westward all over the country, look out. If there is a gentle turn to the eastward, ten to one there is structure there, and if there is structure, ten to one you will get production.

R. H. JOHNSON, Pittsburgh, Pa.—Mr. Hager very clearly makes good his second contention that geology has been of the utmost importance in locating fields in Oklahoma and that structure there is of great importance.

My criticism is with the other contention that "the anticlinal theory holds in Oklahoma." I should rather say that the accumulation of oil there was very largely controlled both by structure and lithology. That is, we should treat the two things as coördinate rather than lay so much emphasis on structure.

The time has come when the producer needs to have his attention called to the importance of subsurface mapping. These maps that we make of the structure of the surface, are of course very valuable, but is not the producer of today depending on them too much? Is he spending enough time on the divergence, thickness and nature of these sands?

Now if our anticlinal theory is to really "hold," then our pools ought to consist of a gas pool, surrounded by a ring of oil wells. If we look over the map of Kansas and Oklahoma, we do not find rings like that except in rare instances. In the Bartlesville region we see that it is quite the uncommon thing. Where the oil does have relation to structure, usually lithology has played so important a rôle as to interrupt this theoretical ring.

The next thing that we ought to find, if the anticlinal theory "holds," would be a ring of water wells around every pool. Now if we examine the reason why these marginal dry holes are dry, we find that in about a third of them, the sand is too thin or too shaly or too close. Gas, as is quite apparent from Mr. Hager's list, can be located with anticlinal aid much more easily than oil, not only in the Oklahoma field but in other fields. One possible cause for this is that the gas moving up a homocline is arrested by the domes. If there is enough gas it will preëempt the entire dome down to the spilling point. The oil will then be forced to spill over into the next dome further up the homocline. If the gas is still sufficiently abundant, the gas will be sent on to the next one. If the gas is still adequate to preëempt all that dome, it is sent on up the homocline. Where does it finally rest? It will escape at seepages or, as I think is much more frequently the case, it will be caught at the upper end of the reservoir, giving us a pool which is essentially a monoclinial pool.

DORSEY HAGER.—Detailed mapping of the Glenn pool area has shown an east reversal of 20 ft., which makes a very low dome, or practically a broad terrace.

I. N. KNAPP.—I like to look at the diagrams of the geologists and read their books, but somehow or other when one drills one seldom finds conditions exactly as pictured or laid down by them. For instance, all writers on oil and gas matters that I have read give it as an axiom that gas is always found mingled with or just above the oil in the same sand.

Well! I am now drilling around an old Trenton Rock oil pool and there is not a bubble of gas with the oil or anywhere above it. This pool was opened in 1897, produced over 1,000,000 bbl. of oil and was abandoned about 1906. There never was any gas in it. A salt water sand, however, closely underlies the oil and there is some salt water with the oil.

The most popular pictures in books on oil geology seem to be ones that show an anticlinal with the gas on top, the oil in the middle and the salt water on the bottom of the same continuous sand rock strata or pitching sand lense. I do not question but that these conditions occur in some places.

I have produced a good many hundred thousand barrels of oil in a field where no such conditions were found, for the oil was in a sand by itself and not connected in any way with salt water or gas. There was, however, enough gas with the oil to make a froth but not enough to flow the oil even in the original wells. The oil was entirely free from water, and there was no evidence of anticlinal structure.

R. H. JOHNSON.—With respect to the Glenn pool and the terrace there, it seems to me important for us to consider, Does the limit of the Glenn pool coincide with the spilling plane of the terrace? Now would it not be true, Mr. Hager, that this terrace would control only a small part of the Glenn pool, and if it controls only a part of the Glenn pool, what would place the oil in the other part? Take the Glenn pool as a whole and that terrace only would be capable of accumulating oil at that particular place. Since we get oil in a large additional area, it seems to me the oil would have been there even if this particular structure had not been present.

DORSEY HAGER.—I do not make the claim at all, that lithological character does not play its part. I claim that if one is hunting oil pools in Oklahoma and Kansas, the only thing one has to work on is structure, and a man is forced to use his surface formations and map his surface structure, and then drill. If one is working in a field where there are plenty of well records and plenty of development, then one can determine structure from the logs, and there is no question about lithology. Lensing does play an important part, but if one has a lensed condition and the

lense lies in a syncline or on a normal dip, one may or may not have accumulation in that lense, but if that lense is on a terrace, the terrace will give a larger drainage area as there will be dips from at least three directions toward the top of that terrace. I think that is one of the main reasons why terraces do produce.

Now you will find that wells that are off structure are not, at the outside, over 7 per cent. There is no way of defining those pools before a well is drilled. If lithologic or lensed structures were on a normal west dip and a man drilled for them he would have an ordinary wildcat chance of getting a pool, which is about one in 150 in Oklahoma and Kansas. If a man is to do any development in that country, he must be governed by what structure he has.

We find in listing the pools that 75 per cent. of the domes of Oklahoma and Kansas have been productive.

With that percentage of successes on structure, I think a man would be very foolish to go out into a wildcat country and not be governed by a structure. You cannot tell anything about lithology until you have well records to work with.

L. L. HUTCHISON.—May I ask Mr. Johnson to define exactly what he means by lithology in this connection?

R. H. JOHNSON.—Lithology in this sense means change of porosity. In reference to what Mr. Hager has just said, any of us, no matter how strongly convinced we are that lithology requires consideration, quite agree that structure is the thing to be sought for in locating the test well. Lithology comes to be important after the test well. When we come to feel out from the first well to get additional wells, we need to keep up promptly and accurately the subsurface mapping so as to know our underground structure and lithology as well as possible.

Overconfidence in the structure as made out at the surface should not lead us to neglect any information we can get from better logs and better studies from those logs.

DORSEY HAGER.—I would like to have Mr. Johnson answer the question why it is that so many of these terraces are not one-sand propositions. Where there are one, two, three or four sands in a field we find very often that all those sands produce. It is not likely that lithological conditions will be the governing factor with all those sands; it might be with one sand, possibly with two, but with three sands, the chances are very much against it and I do know that some pools of Oklahoma are producing from three sands on well-defined terraces. If lithological character governs, the terrace would play a very small part.

Mr. Johnson brought up the point of pronounced pools underground where the surface shows very little. I have some blue prints here of some

work we did a short time ago. At the surface of this pool, the beds show a well-defined nose or plunging anticline. The underground geology, based on well records, show a well-defined closure. The surface, in other words, did not show any closure, in fact showed a gentle dip to the west, and the underground contours based on the sand records show closures.

R. H. JOHNSON.—I quite agree that where you have a series of sands one below another showing oil or gas, you have very strong evidence indeed that you have a strong structural feature. In this instance, if there was a terrace, the terrace indicated a more marked fold below. I am inclined to think that a terrace in the sand would not be important; its importance lies in the hint to us that we have a more marked fold down below the actual producing sand.

The mere arrest of movement of the oil up dip by the terrace, can hardly back up oil enough to make a thick enough pay to be important.

M. M. THOMPSON, Morsemere, N. J.—One point that might well be brought out in connection with the data presented by Mr. Hager is, that the surface formations are more nearly parallel with respect to the oil sands in the case of Oklahoma than most other oil fields. That is really the fundamental feature of structure which makes surface geology as valuable as it is here.

There is pronounced false bedding in practically all of the thick sandstone beds, and extensive areas are blind geologically, due to the lack of sufficient satisfactory outcrops. However, we can frequently follow for many miles a thin layer of limestone which serves as an excellent horizontal marker (or key bed) and basing our elevations on this we can work out our changes in dip and determine with reasonable accuracy the surface folding where such a bed is found in the area examined.

Wherever convergence is more pronounced than surface folding, geology based on surface structure alone is of little value in "wild catting." This was demonstrated to me very clearly in West Virginia. I found the thickening to be in the opposite direction from the prevailing dip of the surface beds, the condition being that the formations assume the shape of a folded wedge rather than that of a folded blanket-type series. My structural map of the deep oil sand was worked out from reliable well records in developed areas. In comparing this map with the structural map of the same region based on surface formations, I found that in many important instances the surface dipped as much as 100 ft. per mile in one direction while the oil sand dipped in the opposite direction. This of course results in a discrepancy between the apparent and actual structure in the oil sand, a fact that is well realized by the operators.

In the case of Oklahoma it is impossible to obtain as satisfactory well records as could reasonably be desired. In my first investigations in Oklahoma, I had hoped to base my work largely on well records. I

obtained from two of the large old established companies about 100 records covering the district that interested me. Not a single record gave the elevation of the top of the hole, and the classification of the key beds encountered in the wells were not reliable. Naturally such records are of little value except for approximate comparison over widely scattered areas.

While we fully appreciate the fact that in new, undeveloped areas, information derived from well records will be necessarily meager, still the geologist must obtain all the data of that nature he can to determine as far as possible what allowance must be made for convergence, pinching out of the sands and other underground conditions that would modify to some extent at least the significance which would otherwise be placed in pure surface indications.

H. A. WHEELER, St. Louis, Mo. (written discussion).—In the tabulation of 72 pools in Oklahoma, the work of the geologist seems to be subordinate to the results of the wild-catter, as the geologist is given credit for locating 31 of the pools, or say 43 per cent., whereas the wild-catter is credited with having opened up 37 pools, or 51 per cent., while 6 per cent. are uncertain.

To one who is acquainted with the Oklahoma fields, like Mr. Hager, this apparent greater success of the "shooting in the dark" system of the wild-catter is well understood, as the old pools in Oklahoma were discovered before the services of a geologist were utilized. For until 4 years ago, the geologist was conspicuous by his absence in the mid-continent fields, as the operators not only had no faith in him, but he was the laughing stock of the "practical" oil men, who then monopolized the field, and was classed with the divining-rod man and other fake oil-finders. Since the great success of the geologists in the famous Cushing pool, the change that has come over the old-time oil operators is revolutionary, as the majority of the large, successful operators now seek and utilize their services where formerly they would have been ashamed to consult them. This is best evidenced by there being a staff of at least 200 in the geological departments of the larger companies, or in consulting offices, where there was one lone, but determined, "voice crying in the wilderness" 4 years ago. Hence the seeming discrepancy in the relative success of intelligently prospecting for oil and "going it blind," as most of the pools were opened in the pre-geological era. For out of 37 pools discovered by the wild-catter, 24, or 70 per cent., were found prior to the employment of geologists.

Three pools credited by Mr. Hager to the wild-catter, or Drumright and Dropright (Cushing), and the Boynton pool, which latter is rated as "questionable," were the result of geologic aid, according to the best information available to the writer. Transferring these three pools to the

credit of the geologists and deducting the 24 pools opened up before their advent, this gives the geologist the credit of opening up 34 out of 48 pools within the past 4 years, or 71 per cent. of the successes.

The wild-catter will always be useful, although extremely costly, in discovering new oil or gas pools, as the lense type of structure can only be found by their "hit-or-miss" system, while some terraces are so difficult to locate, or the evidence is so dim, that most geologists do not have the courage to recommend their drilling, and their future discovery will largely rest with the bolder and more courageous wild-catter.

M. M. THOMPSON.—Another topic which might be properly brought up in this connection is, the kind of advice a young geologist is frequently called upon to give.

Many companies in Kansas and Oklahoma have recently organized large geological staffs made up largely of recent technical graduates. The majority of these men have either not had much experience in other fields or have not had an opportunity to see their judgment confirmed by the drill. They, on the one hand, are often not versed in the logic of the "practical operator's wild-cat" methods, and their employers, on the other hand, are not sufficiently acquainted with the technical and difficult problems a geologist sometimes encounters.

Consequently, it frequently happens that a geologist gets an assignment where there is not sufficient evidence to justify a conclusive report. Some of these young men in their eagerness to acquire leases will report their personal guess, and depreciate the meager and unsatisfactory observations they made in the field. They take the optimistic attitude that if the guess was a lucky one it will be a feather in their cap. However, if the drill proves the land to be worthless, the employer will most assuredly say "the geologists recommended that lease, they gave us bad advice and we will not place so much confidence in them the next time." The practical operator still assumes a patronizing semi-tolerant attitude toward the geologist, which has replaced the former strong prejudice.

My policy has always been in a case of doubt to merely say that the geological facts are thus and so, and let the leaser do his own guessing. If geologists as a class would confine themselves to actual geological facts, they would stimulate far more confidence in themselves and in the profession than they have in the past.

If a geologist must turn in results that are largely guesswork, even in a blind and difficult area he is worse than useless to his employer. Most operators would prefer to obtain reliable information from a scout and do their own guessing, than to depend upon him.

THE CHAIRMAN (M. L. REQUA, San Francisco, Cal.).—I do not suppose the geologist is infallible by any manner of means; he has to do a certain amount of guessing. I made that statement at a meeting of the geological

students of the University of California and I was taken up on it, and told that the function of geology was not a guess.

Now it seems to me that the function of the geologist is to eliminate as much of the chance as possible. I do not suppose there is any individual engaged in the oil business or who contemplates putting money into the oil business who is so foolish as to suppose that the geologist is going to hit it right 100 per cent. of the time, but I am very sure that he is going to hit it right a great deal nearer 100 per cent. of the time than the individual who has not any geological education.

I have been all through that period of decrying geologists in California and I have seen the same revolution take place there that has occurred, you gentlemen have said, in Oklahoma. The geologists have come to be recognized as an extremely important factor in the oil industry, just as they have been recognized for a number of years in the mining industry.

DORSEY HAGER.—In Oklahoma, we have found in summing up the pools that have been discovered by the geologists and those drilled on the geologist's advice, that in one in three of the pools the tests have been successful. That is, $33\frac{1}{3}$ per cent. of the geological tests have been successful. Now I am fairly confident, though I do not want to make any statements that anyone can attack me on, but we have found that there is a great deal of structure that does not exist and if those earlier errors of the geologists in Oklahoma and the later ones too, due to careless work, insufficient time, and insufficient knowledge of the country—if those errors are eliminated, and they have been largely in the past 2 years, the percentage of the successes grows much larger and the geologist gets one success in two, tests eliminating from this calculation structures that are known not to exist.

I. N. KNAPP, Ardmore, Pa. (communication to the Secretary*).—In reply to the discussion of his paper at the above meeting, Mr. Hager said in substance that I was on record in 1912 as saying there was no anticlinal structure in Kansas or Oklahoma. In the *Journal of the Franklin Institute*, November and December, 1912, appeared an article on Natural Gas and Other Bitumens written by myself. I made reference to the anticlinal theory and quote from what I said in 1912 as follows: "Through the investigations of geologists working in the great Appalachian field the structural or anticlinal theory of the accumulation of oil and gas was evolved.

"To such fields as the theory applies it is of economic importance, and the geologist should be consulted to locate favorable points in advance of any drilling."

I then gave a plan and section of a few wells drilled by myself in Kansas and said that the anticlinal theory did not in my opinion apply to

* Received Mar. 1, 1917.

this particular oil pool. The fact is, I was told by a prominent geologist that I had no business to find oil where I did, that it was contrary to all theory.

Quoting again from what I said in 1912: "The conclusion is that each oil and gas field must be considered by itself, both from a geologic and operating standpoint. The geologist, operator, or driller expert in one field has to learn the business over again when he goes into a new field.

I think what I said publicly in 1912 holds good today. If anyone wishes to know what Mr. Hager said about that time on the anticlinal theory, I would refer them to his article on the Value of Geology in the Petroleum Industry, which appeared in the *Mining and Engineering World* of Sept. 2, 1911.

The Need and Advantages of a National Bureau of Well-Log Statistics

Discussion of the paper of W. G. MATTESON, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 287 to 290.

BENJAMIN L. MILLER, So. Bethlehem, Pa.—In discussing the question personally with Mr. Matteson I have found that he simply proposed this as a suggestion and he recognizes, as well as a great many other oil geologists and oil operators, that to put into practice such a suggestion would mean the overcoming of many difficulties. Yet perhaps it may be possible to overcome the various objections that may be offered.

Undoubtedly one of the most serious objections is that of incomplete and inaccurate well records. Faulty records are worse than no records at all. Perhaps, however, definite instructions and insistence on their observation may remedy this defect after a time.

THE CHAIRMAN (M. L. REQUA, San Francisco, Cal.).—This subject is a very important one, but just how far the National Government can go in taking action is a question. It involves, of course, the question of State rights. California, I think, has done very well up to date and probably will do better in the future. The law that was passed at the session of the Legislature 2 years ago is working as satisfactorily as anything of that kind can work with so little experience to guide the framers of the statute.

We recognize in California the vital necessity of conserving the oil of the State. We have come to the conclusion that there is not as much oil in California as we thought there was. In the month of December the State drew on its oil stocks to the extent of 1,800,000 bbl. and I think that the January statistics will show probably as much more. There is more or less of a panic at the present time among the large marketing concerns regarding the question as to where they are going to get their oil for future requirements.

The reversal of form within 18 months has been startling. The whole viewpoint of the large marketing companies has been changed. The Standard Oil Co. has been very busily engaged in acquiring property; the Associated Oil Co. spent upward of a million dollars last year in drilling without appreciably raising its production. The Union Oil Co. that has large contracts in Chile has recently made an arrangement with the Mexican Petroleum Co. to supply practically all if not all of that Chilean business from Mexico, and we of California at the present time are very much awake to the necessity of conserving the oil resources, and property, wherever we can. Proper well locations, proper supervision as to the shutting off of water and plugging of wells that are making water, we believe are a very essential part of the program along with proper well logs.

We have also come to the conclusion that that work cannot be left to the individual producer. It is a curious thing, but no producer is ever willing to admit it is his well that is causing trouble, it is always his neighbors'. Under the existing law, I think the producers of California are well satisfied that a step in advance has been made, and I am sure they will never go backward; they will make the law, if anything, more stringent for future operation, but as to just how far the National Government will enter into that field is a question that seems to me to be very debatable.

I. N. KNAPP, Ardmore, Pa.—It is very true that the average oil operator and driller have not fully appreciated the importance of keeping accurate well logs in the past, but they are all getting educated up to the point of being appreciative.

One is obliged to depend on the coöperation of the oil operator, the contractor and the driller to get accurate well logs and samples. Their hearty coöperation can be had to sustain reasonable laws and regulations as enacted. Impractical laws and regulations soon become dead letters, and are worse than none.

DORSEY HAGER, Tulsa, Okla.—I know good well logs are kept in California. Oklahoma is doing its best now to get good well records. A campaign has been going on, the companies are demanding good records from their men and it is really surprising to note how excellent the records are now, compared to what they were a few years ago. I think it will be a matter of just another year or so before Oklahoma and Kansas will take care of that well-record question themselves without any national legislation. By the time national legislation is put through, the States will have accomplished it themselves.

D. B. REGER, Morgantown, W. Va.—We have had a great deal of work from the oil companies in getting well records in West Virginia. I suppose we have copied maybe 5,000 or more records from various

companies' offices and so on where we could get them. We do not have much trouble now with the large companies; they usually have their records in good shape and they usually give them to us willingly because they realize the value that their preservation and use in making up the structure maps have for their companies. Our main trouble is with the wild-catters who often have inexperienced drillers who do not keep the right kind of records. We have often wished, in our State work, that we had some way to get these records without fail. We have always regarded compulsory legislation as hopeless because so many of the operators think that it is an infringement on their rights.

I found an amusing circumstance 2 or 3 years ago. Some men who had been working in a small way, independently, in one of the counties at which we were at work, refused to let us have any of their records. They did so abruptly because they said they had spent too much money drilling these wells and getting this information to give it to anybody. In the meantime we let the matter rest, for about a year, and sent this particular operator a copy of one of our reports on a county adjoining his own. He went in there, speculated on the subject matter of the reports and made a lot of money that year. The next year when I went back to his office he turned over all his record books to me, left his office and told me to help myself. We usually find a little work of that sort generally brings the desired result.

Of course the wells drilled where no records are kept are gone, and cannot be had. Such a provision as suggested in the paper before us would have a good result, and would of course remedy that.

R. H. JOHNSON, Pittsburgh, Pa.—It seems to me that there are some wells that are so important, for instance, the first test in some foreign concession, that a company is making a great mistake when it relies upon the drillers merely for the log. In a number of such cases I believe an observer should be maintained at the well for the purpose of keeping the log. But of course there is a limit to the possibility of this method and in general we are compelled to fall back, as has been said, on education. I think we ought not to be cynical about well logs. They are so vitally important we must keep at this educational process until drillers keep good logs. The trouble is, of course, the attitude toward this work of many drillers and contractors. It seems to me the executives of the company ought to take the attitude, "If these men are not going to do it we will get somebody that will."

S. A. TAYLOR, Pittsburgh, Pa.—Before leaving the question of records of wells, I think there is one item that it might be well to have brought out. In Pennsylvania, Ohio and West Virginia, and some of the States that are mining coal, they have already passed laws, at least Pennsylvania has, in connection with the mining of coal requiring that the wells

shall have kept a very accurate log and if the well is abandoned it must be plugged below the lowest seam of coal. At the same time an accurate location of the well is determined for the reason that if it is abandoned they must know the location of it in order to protect the mining of coal. I, too, realize that this matter comes back to States Rights and is governed and controlled by the police powers of the State. Consequently each State must pass its own laws, but I am sure those States engaged in the mining of coal or other minerals ought to have an exact location of all wells and complete logs of them.

Another matter of great importance to oil and gas companies, and which has come up, especially, in the State of Pennsylvania and also in West Virginia, is the amount of coal or mineral that is to be left around a well in order to protect it when operating or mining the materials surrounding it. Up to the present time I think the amount that has been decided to be left is very largely guesswork. In some places they leave a certain stipulated amount and a short distance away a larger or smaller amount. Even in different counties in the State of Pennsylvania they require a different amount. This has been decided largely, or rather arrived at, by some petty court decision or agreement and no method really scientific has been applied to this scheme. I think that by keeping an exact record of the overlying strata it will be possible to determine how much coal or other mineral should be left around a well in order to protect it and make it safe for the mining of whatever material is to be mined, largely coal on account of the gas leakage into the mines, etc. On the whole, I am sure that a careful record of wells will be an advantage not only to you as operators or drillers of oil and gas wells but to the entire mining fraternity.

C. NARAMORE, Washington, D. C.—I have been on both sides; I have been a geologist for one of the larger companies, and have been asked to scout around and get well data. Later I have had charge of the work and had the privilege of telling the drillers what I wanted and at first getting the same treatment that the speaker before me did. I find that if you try to drive a driller you lose out; it is a question of team work. You can educate a driller; they are better than the average worker. If you will let the average driller know that you are very anxious for that record he will give you a record which is valuable.

The use of the 5-ft. stick covers only a small interval of the well log because ever so often, if so requested, the driller will string in and check the measurements carefully. I have had to hire many drillers and only had occasion to discharge three or four for absolutely falsifying the logs, after I have worked with them a while.

The driller who is not familiar with log recording is usually opposed to it at first.

In the matter of States collecting logs, I was in the California field when that law was put into effect. As Mr. Requa said, the records in California are accurate, but I am sorry to announce that from now on, they will not be as accurate because of rotary drilling. Where we used to take from a year to a year and a half to drill those deep wells with standard tools, 3,000-ft. holes are now put down in from 30 to 35 days and drilled by contract. On different occasions I have asked for a well record from a contractor and from his reply knew that he had failed to keep a detailed log. In fact, I have received copies of logs of the same well at different times and found that no two were alike. Other contractors keep splendid records. That is leading to very serious practice. On one of the largest and most productive anticlines in California, a large company was drilling wells with the rotary method in 1911 and 1912. The man in charge had his idea where the oil should be found. They drilled through the big oil sand at night, went 200 ft. below and found no oil. Ever since that day that well has been considered as limiting the possible productive oil land on that limb of the anticline. Just a year ago now the superintendent desired to make a producer of this well or to salvage the casing—as casing is at a premium on the coast now.

His engineer had constructed a very accurate peg model, on which it appeared that the oil sands in this well had been passed through and cased off.

Here is a case where practical geology meant something to that company.

With our theoretical point of view he began to abandon this well 50 ft. at a time. One hundred and fifty feet up the hole he found a little trace of oil. After baling a number of days, he then proceeded to pump the well, and for a week nothing but rotary mud appeared. To make a long story short, he brought in a 400-bbl. well of 30 gravity oil.

It meant that the limit of the productive territory near this well had to be extended, so the keeping of well logs is all that we contend. A poorly kept log is better than no log and allows for a partial correlation. If the records are approximately perfect, when you come to project them on a scale of 100 ft. to the inch, they furnish an approximation which is often of much value.

I am sorry to learn that the keeping of logs is handled so carelessly in some of the large producing fields. I may say that in the State of California we have now some sort of a log for nearly every well and the record of enough wells so that we can project almost anywhere within a proven area.

CHAIRMAN REQUA.—How are you going to correct that question of the rotary drilling?

C. NARAMORE.—It is an economic difference; a difference of cost. No

correction is possible. Where the rotary method is the more efficient, it will be used. The same procedure is in order as suggested for cable tools, the coöperation of the driller must be enlisted to furnish us the best record obtainable.

Until recent years, each company drilled its own wells, now many companies have their wells drilled by contract. Some operators are careful to secure a dependable log from the contractor, others are careless in the matter—but with the State Mining Bureau regulating such matters, the State of California is in a better position in this regard than many other States.

I. N. KNAPP.—If you wish to get an accurate log with samples of an unconsolidated formation at any particular horizon in drilling with the rotary, it is necessary to take a core. It is a pretty hard proposition to do this, but it can be done. Arthur Knapp and myself working together finally hit on a design of bit and core barrel with which a core could be taken. The cores we finally secured revealed that we had been fooling ourselves as to what the formations actually were when we were guided by the cuttings from the overflow only. I have many times saved samples from wild-cat wells, also washed out and preserved fossil shells, but the geologists examining the same never agreed on an interpretation of the record.

Geology is not an exact science, and when experts cannot make demonstrations that come somewhat near agreeing, what is the use of expecting that the layman can correctly classify wild-cat formations for record?

The A. T. elevation of the well, the lengths of pipe or casing used, and the depths, can be made matters of record.

Drilling in a dry hole with the cable tools, sudden changes as from a soft shale to limestone can be easily noted. In a great many cases one formation shades off into another and the point of change is guess work, particularly in rotary drilling. Naturally the operator makes measurements necessary for his protection as, for instance, the lengths of casing used and the top of the oil or gas sands.

CHAIRMAN REQUA.—There is no doubt that the rotary drilling in California is very fast superseding the old standard tools. It is purely an economic question and there is so much in favor of rotary, that broadly speaking I think the standard really is practically obsolete for all deep-well work. It is unfortunate that we are not going to get the well records that we have had in the past.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Ore Deposits of the Boulder Batholith of Montana

A GENETIC DESCRIPTION

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(St. Louis Meeting, October, 1917)

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* Assistant Geologist, Anaconda Copper Mining Co.

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INTRODUCTION

THE purpose of this paper is a comparison, based on genesis, of the ore deposits associated with the igneous rocks of the central Montana Rockies. Considered separately, without attention to their origin, the orebodies appear to be distributed haphazard, with no fundamental laws controlling their position and character. It is believed, however, that if all the factors be weighed, this apparent confusion will to a large degree disappear. The first essential to such a comparison is detailed knowledge of the individual mines, as to their geographic position (see Fig. 1), their mineralogical character, and in particular the derivation of their ores.¹

The authors of this paper have pursued such an investigation for the past 6 years, and have accumulated a mass of evidence that enables a preliminary correlation to be attempted. It appears that the majority of ore deposits are genetically related to one or another of the igneous

¹ In the collection of evidence for this paper the authors have personally examined a large proportion of the ore deposits of the Boulder Batholith. Published descriptions, however, particularly those of the U. S. Geological Survey have been used, credit being given in each instance. Particular mention can hardly be made of the universal kindly assistance afforded the authors in their quest for material for this paper, but in no instance was such help refused, although in many cases it necessitated a considerable effort on the part of the donors, or the use of maps and other material not heretofore made public. Particular thanks for the contribution of facts and for criticism are due to Messrs. R. H. Sales, C. W. Goodale, F. A. Linforth, M. H. Gidel, and D. C. Bard. The great mass of detail which would be necessitated by the publication of all the supporting evidence, would, in the authors' opinion, obscure the more important points. Therefore, in so far as possible this evidence has been given graphically in the accompanying plates. It has been the endeavor of the authors to make these plates self-explanatory, in order that written descriptions of them might be omitted from the text. These plates are grouped at the end of the paper.

periods of the region; that when grouped according to this relationship they display resemblances and contrasts that can be traced with certainty to the conditions of their origin; and that common laws governing the derivation of their varying characters can be reasonably deduced. In short, a true genetic classification throws considerable light on the following subjects:

1. The comparison between igneous periods and their accompanying ore deposits.
2. The progression of type of ore deposits within a single igneous period.
3. The relation between the mineralizing activity and the form of igneous intrusions.
4. The geographic variation of vein-forming solutions of common origin.
5. The vertical distribution of sulphide minerals in veins formed by ascending solutions.

This paper, therefore, will summarize as briefly as possible the more significant features of the Montana mining districts, drawing from these such conclusions upon the natural history of magmatic ore deposits as appear inevitably to follow.

PART 1.—GEOLOGIC BACKGROUND

A. INTRODUCTION

From the earliest development of the mining industry in Montana, the close association of the ore deposits with rocks of granitic type has been observed.² In fact, the presence of "true fissures" in granite walls was eagerly sought, and where found was seized as indisputable evidence of the value and permanence of the prospect. The rise of many great mining camps has testified to the foundation of truth beneath this assumption of the prospector, and Bannack, Argenta, Unionville, Wickes, Marysville, and Butte, have successively found granitic rocks a congenial home for ores of the most diverse types.

More recently it has become rather generally accepted that the major portion of these granite outcrops represent a single great mass of plutonic rock and in many instances identity has been established by continuous surface exposure. To this great granitic mass the term Boulder Batholith, originally applied to a limited outcropping area, has been extended.

B. GENERAL GEOLOGY

The Boulder Batholith represents the middle phase of an igneous cycle that during Cretaceous and Tertiary times was the dominating geologic event of the Montana Rocky Mountain region. The rock types of closely associated eruptions pass through a similar sequence; first,

² See the volumes of *Mineral Resources of the Western States* for the years 1866 and 1885.

gabbro, diabase, andesite; second, diorite, quartz-monzonite, aplite (and quartz-porphyry); and third, dacite, rhyolite. These are generally grouped under the terms andesite, granite and rhyolite series, a succession which is found throughout southwestern Montana.

The several eruptions and intrusions are intimately related to the great mountain-making epochs of the Cordilleras, as may be seen by reference to the following chronological table¹ (see Fig. 23).

1. Middle Cretaceous. Period of main Rocky Mountain folding, and formation of large earth-folds in northwesterly direction.
2. Middle-Upper Cretaceous. Extensive erosion and beveling of folds.
3. Upper Cretaceous. Andesite eruption. Deposition of extrusive lavas and breccias west of Rocky Mountain front and formation of tuffs and andesitic sediments on the plains.
4. Upper Cretaceous. Local intense erosion and formation of coarse andesite conglomerates.
5. Upper Cretaceous. Thrust faulting along northwest lines, and local intensification of folding.
6. Eocene(?). Intrusion of Boulder Granite Batholith.
7. Eocene. Extensive erosion, approximating peneplanation.
8. Oligocene, Miocene. Normal faulting; accumulation of river gravels and lake silts; early rhyolite.
9. Pliocene. Same conditions, with extrusion of later rhyolite and dacite.
10. Pleistocene. Two or more glacial stages in the mountains.

It will be noticed that the major igneous periods—the andesite and granite—successively followed the epochs of more violent tectonic disturbance of the region. Thus the andesite eruption, involving the transfer of tremendous masses of rocks from the depths to the surface, succeeded the period of main Rocky Mountain folding, while the granite, even more extensive in its effects, followed the later state of earth-movement represented by the thrust faults. The conclusion that the igneous activity is thus in some way dependent upon the changing stresses and pressures of the mountain-making periods can scarcely be avoided. Further corroboration is afforded by the later rhyolite, which follows closely the period of normal faulting.

C. IGNEOUS ROCKS

The Andesite stage of igneous activity is represented by a very diversified group of rocks: gabbro, basalt, diabase, augite, andesite, andesite porphyry with varying types of ground mass and phenocryst, amygdaloidal andesite, andesite breccia and tuffs are among the forms encountered.

These rocks were erupted from two main vents, one in the Crow

¹ Adapted from The Boulder Batholith of Montana by Paul Billingsley. *Trans.* (1915), 51, 31.

Creek Mountains⁴ near Elkhorn, and the other in the vicinity of Thunderbolt Mountain, east of Deer Lodge (see Fig. 2). In these regions the gabbro, in the form of dikes and intrusive masses, is most abundant; around the vents the andesite breccia is widely distributed, and radiating from these peaks the surface flows spread out in overlapping layers covering the many square miles of the andesite areas. In general, these series of surface flows pass from basalt below to andesite porphyry in the later lavas, and suggest an increasing acidity of magma throughout the period.

The granite stage followed the andesite after a considerable interval in which much erosion, and additional earth movements, occurred. The igneous rocks of this period are closely associated and follow each other with very slight breaks. A basic diorite, locally grading down into pyroxenite and pyrrhotite, is the earliest type. It is found in part as the basic contact of the batholith itself, and in part as inclusions, probably representing a deep-seated earlier chilling of the magma. This is followed by the main intrusive rock, a quartz-monzonite, variable in texture but essentially uniform in composition throughout the extent of the Boulder Batholith. This mass was intruded at a comparatively low temperature (500° C.) and reached its present position, high up in the sedimentary series, by the mechanical action of the magma in breaking off blocks of the roof and penetrating upward into the space thus vacated (magmatic stoping). In general the contact is more basic than the main mass, but locally siliceous stringers of the later segregations have pushed out beyond the borders. The crystallization of the quartz-monzonite magma was followed, while the rock was still warm and possibly viscous, by widespread local intrusions of a more siliceous residue. This forms the aplite, and comprises possibly 8 or 10 per cent. of the total mass. It abounds in pegmatitic segregations, which occasionally pass into nearly pure magmatic quartz. Pegmatite, quartz, and tourmaline, in varying proportions, appear to be essentially contemporaneous in the later aplite stages. Very rarely intrusions of quartz-porphyry are found. This rock is slightly more basic than the aplite, from which, however, it differs mainly in texture. The two are closely associated in the Butte mines, where the quartz-porphyry is best known. Cooler contact phenomena, and subsequence to jointing common to both granite and aplite, place the quartz-porphyry in a later period, but chemical similarity and evidences of transition with depth suggest that the sources of the latter rock must be of aplite type and may be of identical origin with the aplite segregations of the upper horizons.

The final fractionalization of the granite period is represented by metalliferous quartz, which in its distribution closely follows the aplite

⁴ R. W. Stone: Geological Relations of Ore Deposits in the Elkhorn Mountains, Montana, *Bulletin of the U. S. Geological Survey* (1911), 470, 75.

D. CLASSIFICATION OF ORE DEPOSITS⁶

Each period of igneous activity stimulated the formation of ore deposits, the common succession being, roughly, segregations and disseminations within the magma, contact deposits in the intruded rock, fissure veins in the igneous rock, and later mineralized faults. A genetic classification will therefore consider this order, together with the cycle of igneous rocks. The following is adopted here in preference to a system based on form of deposit or nature of vein filling.

1. *Andesite Period.*

1. Andesite Phase.

A. Disseminations.

1. Amygdaloids.

2. General Impregnation.

B. Contact Deposits.

C. Fissures.

2. *Granite Period.*

1. Granite Phase.

A. Segregations.

1. Magnetite.

2. Pyrrhotite.

3. Quartz.

B. Disseminations.

C. Contact Deposits.

1. Contact-Metamorphic

2. Replacement.

D. Fissure Veins.

2. Aplite Phase.

A. Segregations.

1. Pegmatite.

2. Tourmaline.

B. Disseminations.

C. Fissure Veins.

1. Quartz-Tourmaline Veins.

2. Quartz Veins with Sulphides.

3. Fault Veins.

⁶ Compare Knopf's classification in *Bulletin of the U. S. Geological Survey* (1913), 527, which follows:

A. Older Ore Deposits (Late Cretaceous).

1. Magmatic Deposits.

2. Contact-Metamorphic Deposits.

3. Lodes and Veins.

(a) Tourmalinic.

(1) Silver-Lead.

(2) Silver-Copper.

(3) Gold.

(b) Sericitic (May Be Phase of Tourmalinic).

B. Younger Ore Deposits (Not Classified).

Knopf's bulletin will receive further discussion in the descriptions of mines and districts.

3. Quartz-Porphry Phase.
 - A. Disseminations.
 - B. Fissure Veins.
 - C. Fault Veins.
3. *Rhyolite Period*.
 1. Early Rhyolite Phase.
 - A. Disseminations.
 - B. Fissure Veins.
 2. Dacite Phase.
 - A. Contact Deposits (Redistribution of Earlier Ores).
 - *B. Fissure Veins.

It will be noted in this that the ore deposition reached a maximum diversity in the granite and aplite stages. In size and widespread distribution, also, the orebodies of these periods are foremost. The types significant of strong igneous activity, the segregations, disseminations, and contact deposits, are most noticeably wanting in the later igneous phases. Deposits traceable to magmatic waters are persistent throughout, and accompany even the later intrusions of small size and slight effect. In general the variety and quantity of ore deposition is directly comparable with the size and importance of the igneous stage within which it occurs.

PART 2.—THE ORE DEPOSITS

ANDESITE PERIOD

A. Andesite Phase

Orebodies in the andesitic rocks occur as disseminations, a type represented by the native copper of the Baggs Creek district; as contact deposits, illustrated by the Iron Mines of Elkhorn Mountain;⁶ and as blanket fissures, either in andesite breccia, as in the case of the Evening Star mine near Elliston, or in amygdaloidal flows as in the Emery (Zosel) district. Many mines throughout the region have developed fissure veins in andesite walls, but in the majority of these, the ore can trace its origin to nearby granite intrusions, the veins of granite origin extending

⁶ The Ward mountain mines, in Madison County, are located upon replacements of schist by quartz, and auriferous pyrite along the flanks of an andesite sill. These mines, of which the McKee and Tucker are the chief, probably fall within the Andesite Phase of the present classification, but the impossibility of determining the age of the sill prevents their inclusion in the above list.

up into the andesite covers. These deposits, therefore, fall under the granite subdivision of the genetic scheme.⁷

1. *Disseminated Deposits*.—Magnetite and native copper are found in disseminations through certain portions of the andesite rocks. The former occurs as a primary mineral, closely associated with the ferromagnesian silicates, and doubtless represents an early separation of the excess iron from the magma.

Native copper is found in part in similar mineralogical associations, the metal occurring within the augite phenocrysts or groundmass. Frequently, however, it appears in the amygdaloidal upper portions of the flow, with quartz, calcite, and zeolites, and occasionally fills recent cracks in the rock. These latter occurrences cannot be regarded as contemporary with the igneous rock, and it appears probable that wherever found the native copper represents a later concentration of the original copper contents of the flows.

Copper Hill, on Baggs's Creek, east of Deer Lodge, Mont., may be taken as an example of the disseminated copper deposits (see Fig. 3). The metal is found in a rugged hill, which faces the creek in a cliff about 600 ft. high. From the talus slope to the crest, 20 separate flows of andesitic rock can be distinguished, each fine-grained at the base, porphyritic toward the center, and amygdaloidal on top. They dip gently to the northwest, away from the common vent at Thunderbolt Mountain. The rock types are prevailing basic and represent the earlier stages of the andesite eruptions. Basalt, diabase, hornblende andesite, and augite andesite predominate, the latter rock, rare at the base, increasing in amount toward the hilltop. Andesite breccia to the south and east contains fragments of the basaltic forms, but none of the augite andesite, which, therefore, apparently initiates the later outflows of which the light-gray andesite porphyries and latites are so characteristic. Among the successive flows three are of district appearance, with a characteristic pea-green groundmass, and large dark-green augite phenocrysts.

Native copper in appreciable amount is restricted to limited lenses within these flows. The metal occurs in the groundmass, in the augite phenocrysts, and in the amygdaloidal cavities, associated, in the latter

⁷ List of mines in Andesite Phase of mineralization:

Baggs Creek—Copper—East of Deer Lodge.

Iron Mine—Magnetite—North of Elkhorn.

Tacoma—Galena—North of Elkhorn.

Kineo—Galena—South of Elliston.

Hidden Treasure—Gold South of Elliston.

Evening Star—Galena—South of Elliston.

Carbonate Hill—Galena.

Butte Caroline—Galena.

Pierce's Prospect—Galena.

} Zosel district (Emery).

case, with quartz, calcite, and zeolites (rare). It unquestionably represents a concentration by comparatively cool waters of the copper originally widespread as a constituent (0.02 per cent. or less) of the original rock. The cupriferous lenses, with an average grade of 0.8 per cent. copper, form less than 0.1 per cent. of the mass of the cliff, so that the immediately adjacent rock is entirely adequate to supply the metal required. It is, in fact, probable that this concentration and transportation of the copper took place only within the several flows in which it is now found—a conception supported by the barrenness of adjoining flows of equal porosity, but of slightly different rock type. Joints and cracks, however, have provided channels in which recent surface waters have transported and precipitated the metal. As to the source and character of the original solutions that at once dissolved the copper silicate of the original rock and precipitated locally the native copper of the enriched lenses, little knowledge can be gained. The common explanation of a submarine extrusion, with a lava attacked by sea water, will not suffice for this series of flows upon a land surface. Between the alternatives of a meteoric circulation stimulated by residual heat, and magmatic waters of low temperature, field evidence gives little choice.

2. *Contacts*.—The general contacts of the extrusive andesite with the sedimentary rocks show little metamorphism or mineralization. The intrusive phases, which exert greater influence on the older formations are, moreover, so surrounded by breccias and flows that their contact with the sediments can rarely be seen. Andesite dikes in association with ore deposits are more common, but in most of these instances the age of the andesite is indeterminate.

The best examples of an undoubted contact deposit of the andesite stage are the so-called iron mines on Elkhorn Mountain which have been thus described⁸ (Fig. 10).

"On the north side of Elkhorn Peak, there is an iron mine on the contact between andesite and marble, from which a considerable quantity of ore has been shipped to smelters for flux. The ore is along the bottom of the marble mass and is probably a replacement of it. It is a fine-grained magnetite showing grains of chalcopyrite and garnet."

This apparently represents a true contact deposit, formed by the contribution of iron from the igneous rock to the metamorphosed limestone.

3. *Fissures*.—A short distance south of the Baggs Creek deposits of native copper are the mines of the Emery district.⁹ These are located upon fissure and blanket veins within the andesite and may be taken as examples of the fissure mineralization of the andesite stage.

⁸ R. W. Stone: *Op. cit.*, 97.

⁹ See Fig. 3, Map of Emery Mining District.

The ore deposits fall under four heads:

- (a) Fissure veins in andesite walls.
- (b) Mineralization along bedding of flows.
- (c) Mineralization of amygdaloidal lavas.
- (d) Mineralization of andesite breccia.

The three latter types represent an extension of the mineralization from feeding fissures into permeable ground.

(a) The main fissure of the district lies in an approximately central position, cutting the flat dipping andesite flows in a northwesterly direction. Its dip is vertical. The ore, about 10 ft. wide, consists of pyrite, galena, sphalerite, and chalcopryrite, in a gangue of quartz and calcite. The walls are tight, with no gouge. Intimate intergrowths of the sulphides, coarse crystallization and occasional quartz-filled vugs, suggest formation in high or moderately high temperatures. Gold and silver are present in appreciable quantities, the latter, within the zone of secondary enrichment, reaching 40 oz. or more to the ton. Development to a depth of 400 ft. has failed to disclose any material change in the vein filling, other than a diminution of the silver as the primary ore was reached.

(b) and (c) Southward of this main fissure the Carbonate Hill mine has developed a large orebody upon the bedding contact of two andesitic flows. This dips northward at a flat angle (30°). The shoot is about 2,000 ft. long and has been followed on the dip for a distance of over 1,000 ft.

The vein material varies in width from a few inches to several feet, with tight walls and no gouge. On the foot wall the amygdaloidal andesite has been extensively mineralized; pyritization and silicification of the rock is thorough; and the amygdules contain a filling of quartz, calcite, galena, spalerite, millerite, arsenopyrite and some zeolites. Veinlets of quartz and the sulphides are also common throughout this altered rock. The vein itself is well banded; layers of the quartz, calcite, and rhodochrosite gangue alternate with bands of the sulphide minerals, with frequent vuggy faces of quartz and calcite crystals. Apparently at least one reopening of the fissures is represented. The sulphide minerals, pyrite, galena, sphalerite, chalcopryrite, millerite and arsenopyrite, form intimate aggregates of much finer grain than the typical ore of the vertical fissure, which, together with the occasional presence of zeolites, indicates lower temperatures during the formation of the bedded deposit. The hanging-wall flow, a dark hornblende andesite, is entirely unaltered.

Economically this orebody has been exploited for the silver which in the oxidized zone and for several hundred feet below water level has been enriched to a high grade. On the lower levels the diminution in silver was accompanied by an increase in sphalerite, and the ore shoot, while persistent geologically, ceased to be of value to the miner.

(d) A small mine north and east of the Carbonate Hill, in the same district, affords a convenient example of mineralization in the andesite breccia, although larger deposits are known elsewhere.¹⁰

¹⁰ The Evening Star mine south of Elliston.

In the type example, a layer of andesite breccia of irregular width is intercalated between a light-gray andesite flow and a dark hornblende andesite. The whole series dips 25° to the northeast, and movement has taken place along the top of the breccia. The broken and porous zone thus created has been penetrated by mineralizing solutions, which have selectively attacked the breccia groundmass and fault plane. Excess of galena and extreme fineness of texture mark this deposit and testify to the cool conditions of its formation. It carries much more gold than the blanket or fissure-vein deposits.

4. *Summary*.—In general, therefore, the andesite stage of igneous activity concludes with two types of mineralization; the first, a zeolitization of the igneous rock and concentration of its copper content, probably effected by the magmatic waters of dying vulcanism; the second, a contribution by magmatic solutions, which deposited in such favorable localities as fissures, bedding planes, amygdules, and breccias, a representative sulphide ore carrying universally galena, sphalerite, pyrite, chalcopyrite, arsenopyrite and millerite with appreciable amounts of the precious metals.¹¹ The distinction between the deposits formed by concentration of metal originally disseminated throughout the rock and those derived from the metalliferous residue of the igneous reservoirs is a marked one, both in mode of occurrence and in character of mineralization.

GRANITE PERIOD

A. Granite Phase

1. *General Description*.—The earliest ore deposits of the granite phase are the segregations from the molten magma. These may be basic, as magnetite and auriferous pyrrhotite at the margins, or acid, as the quartz segregations within the batholith. Disseminations, mainly of auriferous or cupriferous pyrite, near the basic borders of the granite, are essentially contemporaneous with these. Contact deposits, representing a contribution of metalliferous waters from the intrusive rock, follow closely in point of time. They may be subdivided genetically into the high-temperature deposits marked by garnet, tremolite, and the metamorphic minerals, with associated copper mineralization, and the lower-temperature deposits, generally of auriferous pyrite, replacing the intruded rocks to some distance from the granite contact. Fissure veins, formed subsequently to the crystallization of the granite, mark the last stages of its mineralizing activity. They are confined to the uppermost

¹¹ The gold-pyrite veins of the Radersburg district (Fig. 4) are not included here because although entirely within the andesite walls they show strong affinities with the granite type of vein filling and a source of the solutions within the granite is rendered possible by the proximity of such intrusions.

portion of the granite intrusions, and generally extend well into the overlying rocks. Quartz and pyrite, carrying gold, are the common minerals.

2. *Segregations*.—The approach of the basic rim of the granite to the extreme forms of pyroxenite, peridotite, and even pyrrhotite is not rare. It apparently occurs most frequently where a forerunner of the magma has widened into a peninsula, [nearly surrounded by cool sedimentary rock.¹² Under these conditions the borders of the igneous peninsula display the most basic phases, and in many cases pass into pyrrhotite at the contact itself. The ultra-basic types, and especially the pyrrhotite, may carry gold to a commercial degree.

In the Spring Hill mine near Helena, the characteristics of such a deposit are well shown.¹³ An outlying granite boss penetrates massive Carboniferous limestone with many irregular contacts, at least one projecting mass of the intrusive rock being well surrounded by the sediments. The igneous buttress shows basic faces on two sides with a central core of quartz-diorite. The gradation in either direction from the normal rock through diorite, peridotite, pyroxenite and pyrrhotite is a gradual one, covering 50 to 75 ft., and the gold content increases with equal uniformity as the contact is approached. The limestone is metamorphosed and cut by large and small dikelets of the basic rock, but the ore deposit is restricted to the igneous rock itself.

A similar pyrrhotite orebody occurs in the Golden Curry mine near Elkhorn; and is well described by Knopf.¹⁴ In this mine magnetite, as veins and segregations within the granite near its contact faces, affords the single somewhat doubtful example of this mineral as a product of igneous differentiation.

Contrasted with these ultra-basic separations from the magma are quartz segregations found with frequency throughout the granite areas. The largest, several hundred feet in diameter, lies a mile east of the town of Basin.¹⁵ It consists of massive vitreous quartz, with minute amounts of tourmaline and molybdenite, surrounded by typical unaltered coarse quartz-monzonite. Below the quartz mass is a porphyritic rock composed largely of orthoclase and quartz, probably representing the aplite stage. This has been penetrated by fingers of the quartz rock, which locally approach the composition of pegmatite.¹⁶ It is, therefore, probable that this acidic mass is much more recent than the basic contact differentia-

¹² An interpretation suggested by F. A. Linforth in oral communication.

¹³ Ref. report by F. A. Linforth, A. C. M. Geological Department, Butte, Mont.

¹⁴ A. Knopf: Ore Deposits of the Helena Mining Region, Montana, *Bulletin of the U. S. Geological Survey* (1913), 527, 137-139.

¹⁵ Many similar but smaller quartz masses are found between Boulder and Elkhorn. Others are known north and west of Butte.

¹⁶ See also A. Knopf: *Op. cit.*, 123.

tions, and represents an injection of siliceous-residue rather than magmatic segregation. The molybdenite and tourmaline, however, imply true igneous or pegmatitic conditions of origin.

3. *Disseminations*.—Disseminated ores in the granite consist almost entirely of auriferous or cupriferous pyrite. While largely the product of the primary crystallization, these deposits frequently show, by silicification and sericitization of the rock; that hydrothermal waters have contributed to their extension and enrichment. They therefore fall naturally, in chronological sequence, between the magmatic segregations and the later deposits of true hydrothermal origin. The more important occurrences of such ores are found in the Cardwell mining district, north-east of Whitehall; in the Heddleston mining district north of Helena, and on Red Rock Creek, near Basin.

In the Red Rock Creek occurrence the pyrite is entirely a primary mineral¹⁷—as thin sections show—in a fine-grained basic phase of the granite, near its contact with the overlying andesite. The pyrite is of small size, and not abundant, but is of widespread distribution. It carries enough gold to bring the grade of the general rock type locally up to about 50 or 75 c. per ton. The Heddleston district shows similar basic granite, with cupriferous pyrite to the amount of 5 or 10 per cent. of the rock by weight. The gold content is but a few cents per ton, but secondary enrichment makes small commercial orebodies of disseminated chalcocite.

The ore of the Golden Sunlight mine near Whitehall owes its value to a hydrothermal enrichment¹⁸ of a similar pyritic phase of a laccolith, intruded into and between quartzitic shales. Its upper surface is composed of fine-grained granite porphyry, almost devoid of quartz, and abundantly speckled with primary pyrite. This rock and the overlying sediments are cut by numerous tiny fissures, which have served to introduce the later solutions. The regions traversed are thoroughly sericitized and silicified; veinlets of quartz and pyrite abound; and the brecciated or otherwise porous portions of the slates are thoroughly impregnated with the quartz-pyrite mineralization. In this way the gold originally disseminated throughout the zone of the primary pyrite has become concentrated along the subsequent lines of enrichment and dispersal.

¹⁷ The rock in which this disseminated pyrite occurs is a basic quartz-monzonite. The minerals, in a holocrystalline aggregate, are plagioclase, orthoclase, hornblende, biotite, magnetite, and quartz. Magnetite, the earliest mineral, is unusually abundant, while quartz is rare. The rock is entirely unaltered, showing neither sericite, chlorite, or kaolin. The rare pyrite is probably an original, pyrogenetic mineral, occurring in irregular aggregates very similar to the magnetite masses.

¹⁸ In the Moffett Mine, in Bear Gulch, a similar hydrothermal enrichment is indicated by the presence of tiny seams of chalcopyrite in the basic rim of the granite (F. A. Linforth in oral communication).

The fundamental factor controlling the formation of the disseminated ores is therefore much like that observed with the segregations, the presence of a well-developed basic phase of the granite. In the case of the segregations a peninsular mass thrust out into the intruded rocks is most favorable to the differentiation of an ultra-basic contact; with the disseminated ores the common association is with the crust of the granite, preferably in outlying areas of the intrusive.

4. *Contact Deposits.*—The orebodies that owe their existence to contact action of the granitic magma may be divided into high-temperature and low-temperature contact replacement deposits. The former include the ores closely associated with the high-temperature contact minerals and are generally at or near the contact itself; the latter represent the orebodies formed at greater distances from the intrusive rock by the circulation of solutions of magmatic source.

In the Georgetown district (Fig. 6) west of Anaconda, both types are represented, and their inter-relation is well shown. Of the first group are the orebodies of the Cable mine¹⁹ contained within an inlying mass of metamorphosed limestone. The sedimentary rock has been highly altered, the contact phases ranging from nearly pure, coarsely crystalline calcite to aggregates of garnet, actinolite, magnetite, pyroxene, specularite, epidote, and mica. Large areas of magnetite alone are also frequent. Into this metamorphosed rock magmatic contributions of silica, iron, copper, bismuth and gold have been introduced in the form of quartz, pyrite, chalcopyrite, tetradymite, and metallic gold. The ore, therefore, consists of those minerals, together with calcite and small amounts of the contact silicates, irregularly distributed throughout the metamorphosed area. Their location in detail was determined by the presence of fissures, but the orebodies are in part at least the result of direct contact metamorphic action. Gold is in general the valuable constituent, but some copper ore has been shipped. Most of the many similar deposits on the borders of the granite, however, have been exploited for the secondary enrichment of chalcopyrite, gold being of subordinate importance. The Highlands, French Gulch, Tidal Wave and Ophir districts contain many good examples of this type of ore formation.

A short distance from the Cable mine are the deposits of the Southern Cross²⁰ district. They occur within a crystalline phase of magnesian limestone somewhat farther from the granite contact than the Cable ores. The high-temperature silicates are absent, and the limestone has received no direct accession of silica or iron from the igneous magma,

¹⁹ W. H. Emmons and F. C. Calkins: *Geology and Ore Deposits of the Philipsburg Quadrangle, Montana, U. S. Geological Survey* (1913), *Professional Paper* 78, 222.

²⁰ Emmons and Calkins: *Op. cit.*, 231.

Paul Billingsley: Southern Cross Mine, Georgetown, Mont., *Trans.* (1914), 44, 128.

whose effects are limited to a thermal recrystallization of portions of the sediments. Small dikes and sills are rather numerous, but apparently the metamorphism and ore deposition depend upon the main mass rather than these minor offshoots. The limestone is traversed by numerous fissures of slight displacement which have served to introduce the mineralizing solutions from the granite into the permeable crystalline limestone beds. As originally formed, the orebodies consisted of a pyritic replacement of this rock along the intersection of favorable beds with the fissures. The mineralization was pyrite, magnetite (?), siderite, calcite and quartz, with gold and copper present in small amounts. The orebodies have been peculiarly susceptible to the effects of oxidation, and in addition to the reconcentration of gold in the original shoots many mechanically formed deposits of auriferous cave breccia and corroded limestone have resulted.

In general, therefore, the Southern Cross district represents a more remote phase of contact action than the Cable. The contact silicates, with their addition of iron, alumina, and silica from the magma, are absent, and the limestone shows only the slight recrystallization caused by the heat of the igneous intrusion. The ore minerals, instead of being partly contemporaneous with the contact phenomena, are of subsequent infusion, and owe their source to residual granitic solutions rather than to the high-temperature emanations of the contact itself. In short, the Southern Cross type lies midway between the pyritic replacements of contact metamorphism and the siliceous fissure fillings of the cooling granite.²¹

In the Gold Coin mine, the connection with these latter veins is well-nigh completed. Ores of the Southern Cross type are here found in a fissure in unaltered limestone. There is no replacement of this rock, and the deposit approximates in form and mineralization the true quartz-pyrite gold veins of the granite stage.²²

5. *Fissure Veins*.—The last ore-forming activity of the granite period is represented by fissure veins, largely in the adjoining rocks but in many cases penetrating the granite itself for a few hundred feet from the contact. These veins are filled primarily with quartz and pyrite, but calcite, chalcopryrite, sphalerite, and galena, are occasionally found. The type is widespread, and carries the ores of many districts, among which Garnet (Fig. 7), Scratchgravel (Fig. 8), Marysville (Fig. 9), Unionville, Hassel, Pony, Bannack, and Whitehall are the more important.

The mines of the Garnet district surround an outlying "cupola" of granite, intrusive into Cambrian limestone and quartzite (Fig. 7). The

²¹ The Red Lion district mines (Fig. 5) are entirely of the Southern Cross type of ore deposit.

²² A list of districts in which ore deposits of each type are found is appended to this paper.

veins occur within a limited zone including the peripheral portion of the granite and the neighboring sedimentary rocks (Fig. 21). Five hundred feet within the granite, and possibly twice this figure beyond its borders, will cover this area of mineralization. The strike of the fissures, which are rather narrow (less than 2 ft.), is approximately parallel to the border of the granite, and the flat dips toward the igneous rock are nearly perpendicular to its contact. Despite the varying wall rocks—granite, limestone, quartzite, and andesite (sills in the quartzite)—the mineralization is uniform throughout the district, and minerals other than the prevailing quartz and auriferous pyrite are rare. Primary ore shoots, marked by an increased width and high percentage of pyrite, are numerous and persistent in depth within the limits of the productive zone. Between and beyond these the fissures are represented by only the most minute cracks. Enrichment of the orebodies by descending solutions is universal and is farthest advanced at the bottom of the oxidized zone, and in the partially attacked ore immediately below. At the surface itself the gold is absent, and it is probable that the enrichment has been accomplished mechanically by groundwater circulation. Galena and chalcopyrite have been recorded from the bottom of the veins developed deepest within the granite. In general, therefore, the Garnet veins consist of fissures formed in the solidified crust of a granite boss and in the adjoining intruded rocks, filled by the products of solutions derived from the more deep-seated portions of the igneous rock. The fissure zone is distinctly a halo of the periphery of the granite mass, and the uniform vein filling in diverse wall rocks implies contribution from a common source.

The Marysville district likewise surrounds a small outlier of the main batholith (Fig. 9).²³ The intruded rocks are calcareous and argillaceous slates of Algonkian age. Less extensive erosion than in the case of Garnet has left much of the roof in place, and the details of contact are more completely disclosed in mine workings. Here also the occurrence of mineralized veins coincides with the periphery of the granite and the adjoining metamorphosed sediments, the ores ranging from about 500 ft. within the granite to about 1,500 ft. beyond the contact (Fig. 21). The veins occur in two systems which in a general way are respectively radial and tangential to the igneous contact. Their dips are steep, as is the contact also. Later faults are sometimes slightly mineralized.

The vein filling in Marysville is essentially quartz, with subordinate amounts of calcite, fluorite, galena, pyrite, sphalerite, and chalcopyrite. The quartz, often either chalcedonic or amethystine, has in the typical deposits replaced an earlier stage of lamellar calcite. Open cavities and

²³ For detailed descriptions of many of the more important mines of the district, for longitudinal and cross-sections of veins showing their relation to the granite, and for history of the district, see paper by C. W. Goodale, *The Drumlunnon Mine, Trans.* (1914), 49, 258-283.

comb structure, with crystals of quartz of later generations, are abundant. Fragments of country rock found in many of the veins suggest that the fissures, prior to the mineralization, existed as cavities filled with detritus. On the lower levels of the Drumlummon mine, the vein consists entirely of such a detrital "breccia," cemented by lamellar quartz and calcite.²⁴ Post-mineral faulting along the vein courses is common. The sequence may be summarized as follows:

1. Opening of fissures and partial filling by detritus.
2. Impregnation of fissures by lamellar calcite.
3. Replacement of calcite by quartz and sulphides.
4. Early faulting, partly mineralized.
5. Later faulting, unmineralized.

It follows that the quartz mineralization of the present orebodies is considerably more recent than the original formation of the fissures, which, it can scarcely be disputed, closely succeeded the intrusion and cooling of the granite mass. The characteristics of the Marysville veins, moreover, are those of deposits formed by relatively cool waters at shallow depths. The open cavities, chalcedonic quartz and absence of hot temperature sulphide minerals except in small quantities, are significant in this connection. The 1,600 ft. of depth reached by the Drumlummon mine witnessed the disappearance of the typical primary mineralization of the upper levels, and quartz, calcite, chalcopryrite, and tetrahedrite—the sulphides in very small amounts—alone persisted (see Fig. 18).

In seeking a source for this post-granite, shallow depth mineralization, two possibilities stand out. One of the more recent igneous periods may have contributed these solutions to the fissures of an older period or the granite itself may have yielded them from internal sources. The distribution of the Tertiary igneous rocks renders it impossible to trace the many instances of the Marysville type of vein filling to such a source. This would require that the solutions travel for long distances, in a region where cavities and fissures abound, to group themselves in a series of veins surrounding a granite intrusion. On the other hand, the granite and its associated differentiations afford a source available to every vein in the district. The mineral association is similar to that found in recent and active hot springs of granitic origin. Even the replacement of calcite by quartz is not without parallel among the gold quartz veins of the granite batholiths of the Cordilleras.²⁵ It seems probable, on the whole, therefore, that the Marysville veins represent a late contribution from the granite, wholly analogous to hot springs now found around the batholith, and that while possibly they are as young or younger than the more

²⁴ A "breccia" of this type does not necessarily involve displacement of the fissure walls. It exists in Butte, Georgetown, and Radersburg under circumstances that place its origin in the filling by wash or attrition of open cavities.

²⁵ DeLamar Mine, Idaho. See Lindgren's *Mineral Deposits*, 437.

recent igneous stages, they must still be genetically classified with the granite period.²⁴

The Garnet district, with its deposits of a type characteristic of formation at fairly great depths, lies at an elevation of about 6,000 ft. The surface at the time of vein formation must have exceeded 7,000 ft., the level of the subsequent Eocene peneplain in that region. At Marysville the peneplain lies at 7,000 ft., while the veins outcrop at 5,500 to 6,000 ft. elevation. The post-Eocene erosion is thus essentially the same in both cases. The location of the veins at both Garnet and Marysville is similar in reference to the igneous rock. The great differences in vein type are thus again carried back to a difference in time, with the Marysville mineralization deposited at slight depths probably after much of the post-Eocene erosion had occurred.

6. *Summary.*—The ore deposits of the granite period are associated with the outlying "cupolas" and bosses rather than with the main mass, forming on the map a halo around the larger areas (see Figs. 1, 23). The true significance of this lies in the relation of the smaller intrusions to the batholith. These are most abundant on the flanks of the eroded mass, diminishing in number and size with distance from the contact. There is evidently a gradation downward from small dikes, the vanguards of the intrusion, through a phase of larger irregular stocks and bosses to the connected mass of the batholith. The vertical distance from the remotest advance intrusions to the main mass is in general less than 2,000 ft. Within this zone the phenomena of the granite mineralizations have occurred; segregations on the borders of the protruding tongues of magma; disseminations in basic border phases; contact-metamorphism accentuated by contributions of iron and silica from the magma; replacements of the intruded rocks by the iron-silica solutions; and fissure filling by the same products of magmatic segregation, all

²⁴ This theory has already been advocated for the views of the Marysville district by Barrell and Weed, and is well summarized by the latter. "There are many productive mines working veins cutting the igneous rock and the contact rocks above them. Such vein fissures are caused both by the contraction due to the crystallization and cooling of the igneous rock and by the shrinkage of the metamorphic zone above the igneous rock. As the magma and its surrounding shell of heated sediments cools down it must contract, and this contraction will result in a cracking both of the igneous rock and the contact zone; and if the rocks of the contact zone are homogeneous the cracks will assume a more or less radial position. If these cracks extend to a depth sufficient to reach this molten magma, they will be filled and dikes will be formed; if not, cracks become channels for pneumatolytic vapors and later circulating waters, and thus pegmatitic veins and true mineral veins may be formed and may merge into one another."

This hypothesis is disputed by Knopf in the case of Marysville on what seem to us insufficient grounds, and upon due consideration and investigation of the many districts similarly situated we feel that Weed's and Barrell's stand is strengthened rather than weakened by the additional evidence gathered.

are in the main limited to the smaller masses. It is evident that in these, magmatic instability, witnessed both by basic segregations and acid residues, was accentuated to a degree never reached by the more slowly cooling mass underlying.

B. Aplite Phase

1. *General Description.*—Aplite appears within the batholith in three phases: (1) The earliest are segregations on the contact, and acidic dikes in advance of the main intrusion. (2) Following these in time come the main aplite injections—bodies of small dimension and irregular outline, most abundant in the peripheral portions of the granite, and forming in general about 10 per cent. of the total mass. The contact indicates that the granite was still hot when these intrusions occurred, and the absence of connecting links suggests that the aplitic material was a residue of widespread local occurrence in the crystallizing batholith. (3) Later dikes of more basic rock, quartz-porphyrries or pyroxene aplites, are believed in some instances to be offshoots of large aplitic masses below. It is believed that these larger deep-seated bodies are of identical origin with the smaller injections of the outer crust, but that their size and position resulted in slower cooling and partial differentiation, of which the porphyry dikes are the product. These show by their contacts that the granite was cool at the time of their intrusion. The importance of the associated mineralization has led to the creation of an independent group in the genetic classification for such quartz-porphyry.

2. *Segregations.*—A siliceous residue itself, the aplite is particularly susceptible to a differentiation and segregation of its acid elements. In the outlying dikes it is not rare for the rock to grade from the aplite into alaskite and even unalloyed quartz at the extremities.²⁷ Throughout the aplite areas segregations of pegmatite, often highly siliceous, abound, and igneous quartz, with traces of molybdenite and tourmaline, is not unknown. The smaller dikes within the granite pass into quartz or tourmaline or both with no crystalline break. Sulphide minerals are occasionally intergrown with the tourmaline.

These conditions can result only from a close association of high-temperature solutions and vapors, rich in silica and boron, with the aplitic magma. They must, in fact, be regarded as contemporaneous emanations from it. In part, these solutions are caught and crystallized within the cooling rock itself; in part, they escape and, being more mobile than the aplite, penetrate the surrounding rock in the form of pegmatite and quartz-tourmaline dikes.

In the Valley Forge mine, at Rimini, and in general throughout that district as well as in the Oro Fino district near Deer Lodge, some sulphide

²⁷ Lost Creek, Deer Lodge, Mont.

minerals are found intergrown with the quartz-tourmaline segregations from aplite dikes.²⁸ Galena, sphalerite, pyrite, and chalcopyrite are known in this association. Such ores are of little economic importance, however, the vast bulk of the sulphides occurring with a later quartz mineralization, which also carries some tourmaline.²⁹ The earlier igneous ores with their parent aplite may be seen, in the Valley Forge mine, as a

²⁸ Gold is the only mineral of economic importance found and is associated with the pyrite, which is the most abundant sulphide. Abortive attempts have been made in the Oro Fino district to stope gold ore from these quartz-tourmaline-pyrite dike veins. Possibly the unusually high gold content of the lead-zinc-silver ores of the Rimini district is also connected in some way with the tourmaline dike type of mineralization.

²⁹ This differentiation between the tourmaline-quartz-pegmatite dikes and the sulphide veins which also show some tourmaline, is opposed to the conclusions reached by A. Knopf, in *Bulletin* 527, of the U. S. Geological Survey. It is adhered to by the authors of this paper, however, for the following reasons:

I. Throughout the Rimini district there are many evidences of the priority in age of the tourmaline to the lodes. For example: (A) In the Valley Forge mine the tourmaline occurs in a zone of many parallel bands with strikes nearly east-west, while the vein traverses the belt with a course of N 70° E, cutting the successive tourmaline dikes at an acute angle. In detail the vein can be seen at many points truncating tourmaline bands. The vein proper is formed by mineralization along a fault fissure, which is definitely subsequent to the tourmaline period. (B) In the Merritt mine a drag block of tourmaline, with some quartz and sulphides, lies between two fault slips, which carry the later sulphide mineralization along them. (C) In almost all the mines a breccia of aplite, quartz and tourmaline, cemented by an ore of quartz, tourmaline, and sulphides is of common occurrence.

II. Two types of tourmaline are present. The earlier (as is evidenced by structural contacts) is coarse-grained and of uniform texture across maximum widths of 10 ft. Its contacts with the granite are sharp and definite. The later is present as a component of the "black quartz" that accompanies the veins. It is of microscopic size and forms an irregular proportion of the aggregate.

III. There is much evidence of the contemporaneous formation of the tourmaline proper and the aplite (A) 400 ft. south of the Lee Mountain vein, are unaltered aplite dikes in fresh granite. They show numerous crows-foot segregations of tourmaline. Near the Comet mine also such segregations in unmineralized aplite are common. (B) Dikes of aplite occasionally grade with crystalline continuity into pegmatite, quartz-tourmaline, and coarse-grained tourmaline.

IV. There is no uniformity of distribution between the tourmaline dikes and the veins. (A) On the Sixty-ninth Hill tourmaline is found at points remote from the Lee Mountain vein on either wall, and with varied strikes. In many of these cases no associated mineralization whatever can be observed. (B) In the Stanton mine very rich lead-zinc-silver ore is found, sometimes up to 5 ft. wide with no tourmaline either associated with the ore or present in the vicinity. (C) In the John McGrew prospect the "black-quartz" type of tourmaline occurs along a fault vein, with none of the coarse-grained tourmaline near. In brief, the quartz-sulphide ores are found throughout the district. The "black quartz" sometimes accompanies these ores, but not invariably. The coarse tourmaline occurs almost exclusively in the northern part of the district, and is here seen to be earlier than the vein mineralization where the two happen to intersect.

fragmental breccia cemented by the later hydrothermal minerals. In general, therefore, the metalliferous tourmaline segregations are merely significant antecedents of the great tourmalinic ore-bearing mineralization that followed.

The single instance of economic exploitation of a quartz segregation is the Quartz Mass, previously described. This, said to run 99 per cent. silica, was formerly used as converter lining in the Butte copper smelters.

3. *Disseminations*.—Pyrite and chalcopyrite are common as primary minerals of the aplite, and molybdenite is a frequent constituent of the pegmatitic phases. In general, however, these sulphides are in very sparse quantity.

The Modoc mine, in the Red Lion district, affords the single instance where a disseminated sulphide has raised the aplite to a value sufficient to encourage prospecting (Fig. 5). This mine has developed an aplite boss of elliptical plan, with axes of 100 and 50 ft. respectively, which is intruded along the contact of a granitic batholith. The boss contains much primary pyrite and chalcopyrite in a rock of typical aplite constitution. On the surface, and at shallow depths, oxidation and enrichment are apparent, with the chalcopyrite showing alteration halos of bornite and chalcocite. On the lowest level (200), however, the rock is unaltered. Within the zone of secondary enrichment the boss carries about 1.5 per cent. copper but the primary grade is less than 1 per cent.

4. *Fissure Veins*. (a) *Origin*.—The distribution of the aplite intrusions throughout the batholith is also that of the great quartz mineralization that followed. In the majority of instances the vein-forming solutions have penetrated the same fissures that the aplite had earlier found, or have occupied fractures in the aplite, reopening along the same general zones. The aplite, in its ultra-siliceous phases, forecasts, with its quartz, tourmaline, and sulphide segregations, the composition of the later ores. In more than one case the boundaries of an aplite dike on the surface become the boundaries of a later quartz vein below, the latter feathering out into quartz stringers in the dike at intermediate points. Identity of position, similarity of composition, and deep-seated origin, therefore, imply a common source for the aplite magmas, the pegmatitic vapors and the vein-forming siliceous waters. Their chronological sequence suggests that the several stages represent increasingly acid residues from the local reservoirs of aplite magma.

(b) *Distribution*.—The occurrence of these veins of aplite source is widespread. They contain most of the ores of the Elliston, Rimini (Fig. 13), Lump Gulch, Wickes, Comet, Basin, Jack Mountain (Fig. 12), Little Boulder and Homestake districts, and in part those of the Butte district. They are found several hundred feet above the granite contact, and their barren roots are exposed in the deepest portions of the batholith laid open by erosion. Nine-tenths of the silver, lead and zinc ore of the

region exclusive of the Butte district has been extracted from veins of this series.

(c) Character.—The following characteristics are universal:

1. Position near the contact of the main batholith, with fissures mainly within the granite.

2. Strike approximately east and west.

3. Dip steep.

4. Many parallel and reticulated fissures rather than a single large opening.

5. Extensive replacement of the wall rock by the mineralizing solutions.

6. Sericitic alteration of the wall rock.

7. General presence of the sulphides of all the base metals—lead, zinc, copper, and iron, in all the veins.

8. Predominance of each metal in turn in an orderly succession proceeding from the source of the solutions.

9. Increase in proportion of quartz relative to total sulphides with increased depth.

A fuller description of these common traits may be of profit.

1. The great majority of the ore deposits contained in these veins of aplitic origin are situated within a short distance, vertically, from the contact of the main granite mass. Some, such as the Alta and Gregory, extend upward into the intruded rock, but the majority, among which can be numbered the Bertha, Comet, Eva May, Crystal Bullion, Hope, Valley Forge, Lee Mountain, Ontario, and Monarch, are within the granite at distances less than 1,000 ft. from its upper surface. The few veins more deeply placed contain little but quartz and suggest the eroded roots of veins of normal type above. One hundred and nine veins outside of the Butte district have been identified as belonging to this group, and wherever possible the position of their outcrop with reference to the granite contact has been determined. The restoration of the contact has been accomplished in many cases by connecting residual fragments of cover; in other instances, as at Marysville and Philipsburg, the physiographic evidence of drainage has been used; in still others the limit of metamorphism in the sedimentary rock is significant; but in a surprisingly large number of places the contact itself is preserved and gives a definite plane for measurement. In the Wickes, Comet, Basin, Jack Mountain and Rimini districts this is largely the case. The results of the classification of these orebodies according to position are striking. Of the 109 listed, nine veins could not be definitely placed with reference to the top of the granite. Of the remaining 100, 15 are contained for the most part in the overlying cover, 54 lie within a distance of 500 ft. below the contact, 28 outcrop between 500 and 1,000 ft., and including the deep developments on the above, only four veins are known at depths below

1,000 ft. With the exception of the Granite-Bimetallic vein, replacement ores in limestone, and possibly some of the Butte silver veins, the productive portions of these fissures are limited to the region between 1,000 ft. above and 1,000 ft. below the contact (Figs. 22 and 24).

2 and 3. The close approach of the strikes of these veins to an east-west course, and the prevailing steep dips, cannot be without significance.

A reconstruction of the granite surface indicates that above the vast majority of these veins the contact lies at a flat angle, to which the steep dips are approximately perpendicular (Figs. 2 and 17). The main granite mass, within which the greater part of the veins lie, is shown by such a reconstruction of its original form to be elongated on its summit, in a north-south direction, while comparatively narrow between its eastern and western contacts. These are appreciably steeper than the northern and southern boundaries.³⁰ The fissures, therefore, are normal in strike to the long axis of the intrusion, and lie at high angles to the two roughly parallel bounding contacts on the east and west. Their distribution, coextensive with the granite, suggests a common origin connected with the dynamics of that rock. These several requirements are fulfilled by regarding the fissures as contraction cracks formed in the outer mile of the cooling granite, following in their position the habit of such fractures.

4. The detailed structure of the veins further indicates that no great displacement accompanied their formation.

The structure of several typical veins of this series is shown in Figs. 14, 15 and 16. By the use of solid lines for vein material, quartz, sulphides, etc., and dotted lines for post-mineral faulting, it is attempted to represent the character of the fissure, the distribution of its mineral contents and its influence on subsequent movement of the ground.

The Crystal vein is at once the simplest and most representative of the type (Fig. 14). Its total developed length is about 6 miles; its average width about 30 ft., and it has been explored from within the andesite roof blocks to a depth of about 1,500 ft. within the granite. The section shown in the Crystal mine is typical of the entire extent. Here the total width of mineralization is about 50 ft., with a bordering zone on either side of silicified and sericitized aplite. Within this 50 ft., bands of true fissure filling—quartz and sulphides—alternate with belts of mineral-

³⁰ While in a general way the top contact of the granite is nearly flat, sloping gradually under the cover rocks in quaquaversal manner, nevertheless the dome has a distinct long axis, with a general north-south direction. Along this axis the top is nearly level, with local irregularities, and when the granite finally goes under cover to the north and south its shallow depth is suggested by the numerous and comparatively large outliers that mark its course (see Fig. 1). On east and west, however, such outliers are both fewer and smaller, indicating a deeper submergence of the granite on these flanks, and hence a steeper dip to these lateral contacts. A contour map of the restored granite surface shows graphically the longer north-south axis, and the steeper dips on the northwest and southeast contacts of the main intrusive mass.

ized country rock traversed by numerous tiny stringers. The vein bands average 2 to 5 ft. wide; the intervals may run up to 10 ft. Along the course of the vein great variation is found. The several quartz-sulphide fissures terminate or finger out, and new bodies appear in the mineralized aplite between. The richer ore ceases in one belt, to begin in another hitherto barren. In short, while the zone itself is continuous, the separate fissures and oreshoots that compose it are of limited extent, existing only as overlapping lenses and reticulated cracks within a belt of silicified and sericitized country rock. The belt has in the Crystal, as almost invariably throughout the region, been followed by post-mineral faulting, which closely parallels the banding of the vein system.

In the Comet mine the general zone of mineralization is unusually wide—nearly 150 ft.—and may be subdivided into three minor belts (Fig. 15). Each of these has the same characteristic features that are found in the Crystal. On the lower levels they diverge, forming three parallel mineral zones rather than a single wide one. The rock alteration in the Comet is more pronounced and extensive than in the Crystal, a fact in accord with the obviously greater fissuring and more intense mineralization at the former mine. The Comet vein is known for a distance of about 5 miles, with a width of 100 ft. and a vertical development from the andesite contact to a depth of about 1,000 ft. within the granite. The Baltimore vein illustrates a feature also found in Butte—the spraying, as it were, of the solutions from the main fissure into a series of smaller cracks at large angles with its course (Fig. 16). This structure, called “Horsetail” in the copper mines, implies the presence of permeable fissures which were traversed by the major vein openings, and filled by the mineral solutions. The close association of these zones with the larger east-west veins, and their detailed structure in the Butte district, indicates that their origin is in some way traceable to the dynamics of the major fissuring, of which they may be mechanical resultants. The “Horsetail” structure is at any rate more than a local feature, and must be due to some widespread and fundamental cause.

In general, then, the veins of the aplite stage fill the openings along wide zones, often several miles in length, and composed of many small overlapping fissures (see Fig. 12). The present continuity and unity of the veins is due to the diffusion of the mineralization along these indefinite zones, the cementation with quartz and sulphides of disconnected fissures, and the silicification of the intervening permeable country rock. Post-mineral faulting, following the lines of weakness these zones provide, gives a false appearance of definite and extensive walls to the orebodies.

The fissure zones which have thus determined the position and character of these ore deposits of the aplite state apparently represent a separation of the walls along their course. This is suggested by the

short lenticular openings as well as by the absence of evidence of movement, such as shearing in the intervening country rock, or displacement of intersected dikes of aplite. Such a drawing apart must imply a state of tension in the granite crust, a state which of course existed during its cooling and contraction. So from this angle also the origin of these fissures seems to fall in with the conclusions already arrived at—that they mark the lines where the shrinkage of the batholithic crust found expression in tension cracks.³¹

5 and 6. It has already been observed that there is intense alteration of the country rock in the general vein zone. The new minerals formed are sericite, pyrite, and secondary quartz; the old minerals attacked are the feldspars and ferromagnesian. Occasionally needles of tourmaline are added. Knopf finds in the Rimini district an actual gain in silica, iron, and sulphur, and a loss in all the bases except potash.³² The actual gain in mass by his analyses is 8.87 per cent. of the original. The altered rocks, of which this may be considered representative, are thus marked by the absence of any dark minerals (unless tourmaline is present), by a high proportion of quartz, and by numerous specks of pyrite. The limits of alteration are indefinite, the change penetrating along joints and fissures far beyond the main belt. In general, the size and intensity of mineralization of the ore deposit is directly proportional to the amount and extent of sericitic rock alteration. Many of the smaller veins and shoots show only a few feet of pyritized granite along their walls. Chloritic alteration is common along the post-mineral faults.

7. The primary vein filling of the aplite stage includes the following minerals: quartz, rhodochrosite, rhodonite, tourmaline, calcite, siderite, and fluorite, in the gangue, and pyrite, arsenopyrite, galena, sphalerite, chalcopyrite, tetrahedrite, bornite, stibnite and argentite, among the sulphides. Of known occurrence, but rarely encountered, are hubernite, molybdenite (oftenest with pegmatite), proustite, pyrargyrite, gold, magnetite, hematite, and cassiterite. The great majority of the ores consist of the sulphide minerals listed above in varying proportions, in a gangue of quartz.

Using the same examples previously considered with regard to position of outcrop, it appears that the mineral distribution is as follows (detailed information is available in 93 cases):

³¹ Prof. Clapp, of the Montana State School of Mines, argues (in oral communication) that the state of tension was due rather to great dynamic stresses, probably those which found expression later in the Basin Range normal faulting. The suggestion is doubtless of value, and may be an additional explanation of the fissure occurrence. It has seemed to the authors, however, that the interpretation given in the text is at once simpler and sufficient.

³² A. Knopf: Ore deposits of the Helena Mining Region, Montana, *Bulletin of the U. S. Geological Survey* (1913), 527, 50.

93 Fissure Veins of Aplite Stage

Pyrite	is present in 86 veins, or 92 per cent.
Galena	is present in 79 veins, or 85 per cent.
Sphalerite	is present in 63 veins, or 68 per cent.
Chalcopyrite	is present in 30 veins, or 32 per cent.
Arsenopyrite	is present in 27 veins, or 30 per cent.
Tetrahedrite	is present in 13 veins, or 14 per cent.
Bornite	is present in 7 veins, or 8 per cent.
Stibnite	is present in 3 veins, or 3 per cent.
Tourmaline	is present in 16 veins, or 17 per cent.
Rhodochrosite	is present in 16 veins, or 17 per cent.

It is thus evident that the typical vein aggregate is pyrite, galena, and sphalerite in a gangue of quartz; that pyrite and galena exceed any other combination; and that the copper, arsenic, and antimony compounds follow considerably behind the iron, lead and zinc. The unusual gangue minerals, tourmaline and rhodochrosite, are each found in about one-sixth of the cases—rarely occurring together. Tourmaline is found in the regions where it is also abundant as a constituent of the aplite.

The occurrence of the sulphides of iron, lead, and zinc in nearly 70 per cent. of the examples used is striking. Moreover, since sphalerite, as will be seen later, is most abundant at moderate depths, it is possible that deeper development would add it to most of the additional 15 per cent. of cases in which pyrite and galena exist together. It is thus safe to say in 75 or 80 per cent. of the best-known quartz veins of aplite origin these three base metals occur in sufficiently high proportions in the solutions to be conspicuous in the resulting mineral aggregate.

The presence of copper, arsenic, and antimony is not nearly so widely evident, and in possibly half the cases these elements cannot be detected. Where they are found, they exhibit wide variation in quantity, ranging from the merest trace to a high percentage of the total vein matter. The similarity of the vein-forming solutions over wide areas is best seen in the pyrite, galena, and sphalerite; the geographic variation finds its best expression in the arsenopyrite, tetrahedrite, and stibnite. The manganese minerals and tourmaline are also geographic variables.

The general significance, therefore, of the mineral distribution may be summarized in a few sentences. The siliceous residue of the aplite stage that formed the quartz veins had essentially similar composition over wide areas, evidence of which lies in the almost universal presence of appreciable pyrite, galena, and sphalerite in the veins. Beyond this fundamental similarity, however, it varied from place to place in the amount of copper, arsenic, antimony, and manganese, these elements generally in small amounts, locally increasing sufficiently to form in their compounds a prominent proportion of the ore.

8. In the various mining districts, deep development has been prose-

cuted on ores of the most diverse types outlined above. Lead-rich veins, with accessory zinc; zinc-rich veins with the other base metals subordinate; even arsenic-rich veins marked by much arsenopyrite, have all been followed to considerable depths below their outcrop, and incidentally below the top of the granite. Aside from the changes in the proportion of primary sulphides, which will be discussed later, the universal trend in the composition of these veins has been an increase with depth in the proportion of quartz to ore minerals (see Figs. 19, 20, 22, 24). In mine A, for instance, the percentage of SiO_2 in the mill feed increased from 55 per cent. on the 400 level (about 700 ft. below the andesite) to about 75 per cent. on the 600 level. In mine B the ore shoot changes, on the 600 level from a galena sphalerite aggregate with little quartz to nearly clean quartz, and much development below has demonstrated the persistence of this siliceous phase. In mine C, 1,400 ft. of development on the 1,200 level, about 2,000 ft. within the granite, encountered only quartz sparsely specked with pyrite, although the work is directly below a rich lead-zinc oreshoot on the upper levels. It is a striking corroboration of this tendency that the vein roots, exposed where the granite is deeply eroded, are without exception very siliceous, showing little but quartz.

9. While the total amount of sulphides in the veins thus diminishes with depth, the proportions of the different sulphide minerals themselves vary also within the vertical range of the ore shoot³³ (see Fig. 24). In certain cases the change can be followed throughout in a single mine, as in the Valley Forge, Comet, Alta, and Eva May, where exploration has traced the oreshoot through a considerable vertical range. The results may be summarized without unnecessary details of description. In each of these mines, with ores of different composition, the various minerals behave similarly. The lead content of the vein is found above, generally within 1,000 ft. of the granite upper contact. Whatever zinc is present is in the main massed just below the lead zone, which places it between 1,000 and 1,200 ft. from the roof. Pyrite is present throughout, but persists to greater depths than does galena or sphalerite, and is apt, below 1,200 ft., to be associated with the primary copper minerals—chalcopyrite, bornite, and tetrahedrite.

The evidence obtained from neighboring veins, outcropping at different depths within the granite, corroborates the conclusions reached from

³³ In this discussion of vertical zoning secondary effects are excluded. Of these, enrichment of the gold and silver, particularly the latter, is the most marked. It is probable that well-nigh all the argentite found in these veins is a secondary mineral. The greater resistance of galena to oxidation results in residual bunches in the oxidized zone, but has no influence on the occurrence of that mineral below water level. There is no evidence whatever of migration and reprecipitation of either galena, or the more soluble minerals sphalerite and chalcopyrite. Secondary enrichment is represented almost exclusively for the base metals, by chalcocite, with wurtzite (of doubtful occurrence) as an occasional possibility.

these completely developed ore shoots. Consider the Eureka-Stanton group at Rimini, with outcrops ranging from 500 to 1,000 ft. below the andesite capping (Fig. 13). Highest is the Eureka vein, with an ore shoot 400 ft. deep of pyrite, arsenopyrite, and galena. Similar ore in the Bunker Hill ranges between 800 and 1,000 ft. within the granite. The upper workings of the Stanton, a short distance below this, show galena and sphalerite in equal amounts, with some pyrite; while in the lower tunnel little galena is to be found, with sphalerite in overwhelming excess. The zoning thus shows arsenopyrite, and galena above, and sphalerite below, all within a probable maximum distance of 1,200 ft. from the batholith roof.

The numerous small mines on Big Limber Creek, near Basin, afford an equally good example of the tendency (Fig. 12). This creek heads within the andesite, and cuts down into the granite to a depth of possibly 750 ft. A depression in the old granite surface brings down the andesite again near its mouth at Basin. From north to south, therefore, the vein outcrops in this gulch progress from the andesite itself down to a maximum of 750 ft. within the granite, rising again to proximity with the contact at Basin. The northernmost vein is the Custer-Hiawatha, outcropping within the andesite. It contains tourmalinic quartz, with some arsenopyrite, and carries gold. Some galena appears on the 200 level. To the south, possibly 300 ft. below the andesite, are the Buckeye, Boston, Minneapolis, and Virginia veins. These show manganese-galena ore to a depth of about 200 ft. with sphalerite in considerable amount below this level. Still farther south, and deepest within the granite, are the Deer Lodge and Copper King veins, with chalcopryite, bornite, and tetrahedrite. The Copper King has no galena or sphalerite but these are present in the relatively higher Deer Lodge. As the andesite cover to the south is approached, the Hope mine shows a reversion to the typical manganese-lead type of ore with much sphalerite below the 100, and some chalcopryite on the bottom level (600).

The zoning in this district thus shows an upper belt of quartz, tourmaline, and arsenopyrite, a lead horizon between 200 and 500 ft. below the andesite, a zinc zone immediately below this, and a lean lowermost zone marked by the appearance of the copper minerals. The small vertical ranges of the different belts are probably due to the small size and slight intensity of mineralization of the veins.

Examples could be multiplied, and will appear as the different districts are described. It must suffice here to state that in all the veins known the facts corroborate the conclusions formed above: that arsenopyrite and galena are found in the higher portions of the ore shoots; that most of the zinc present occurs below the lead zone; and that this in turn gives way in short distance to lean pyrite, accompanied by small amounts of the copper sulphides. The measure of these changes varies in different

veins—most markedly between large and small veins. It can be safely said, however, that little lead is found more than 1,000 ft. within the granite; that little zinc is found below 1,200 ft., being mostly just below the lead zone, and that copper, while often present throughout a shoot, will in primary form predominate only in the depths.³⁴

(d) Geographic Variation.—As they are found throughout the area of the batholith, these veins fall naturally into several groups, each group in general coincident with the extent of a so-called mining district. Between such districts veins are few and insignificant. This geographic grouping suggests a localized source for the filling of the veins of each group, a probability increased by their own similarity of composition which contrasts strongly with their general unlikeness to the mineral aggregates of other districts.

Thus the veins of the Rimini district (applying the term to the area around Red Mountain and the headwaters of Tenmile Creek) are characterized by an unusual proportion of arsenopyrite in their upper zone. This mineral, which carries some gold, has supplied the ores of numerous gold mines in the higher portions of the granite. Lower down, the veins of the region show galena and sphalerite, with comparatively little pyrite. Tourmaline is abundant in association with the earlier stages of mineralization, which include also the arsenopyrite, but is absent from some of the later lead-zinc ores.

In the Jack Mountain and Comet districts to the south, zinc is the element found in excess. Arsenopyrite is rare, and pyrite is more abundant than at Rimini. Thus, even in the uppermost shoots, such as the Alta and Gregory, much sphalerite is found with the galena, while with the scant arsenopyrite the gold value is much lessened. Below the profitable sulphides, too, the ore shoots contain much pyrite (Comet, Alta, and Eva May), in contrast to the straight quartz of the northern district.

Still farther south, in the Basin district, rhodocrosite, the manganese carbonate, is widely present, the ores otherwise resembling those of the Jack Mountain and Comet areas. This tendency is found in much greater degree in the western portions of the batholith, and in the Philipsburg, Elkhorn (on Wise River), and Butte districts the manganese minerals form a large proportion of the gangue. In the first two of these camps they are associated with the complex silver sulphides of primary source, which are elsewhere rare in the vein fillings. Arsenopyrite is also abundant in these western districts. Copper, as an element in the aplite

³⁴ An ore high in primary chalcopyrite will often carry enough secondary chalcocite near the surface to be mined as a copper ore, despite the presence of much galena and sphalerite below. The primary ore of such a mine is subject to the same tendencies as have been found to prevail throughout the batholith. The copper above is purely a secondary modification. The Crystal, Bertha, and Receiver mines may be listed here.

mineralization, is never abundant and is less subject to extreme fluctuation than the above metals. There is, however, a zone in the batholith, including the Wickes, Jack Mountain, Little Boulder, and possibly the Butte districts, in which chalcopyrite is found throughout the veins in somewhat greater proportion than is elsewhere usual. It is in this belt that the shallow orebodies of secondary chalcocite occur, often associated with the galena and sphalerite of the upper zones. The Crystal, Bertha, and Receiver are good examples of such seemingly inverted veins. The great primary enrichment in Butte, associated with the later quartz-porphry, must not be confused with this type of ore deposit, which in its primary form is a lean chalcopyrite-pyrite mixture far different from the enargite, bornite, chalcocite, and covellite ores of the big copper camp.

To summarize, the geographic variation is represented by arsenopyrite and galena ores in the northern districts; by galena sphalerite, and pyrite ores further south; by manganese and silver ores along the western flank of the batholith; and by chalcopyrite-rich aggregates in a centrally located belt running from Wickes to Butte. These differences are primarily of proportion only, since in mineral composition, as we have already seen, all the veins are strikingly similar.

(e) Recapitulation.—In sum, therefore, the fissure veins of the aplite stage may be regarded as conforming to a widely prevailing set of conditions. They indicate, by similarity of composition, and origin closely associated with the pegmatitic after-phases of the aplite intrusions, and in occurrence, that they are so identified with the aplite as to render this source reasonable. Their position is near the outer contact of the batholith (see Figs. 22, 24), and the course and reticulated character of the fissures suggests that they represent tensional contraction cracks in the cooling mass. From these irregular openings the mineralization has diffused throughout a wider zone, and a halo of sericitic alteration marks the outer limits of the penetration. The ore minerals are in general the same in kind—galena, sphalerite, and pyrite—throughout the batholith, although in somewhat varying proportions. A vertical primary zoning of galena above, sphalerite for a short distance below, and pyrite more or less persistent throughout, is common, together with a general decrease in the proportion of sulphides with depth. The rarer minerals—arsenopyrite, chalcopyrite, rhodochrosite, tourmaline—are locally abundant, and it is in these that the geographic variation of the mineralizing solutions finds its chief expression. Arsenopyrite in the northern districts, rhodochrosite in the western, chalcopyrite in a central belt, suggest a slight regional variation in the granitic magma, of which the vein fillings are the ultimate concentrate.

C. Quartz Porphyry Phase

1. *General Description.*—The quartz-porphyry, which gives its name to this period of ore deposition, is best known in Butte,³⁵ although similar dikes have been found elsewhere in the batholith. As has been stated already, it shows, in the deep Butte workings, a trend toward aplitic character, together with a widening of its borders and loss of clean-cut contacts.³⁶ In short, it approaches, in texture and contact type, the more slowly chilled form of the true aplite intrusions. In composition the two rocks are closely allied, the quartz-porphyry being slightly more basic.

	SiO ₂	Al ₂ O ₃	Fe ₂ O ₃	FeO	MgO	CaO	Na ₂ O	K ₂ O
Quartz-porphyry.....	69.9	15.1	0.4	0.8	0.6	1.5	2.7	6.4
Aplite.....	76.2	12.9	0.7	0.1	0.2	0.9	2.6	5.6

The low iron, magnesia, and lime, and the high potash, differentiate either from the normal granite or rhyolite type.

Thus the distinction, decidedly apparent on the upper levels, is primarily one of texture and acidity, both of which differences are diminishing with depth. It would seem, therefore, that the quartz-porphyry dikes were relatively basic, quickly chilled offshoots from deep-seated masses of aplitic magma which, because of size or depth, or both, were enabled to fractionalize somewhat before solidification.

In the upper levels, at least, the quartz-porphyry dikes are materially later than the aplite, as is witnessed by the cooler contact phenomena and the subsequence of the porphyry to joints common to both granite and aplite.

2. *Disseminations.*—In certain places the quartz-porphyry contains disseminated pyrite of hydrothermal origin, slightly cupriferous, and under favorable conditions of enrichment the development of secondary chalcocite is sufficient to result in a commercial ore. These deposits are in genesis strictly parallel to such better-known examples as the Ely porphyries, but in Butte are on a comparatively insignificant scale.

3. *Fissure Veins.*—The fissure-vein mineralization of the quartz-porphyry phase is, so far as known, restricted to the Butte district, where the great copper deposits are genetically referred to these intrusions. These deposits have been thoroughly described by R. H. Sales³⁷ and it

³⁵ See Weed. *U. S. Geological Survey, Professional Paper 74.*

³⁶ This is strikingly shown, to name only one instance, by the Modoc dike, as now exposed from the upper levels of the Modoc mine to the lower levels of the Leonard. Above, it shows large orthoclase phenocrysts often an inch in length, at midway points it has the normal quartz-porphyry texture, with small rounded quartz-phenocrysts; below it is holocrystalline, and with difficulty distinguished from normal aplite.

³⁷ *Trans.* (1913), 46, 3.

is necessary only to outline here the features of significance to the general correlation of ore deposits of the batholith.³²

1. General Description. The general situation is as follows:³³ A series of east-west fissure veins about 1,000 ft. apart and 4 miles long, extends across the district, cutting alike granite, aplite, and quartz-porphyry, but cut by the Big Butte rhyolite. These veins are the earliest of the district. Cutting and displacing these are a series of northwest mineralized faults, and a later succession of northeast faults. The latter series is divided into three groups—south-dipping mineralized, north-dipping barren, and south-dipping barren respectively. The faults, particularly the northwest series, are most numerous near the eastern termination of the early east-west veins in the area now known as Anaconda Hill.

2. East-West Fissure Veins.—These strong fissures are subsequent to the aplite in age, and probably follow the quartz-porphyry also, although only their extreme eastern portions are found in contact with the latter. Their mineral filling is complex, and varies greatly along their extensive course. It can in general, however, be everywhere separated into two ages. The earlier type is essentially quartz, pyrite, and sphalerite, with small galena ore shoots near the surface, and little silver except where secondary enrichment has contributed. This stage exhibits roughly the typical primary zoning of the aplite-period fissures, galena uppermost, sphalerite below, and lean pyrite-chalcopryrite in the depths. The zones do not, however, parallel the present surface and the upper forms of mineralization occur mainly in the western and northern portions of the district with the lower pyrite zone near the surface to the south and east. This mineralization is accompanied by little rock alteration.

The later vein filling occurs often as a cement for a breccia of the earlier ores, and in other instances appears to replace them. In the western part of the fissures this later mineralization is most frequently quartz, rhodocrosite, calcite, and sphalerite, with occasional high-grade silver ore shoots. In the eastern portions it consists of the copper-rich sulphides, enargite, bornite and chalcocite, with very little sphalerite. The western phase occurs with the lead-zinc ores of the upper zones of the earlier mineralization, the eastern with the quartz pyrite of the lower horizons. To the east, also, the copper mineralization is very intense,

³² The conclusions drawn in the following description do not exactly coincide with Sales' views, especially the conception of superimposed mineral zoning of two distinct ages with the copper mineralization in the later period.

³³ See maps of vein and fault structure by R. H. Sales: *Ore Deposits at Butte, Mont.*, *Trans.* (1913), 46, Plates 1 and 2; also maps by W. H. Weed, in *Geology and Ore Deposits of the Butte District, Montana*, *U. S. Geological Survey* (1912), *Professional Paper* 74.

and the early minerals are often obscure, but the undoubted priority of the great quartz-pyrite masses to the copper enrichment is witnessed both by an intervening period of brecciation and by the details of contact between the two.

There is almost no residual zinc within the "dome" of copper mineralization that thus penetrates upward into the eastern end of these fissures.⁴⁰ This fact is doubtless largely due to the low zone of early mineralization represented—a zone which was probably below the precipitating horizon of sphalerite—but on the upper levels and in outlying veins it seems likely that some sphalerite has been driven out by the copper solutions. A "halo" of zinc ore richer than the average with two generations of sphalerite, frequently fringes the copper shoots in the east-west veins, and this may be partly due to such a migration and reprecipitation of the early sphalerite. This hypothesis, suggested by M. H. Gidel of Butte as long ago as 1912, seems increasingly probable with additional developments.

3. Northwest Fault Veins.—This system crosses the district from northwest to southeast, with its greatest concentration toward the east end of the earlier fissures. The faults have a nearly horizontal movement, and as suggested by F. A. Linforth, form a great shear zone, with connecting branches and intervening rhombohedral blocks. They are subsequent to the aplite and quartz-porphyry, but probably prior to the Big Butte rhyolite.

Their mineralization is most intense at the southerly end of the system, where the copper "dome" in the east-west veins is found, and it may be reasonably regarded as an extension of this into the later fractures.⁴¹ Most significant is the entire absence of the massive quartz-pyrite type of ore from these faults, an absence that must be regarded as corroborating the evidence already given of its priority to the copper mineralization. In respect to the copper minerals themselves, however, there is little distinction between the fault veins and the earlier fissures. Possibly less enargite, more bornite, and more chalcocite characterize the typical aggregate in the later ores.⁴² Sphalerite is found in the fault veins, in zones peripheral to the copper, and probably represents largely the original zinc content of these solutions, carried to remoter and cooler

⁴⁰ Ray claims to have found some residual zinc among the copper minerals, but such occurrences, while frequent on the border of the copper zone, must be very rare within it.

⁴¹ In Fig. 25 the range lines of the ore shoots in the Northwest Vein Series are arranged to show the "doming" in the copper mineralization of these veins. The veins on the edges are veins near the borders of the copper area of mineralization.

⁴² See Sales, *Trans.* (1913), 46, 3, for detailed discussion of these variations in the copper solutions.

regions before precipitation.⁴³ It is certainly a later generation than the residual sphalerite in the older ores, although Gidel's migration hypothesis might explain even this without necessitating the presence of zinc in the copper mineralization. Like the later ores of the western area, this zinc is accompanied by manganese, and on the whole we prefer to attribute both occurrences to peripheral phenomena of the copper "dome."

4. Northeast Fault Veins.—The mineralized fault veins of northeast strike are so intimately associated with the northwest system as to strongly suggest mechanical resultants of that movement. Their age, also, must be nearly the same, for while normally they cut and displace the northwest faults it is not infrequent for renewed movement on the latter to reverse this relation. The mineralization is nearly identical, but is less abundant in the later fractures, which mark its waning and final cessation. All the minerals of the earliest copper veins are repeated in these, the latest and last, but chalcocite is increasingly predominant and enargite insignificant.

5. *Summary*.—This is the briefest possible recapitulation of the bare facts. Interpretations of their genetic significance must and do vary, and we can here only suggest one that seems to fit the internal evidence of the district as well as the general tendencies of the batholith.

In the first place, the early mineralization of the east-west veins, which is known to slightly antecede the copper period,⁴⁴ may safely be correlated with the aplite type so widely distributed throughout the granite mass. In mineral composition, in reference to the granite surface⁴⁵ and in primary zoning, it possesses the requisite characters. It seems clear that at this stage no unusual sources of mineralization had yet been tapped, although quite possibly the quartz-porphyry dikes had already penetrated upward to their present position.

Structural evidence within the veins seems, in the second place, to postulate a slight pause after the conclusion of this period—possibly representing the delay while new disturbances of the quartz-porphyry reservoirs released the great copper-bearing solutions that now began to migrate into the east end of the fissure system. At their earliest stage these solutions seem to have carried all the elements that enter into the

⁴³ Sphalerite has been shown in the laboratory to precipitate at lower temperature than the copper sulphides.

⁴⁴ This idea is also held by W. H. Weed, *Trans.* (1903), 33, 747.

⁴⁵ "The older primary quartz-pyrite veins were reopened by later movements, correlated with a period of volcanic activity; and were penetrated by hot alkaline waters carrying copper and arsenic in solution."

⁴⁶ The projection of the granite cover over the Butte district involves more than the usual difficulties, but it can nevertheless be approximated as the flat upper contact of the granite is known to the southeast, south, west, and northwest, and roof pendants or large inclusions of andesite are found within the productive area itself. Such foreign rocks are never found elsewhere very far from the granite contact.

composition of the many copper-rich sulphides that characterize the Butte district, so that from the first, enargite, tennantite, tetrahedrite, bornite, covellite, and chalcocite were formed in the veins. Intense rock alteration, marked by sericitization, pyritization, and silicification, also marked the copper period from the beginning. Throughout the formation of the northwest and northeast faults, this mineralization continued, undergoing the gradual transitions already outlined, but without any real cessation until its final stop, probably just before the rhyolite eruptions to the west.

This long-extended period of ore deposition we attribute to the contribution of deep-seated reservoirs, probably fractional residues of the quartz-porphyry period—the quartz-porphyry itself, it will be remembered, being merely a partially differentiated, slow-cooling aplite. Such deep sources had the advantage of further fractionalization than the more superficial residues of the aplite stage, and they contained a higher concentration of metallic contents. It is doubtful, however, whether this alone is adequate to explain the great richness and amount of the copper deposits, and it seems that possibly an originally high copper content of the local portion of the granite magma is required in addition. The copper-rich belt that has been mentioned as crossing the batholith from Wickes to Butte may be a significant guide along this line of speculation.

As might be expected with the above interpretation of the quartz-porphyry and the mineralization, the ores of Butte are found deeper within the batholith than in the case of the true aplite-derived veins. These, as we have seen, carry their lead and zinc not lower than 1,000 or 1,200 ft. within the granite. These depths will also include the upper zones of the early Butte mineralization. The shallow mines of the western and northern sections of the camp can hardly have penetrated that far into the batholith, since its cover is now found to the north, west, and south at elevations lower than many of the shaft-collars, and the contact is everywhere nearly flat.

The copper ores, however—a primary enrichment by solutions from a deeper reservoir at a later period—found the precipitating conditions at lower points, so that they range from 1,000 to 1,200 ft. within the granite at their outcrop to 4,000 ft. as now developed, and show no signs of an approaching bottom. A suggestion that this great vertical range is due to the long continuance of the mineralization, and the downward migration of the precipitating point, is given by the tendency of the later ore shoots, as in the northwest faults, to begin at lower elevations than is the case in the east-west veins⁴⁶ (see Fig. 25).

⁴⁶ This tendency is also graphically shown by contrast of the longitudinal sections shown in Plate 5 and Fig. 6 with Fig. 6a in *Ore Deposits at Butte, Mont.*, by R. H. Sales, *Trans.* (1913), 46, 36, 74.

The only other deep ore deposit of the batholith, the Granite-Bimetallic at Philipsburg, also shows evidences of later primary reënrichment. (See Figs. 11, 20, 24.) Of this ore W. H. Emmons writes as follows:⁴⁷

"The paragenesis indicated above suggests that the ore first deposited consisted mainly of quartz, rhodochrosite, calcite, pyrite, arsenopyrite, stibnite, tetrahedrite, tennantite, galena, and zinc blende. Pyrargyrite, proustite, realgar, and orpiment are present in this ore in smaller amount, and it is possible that they have formed at the same time. After the first deposition of the ore the veins at some places were fractured and reopened. Some of these fractures were parallel to the walls and others cut across the veins and produced slight faults in the bands of vein quartz.

"Solutions more highly carbonated, presumably introduced from below, deposited much rhodochrosite, some calcite, and quartz, with a minor quantity of the sulphides,"

In the above statement, the earlier ore is seen to be, aside from the listing of the rare minerals, essentially similar to the early Butte ores, and to that of the aplite veins in general. The later solutions are entirely different, and it is probable that wherever found these, the products of an extra fractionalization, carry to an extreme point the original peculiarities of the magma in that region.

3. RHYOLITE PERIOD

A. *Early Rhyolite Phase*

1. *General Description.*—The early rhyolite is a rock widely prevalent over the northern end of the batholith. It is considerably later in age than the granite, having flowed out on an eroded surface of that rock. It is also subsequent to the Eocene peneplain, with which its lower contact often coincides, but is prior to the inner valleys and the later Tertiary gravels. The north-south normal faults occur in the rhyolite as well as the earlier formations. Probably Oligocene age most nearly fits these conditions.

The magmatic waters of these rhyolites contain typically only silica and iron; and quartz and pyrite, either in disseminations or veinlets, form the bulk of their mineral deposits. Accessory gold furnishes the commercial element, and mechanical enrichment by descending waters is generally required to raise this to a profitable grade.

2. *Disseminations.*—The best examples of disseminated ores are the mines strung along the Tenmile-Basin Creek Divide south of Helena (see Fig. 13). They exploit low-grade gold ore in the early rhyolite dikes and flows that cap this ridge. The Porphyry Dike may be taken as a type.

⁴⁷ Emmons and Calkins: *Op. cit.*, 177.

The rhyolite here consists of a succession of alternating breccia, ash beds, and porphyritic flows with an intrusive dike along the western edge of the series. The effusive rocks lie upon a somewhat irregular granite floor, which must by the basic character of that rock, be nearly identical with its former contact surface. Faults of the north-south Continental type cut and displace all the rock series.

The gold is primarily associated with finely disseminated pyrite in the fine-grained intrusive rhyolite and has been enriched along the vertical joints of this dike. The breccia and ash beds are practically barren, but the flows carry some gold. Samples along the faults are very erratic, suggesting mechanical transportation of the precious metal, with impoverishment in some places and enrichment in others. Practically the entire gold content can be recovered by amalgamation, which suggests that the native metal itself must be a primary mineral of the rock. This rhyolite gold is low-grade, running from \$7 to \$11 per ounce, and can, even in the placers, be readily distinguished from the \$16 or \$18 gold of the granite period veins.

3. *Impregnations*.—This term covers the instances where the ore deposit has been formed by the penetration and diffusion of magmatic waters into and through portions of the rhyolite. The Woodrow Wilson, south of Rimini, is a good example. The ore here is a rhyolite of the early period intensely silicified and pyritized, and traversed by numerous quartz-pyrite (largely oxidized) stringers. A certain amount of gold accompanies this alteration and mineralization. The occurrence differs from the Porphyry Dike in the evidence of hydrothermal action, and the contribution of silica and iron to the rock. The source of these additions is probably the deeper portion of the same dike.

4. *Contact Deposits*.—The only occurrence of contact mineralization at the edge of the early rhyolite is the Monte Cristo mine in the Rimini district. Here an orebody has formed on the wall of a rhyolite mass intrusive into the granite. The ore consists of specular hematite and cuprite, with some very rich silver. Development has been shallow, so that no evidence of the character or persistence of the deep-seated ore is available.

B. Dacite (or Later Rhyolite) Phase

1. *General Description*.—These rocks are readily placed as to age, since their ash and tuff phases are intercalated with the Pliocene gravels of the Tertiary valleys. They are thus long subsequent to the cooling of the granite, and were extruded upon a surface essentially in its present form. In some localities, notably north of Butte, these dacites have piled up to a thickness of over 1,000 ft., with individual flows ranging from basic andesitic types to quartz-porphyrines and true rhyolites. Some quartz is almost always present, however, and biotite is very char-

acteristic of all the phases. In a general way the upper flows are more acidic than the lower ones, and this tendency is seen also in the intrusive areas; where more acid types penetrate earlier more basic masses.

2. *Fissure Veins*.—This dacite phase lacks, in its ore deposits, the disseminations, segregations, and contact phenomena of the more energetic igneous periods, and its production is confined largely to fissure veins, filled by magmatic solutions. The mineral filling is almost always limited to quartz and pyrite, lacking the sulphides of the rarer metals. Much of the silica is amorphous and chalcedonic, and the frequent calcite increases the impression of cool and shallow conditions of deposition.

The best examples of this type are the mines of Lowland Creek, the Ruby, Kit Carson, Memphis, and Gopher. The first-named has been well described by Knopf.⁴⁸

"The ore-bearing zone extends from the main shaft to the Columbia claim, a distance of several thousand feet, the general trend being S 20° E. Within this zone the ore occurs in shoots or more properly in parallel veins . . . disposed roughly in echelon fashion. . . . The ore consists of angular dacite fragments cemented by quartz, calcite, and minor adularia. The adularia, where embedded in quartz and calcite is not easily distinguishable. . . . The quartz is commonly clear, glassy, and drusy, but where it is solid it is of compact saccharoidal texture. Some of the siliceous veinlets show a porcelain-like texture, but such cryptocrystalline quartz is far less common at the Ruby mine than at the surrounding properties. The sulphides, which are confined to the gangue material that cements the dacite fragments, comprise pyrite, argentite, and possibly others. . . ."

This mineralization is typical of many such fissures and shear zones in the later rhyolite, wherever they are found.⁴⁹

C. Summary

Thus in both rhyolite phases the normal mineralization is generally restricted to the commoner elements, silica, lime and iron, contributed to the cooled intrusions from deeper-seated portions of the same material. As might be expected from the slight erosion that has taken place since their occurrence, the rhyolitic deposits have the characters suggesting formation at shallow depths. Knopf summarizes their character as follows:⁵⁰

⁴⁸ Knopf: *Op. cit.*, 126.

⁴⁹ Knopf places many of the orebodies of the Clancy district in this dacite period. Among these are the King Solomon, Free, Coinage, Fleming, etc. From our own investigations we doubt whether in some of these cases, the dacite has done more than enrich preëxisting shoots, so that in want of conclusive evidence we will leave these occurrences out of the discussions.

⁵⁰ Knopf: *Op. cit.*, 55.

"The orebodies are mainly fissure veins of branching and irregular character. . . . As a rule, the orebodies carry subordinate quantities of sulphides, and are worked for their content of precious metal alone. A prominent feature of the deposits is the prevalence of crypto-crystalline quartz in the gangue material. The crypto-crystalline quartz appears in three varieties: (1) A dark gray blue flinty variety; (2) a dense-grained white variety resembling porcelain; and (3) chalcedony. These three modifications, however, are likely to be present together in the same deposit, confusedly intergrown and each variety grading into the other. . . .

"The metasomatic alteration of the wall rocks of the late Tertiary veins is principally in the nature of a thorough sericitization, accompanied by the introduction of carbonates and locally by the development of chlorite."

These statements of Knopf are in essential accord with our own observations of ore deposits of the two rhyolite periods, although more applicable to the later dacite stage.

The characters found are those approached by the weak dying phases of other periods of mineralization, and these deposits are unique in the absence of the stronger hot temperature features rather than in the possession of the chalcedonic quartz and calcite.

PART 3.—CONCLUSIONS

A. ASSOCIATION OF ORES AND IGNEOUS ROCKS

1. *Magmatic Source of Vein Filling*

The ores are associated with intrusive rather than extrusive forms of igneous rocks. Exceptions are Baggs Creek, Porphyry Dike (partly), Emery (partly), Evening Star, and Elkhorn iron mines, none of which are very important. In the cases of Emery and Evening Star their position is due to the favorable cavities in flows, with a probable source below. Intrusive phases have produced at least 99 per cent. of the mineral wealth in the area described. Therefore, mineralization depends upon properties held exclusively by intrusives, which include: (a) the chance for differentiation and fractional crystallization, with the accumulation of magmatic residues in deep-seated reservoirs; (b) the development of fissures, by shrinkage, which tap such reservoirs. Hence we conclude that magmatic waters have been the predominant source of ore deposition in every igneous period. We see no reason why meteoric waters should thus favor dense crystalline intrusives over bedded breccias and tuffs and porous flows. Against lateral secretion is the undoubted fact that the original disseminated deposits, such as Red Rock, Heddleston, and Porphyry Dike, have remained disseminated, with no primary

concentration into veins. Acceptance of a plutonic magmatic source for the ore deposits seems quite unescapable.

2. *Intrusive Wall Rocks*

It follows that the igneous periods which are most largely of intrusive type have the most mineralization. Hence the granite period leads the others while the andesite comes ahead of the rhyolite.

3. *Ores and Successive Igneous Phases.*

Within an igneous period, the successive phases have increasing proportions of vein formation, which suggests the dependence of this upon magmatic differentiation. Thus, in the granite period the ores of the granite phase are neither proportionately as numerous nor as important as those of the aplite phase, which has only 10 per cent. or less of its extent, while the very small quartz-porphyry phase carries a relatively large and rich mineralization.

B. PROGRESSION OF MINERALIZATION

1. *Common Order*

In each phase the mineralization follows a common order: first, the products derived from the outer portion of the intrusive, found as border disseminations and differentiations and as contact deposits; second, primary segregations within the mass of the rock; and third, contribution from deep-seated reservoirs to fissures and faults.

These have connecting links. The border differentiations are of the basic and quickly crystallizing elements, while the internal segregations are a siliceous and metal-bearing residue from slower cooling. These latter in turn are analogous to the sources of the fissure mineralization, and represent probably the smaller reservoirs that were not tapped by any channels of circulation. Thus the whole succession follows the accepted laws of the crystallization of igneous magmas.

2. *Common Variation in Character*

The character of vein filling also shows a progressive change within the limits of an igneous period. The earliest ores, border segregations and contact deposits, are the highest in iron; and magnetite, hematite, pyrrhotite, pyrite, and chalcopyrite are found with little accompanying quartz. The first fissure veins most nearly approach the border types of mineralization. Thus, in the early granite fissures, quartz and pyrite

with accessory gold are the predominant minerals, the pyrite often in excess. In the later stages, as at Scratch Gravel and Marysville, lead, with its accompanying silver, becomes more important, while the iron content has decreased appreciably.⁵¹ Similarly, in the aplite phase the early ores are tourmaline, quartz, and pyrite with gold, following the type of the pegmatite segregations. The base metal sulphides, with accompanying silver, appear in the later vein filling. In the quartz-porphry phase the long-continued mineralization shows progressively decreasing iron and increasing copper in the solutions while gold is present only in the most minute quantity. All the phases in their dying stages show a highly siliceous type with little sulphide content, and with chalcidony and calcite in some amount.⁵²

C. RELATION OF ORE DEPOSITS TO FORMS OF INTRUSION

1. *Contact Deposits*

The border deposits, included in the first of the above divisions, can generally be referred to certain forms of the intrusive rock. They particularly favor protruding cupolas or peninsulas with their facilities for rapid cooling. Such masses both contain the great proportion of disseminations and segregations and have most often contributed the solutions to outlying contact deposits. Small dikes, on the other hand, are of little value in this respect, possibly because too quick chilling has prevented the segregation or escape of the magmatic waters. Where the intrusive mass is of laccolithic form, as in the Golden Sunlight mine, the upper contact is the locus of mineralization, while the lower one is comparatively barren. Thus gravity appears to occupy a subordinate place in fixing the position of such ore deposits except where, as in basic flows, the elements of pneumatolysis and magmatic water pressures are negligible.

2. *Segregations*

As has been stated above, the deep-seated segregations are acid, and are apt to contain such high-temperature minerals as tourmaline, hubner-

⁵¹ In fault veins later than the fissure veins of the Empire mine, Marysville district, silver is the important ore mineral and gold has ceased to be an appreciable constituent of the mineralizing solutions.

⁵² This progression in type suggests rather wide possibilities of correlation with the parent magma. The early ore deposits are derived immediately from the borders of the intrusive rock, which is in the great majority of instances more basic and higher in iron than the deeper-seated portions. Hence both segregations and mineral residues will reflect in their composition this feature of their source. The later deposits, such as fissures penetrating through the chilled crust and tapping deeper portions of the magma, will by their decreased iron indicate the relatively greater acidity of these regions, not affected by the basic contact differentiation. Thus the orebodies seem to reflect the variation in the igneous mass, slowly cooling and crystallizing from its surface toward its interior.

ite, molybdenite, and cassiterite. They are most frequent in regions where fissures are rare, and are generally of such small size as to have been readily missed by even abundant fissuring.

3. *Fissure Veins*

The fissure ore deposits are generally situated above the high points of their accompanying igneous intrusions (see Fig. 23). Thus the granite fissures top the outlying cupolas of the batholith, and are absent from its central portion, where erosion has obliterated these protruding masses. The aplite and quartz-porphry fissures, likewise, are above the main bulk of these rocks, accompanying dikes which have probably pushed upward from larger masses in depth. This is particularly evident in the case of the Butte quartz-porphry. The undoubted fact may be due to several interacting causes. (a) If preëxisting fissures were available, the igneous rock would push ahead along these lines of weakness. (b) Such local projections would cool quickly and form internal fissures. (c) These, and the preëxisting fractures, would become convenient outlets for the metalliferous and siliceous residues of the slower cooling magma below. In any case, strong upward pressure for the solution is necessitated.

D. GEOGRAPHIC VARIATION

In the case of the aplite fissures the geographic variation of the vein filling has been described. Some such differences are also apparent with the ore deposits of other phases. Thus the contact ores of the granite phase are predominantly of the iron-gold type to the west but are relatively high in copper to the south. The granite fissures show little variation, although somewhat more apt to contain lead and zinc along the northern and eastern borders of the batholith than elsewhere. With the fissures of the aplite stage, provinces of zinc-rich, lead-rich, arsenic-rich ores can be readily distinguished. Still greater variability is suggested by later mineralizations which in Butte bring in the unusual copper-rich sulphides, enargite, bornite, chalcocite, etc., while in Philipsburg complex silver minerals were introduced.

Since all these ore deposits are of undoubted primary origin with sources within the igneous bodies in which they occur, such variations can be referred only to differences between the contents of the reservoirs. As these are the end products of a series of fractional crystallizations, the varying proportions of the different elements are in turn carried back to an original irregularity in their distribution in the parent magma. The facts are in further accord with this conception. The earlier veins, products of the first crystallization, are far less diversified than those formed after the later stages, and in the latest ores of all, as we have

already seen, the variation is most pronounced. If the metallic contents of the original magma were left behind and concentrated during a succession of partial crystallizations, the geographic variation would be accentuated in the later phases of vein formation. This is, in fact, the case.

E. FACTORS GOVERNING VERTICAL DISTRIBUTION OF ORE SHOOTS

1. *Mineral Ranges*

Within the veins themselves certain tendencies are widespread. Most noticeable is the decrease, with depth, of the proportion of sulphides to quartz (see Figs. 17, 18, 19, 20, 21, 22, 24, 25). This is true of all the veins of all periods.⁵³ In mineralization of long duration, as in Butte, there is greater range of sulphide precipitation than in any of the more restricted phases (Fig. 25). Of the sulphides, pyrite exhibits the greatest range, and is generally found from top to bottom of the veins. Galena, also widely distributed, seldom occurs in the lower portions, but travels far in the cooler solutions of the upper zones before precipitating. Its range is essentially that of arsenopyrite also. Sphalerite is a close associate of galena, but its greater portion comes down at points below the horizon of greatest galena precipitation to which it forms a sub-zone of comparatively shallow depth. The copper minerals are generally associated with the deeper horizons, and appear in greatest amount below the galena and sphalerite zones.

The manganese minerals, rhodocrosite and rhodonite, are oftenest found well toward the top of the mineralization. Some elements are found in different forms at different horizons: Scheelite is associated with the upper gold-pyrite ores of the granite fissures while hübnerite occurs with galena and sphalerite in the central zones of the aplite veins; stibnite and arsenopyrite are found at or above the lead-zinc horizon, while tetrahedrite and tennantite accompany the deeper copper ores.

Too definite boundaries cannot be assigned to these horizons, and since dying phases of mineralization, or later renewals, may encounter

⁵³ In the longitudinal sections (Figs. 18, 19, 20) the largest orebodies of each stage of mineralization, in which the bottoms of the richer ore shoots are approximately known, were selected. The sections show the decrease in the grade of the ore by the smaller sizes of the stopes at each successive level in depth, and in the case of the Drumlunnon mine, an absolute absence of minable ore is shown on the lowest level. The larger the shoot of ore is laterally, the greater its vertical dimension usually is, so that the conclusions so forcibly shown by the longitudinal sections of the largest mines of each type may be applied without reservation to the smaller ore shoots of the same types. The ore shoots of the Hecla mines and the Elkhorn mine are also smaller and leaner on the lower levels. In Butte this tendency has not yet become manifest in the primary ore shoots, either of the zinc or copper type.

precipitating conditions at lower levels than the first energetic outbursts, there will necessarily be much overlapping and migrating downward of zones. For a given stage of mineralization, however, the succession outlined above and recapitulated below seems to be universal and to rely on fundamental causes.⁵⁴

	Top. Much Sulphide	Bottom. Little Sulphide
Stibnite		
Rhodocrosite		
Arsenopyrite		
Galena		
Sphalerite		
Chalcopyrite		
Tetrahedrite		
Bornite		
Pyrite		
Quartz		

⁵⁴ The idea of a primary zonal arrangement in the metallic sulphides deposited from hot ascending solutions is not a new one. Several writers on mining geology have mentioned the chemical probability, and many instances are cited in the literature on ore deposits. A series of relative solubilities for metallic carbonates has been established by R. C. Wells and others, and a series for the orderly precipitation of the sulphide minerals from cold acid solutions has been determined by E. Schuerman. With the exception, however, of a statement by F. W. Clarke that sulphides of tin, arsenic, and antimony are more soluble than other sulphide ores in alkaline sulphide solutions, we have found no reference to this group, most important in the consideration of primary zoning of ores.

(A) W. H. Weed: Ore Deposition and Vein Enrichment by Ascending Hot Waters, *Trans.* (1903), 33, 751.

"It is well known that the metallic sulphides are soluble in alkaline solutions under heat and pressure. . . The more soluble substances will be carried further upward before precipitation, and one might even suppose, if the solubilities of the substances were sufficiently unlike, that zones would be found each one of which consisted mainly of the particular substance thrown out by the change of pressure. This would produce an orderly distribution of the ores in a vertical direction. . . In the writers' own experience (Geology of the Castle Mountain Mining Districts, Montana, *U. S. Geological Survey, Bulletin* 139 (1896). Geology of the Little Belt Mountains, *U. S. Geological Survey* (1900), *20th Annual Report*, Part III, 271-461) the order appears to be galena on top, passing into highly zinciferous ores below, and this into low-grade pyrite."

(B) *Types of Ore Deposits*, edited by H. F. Bain, 347. San Francisco, *Mining and Scientific Press*, pub., 1911.

"An apparent, but deceptive, relation of the nature of the ore to country rock may sometimes be due to the fact that, in solutions rising in a fissure, as the pressure and heat diminish, certain ores may be deposited at a certain depth and others at another depth, according to the conditions of pressure and temperature that permit the precipitation of each."

(C) Beyschlag, Vogt, and Krusch: *Ore Deposits*, 1, 211.

"Since pressure and temperature, which are regarded as causing these variations in primary deposition, generally promote the solubility of substances, those then which are the most easily dissolved would separate the less readily from solution and would consequently be found in the neighborhood of the surface. This is probably

The evidences of this, for a particular phase in particular districts, are seen on the accompanying zoning diagrams (Figs. 21, 22, 24).

Wherever the tops of ore zones are visible in several veins in a district, it is apparent that all the horizons reach higher levels in large than in small veins. Each zone is also apt to cover a greater vertical range in the large fissures.

the reason why minerals which form many easily soluble compounds are found in the upper levels. Again, differences in the filling of deposits, such as may be referred to change of temperature and pressure, may make themselves evident not only in the particular ores deposited, but also in the internal structure of the deposit."

(D) Such vertical arrangements of minerals due to primary mineralization have been noted by others in widely separated districts. A brief list of some of the better known occurrences follows, the order of naming the predominant minerals being from the top downward.

1. Change in character of mineralization:

(a) Castle Mountains, Montana, W. H. Weed: galena; sphalerite; low-grade pyrite.

(b) Little Belt Mountains, Montana, W. H. Weed: galena; sphalerite; low-grade pyrite.

(c) Neihart, Montana, R. H. Sales: galena; sphalerite and pyrite; quartz and carbonates of iron, calcium, and manganese.

(d) Dolcoath mine, Cornwall, C. Le N. Foster: copper to 1,000 ft., predominately in slates; tin below, chiefly in granite.

(e) Clausthal district, Germany, Vogt: galena; sphalerite.

(f) Berg district, Germany, Vogt: galena; sphalerite; siderite.

(g) Pyrenees Mountains near Angeléze Gazost, Vogt: galena; sphalerite.

(h) Wood River district, Idaho, Lindgren: silver; galena; sphalerite; pyrite and quartz.

(i) Slocan district, B. C., Lindgren: rich silver, galena, sphalerite; poorer ores with more siderite, pyrite, and quartz.

(j) Coeur D'Alene district, Idaho, Ransome: rich silver; galena; poorer ores with more pyrrhotite, pyrite, and sphalerite.

(k) Przibram district, Bohemia, J. Schmid: galena, sphalerite, chalcoppyrite, pyrite, and arsenopyrite, with a gangue of calcite, siderite, and quartz; quartz and sphalerite increase proportionately.

(l) Hualpai district, Arizona, F. C. Schrader: rich silver, galena, sphalerite, chalcoppyrite, and pyrite; galena decreases and chalcoppyrite increases to 700 ft. depth.

(m) Freiberg, Saxony, Alte Hoffnung Gottes Mine, R. Beck: galena decreases until at 1,850 ft. it is almost entirely replaced by sphalerite and pyrite. The silver content of the galena has also decreased.

(n) Bingham Canyon, Utah, Perry, Locke, and Bateman: relative to igneous rock, outer zone has predominant galena; median zone sphalerite; and inner zone pyrite and chalcoppyrite.

(o) Bersbo mine, Sweden, Sjoengren: copper decreases and zinc increases in depth. (This is a contact deposit, and the inversion of the usual relation may be due to the position of the ore shoot relative to the igneous contact.)

(p) Sulphur Bank, California, W. H. Weed: sulphur is found in the upper 200 ft., giving place to quicksilver ores below that level.

(q) Boulder County, Colorado, Lindgren: association of scheelite and ferberite with gold-telluride ores noted at shallow end of zone, giving place to the sphalerite, galena, chalcoppyrite, pyrite ores of Leadville at the deeper end.

2. *Factor of Progressive Cooling*

A comparison of the vertical position of ore shoots in the different phases of mineralization included in the granite period shows downward migration, with the lapse of time, of the critical conditions of precipitation. This is well seen in Fig. 25. Thus it is evident that while solubility, pressure, and chemical reactions must have borne important parts in the deposition of ore minerals, the factor that determined the general position of the shoots was the critical temperature of precipitation, modified, however, by the other conditions. In addition to lowering the horizon of precipitation to points further within the igneous rock, the progressive cooling and crystallization of the batholith necessitated the derivation of the vein-forming solutions from greater and greater depths. Thus we have both elements essential for the repeated phases of mineralization as they are now found—sources progressively lower in the batholith, and a horizon of precipitation at lower levels with each stage.

F. SUMMARY

Such are the conclusions that have resulted from the correlation and genetic classification of the ore deposits of the Boulder batholith. Over 500 mines, including most of the important producers of the region, have been personally examined and the entire area of the batholith has been geologically mapped. The ideas of ore deposition outlined above have evolved gradually during the accumulation of data, and have finally thus crystallized. Information has subsequently been obtained of new districts and mines, and this has without exception confirmed the conclusions already arrived at.

It has so happened, also, that both authors have recently seen much of the great Idaho batholith, and many of the theories evolved in Montana have proven applicable to this neighboring granite area, especially as regards the gold-pyrite veins.

A brief recapitulation of old conclusions confirmed and new ones reached may fitly close this discussion.

-
2. Decrease in metallic content in depth without change of type of mineralization:
 - (a) Rio Tinto mine, Spain, Vogt and Beck: decrease of chalcopyrite in depth.
 - (b) Tin mines of Bolivia, Vogt: stanniferous pyrite becomes poorer in depth.
 - (c) Falu mine, Sweden, Sjoengren: decrease in copper and silver in depth.
 - (d) Sala mine, Sweden, Sjoengren: galena decreases in depth to 100-ft. level; silver decreases more rapidly than galena.
 - (e) Empire district, Colorado, Spurr and Garrey: chalcopyrite decreases in depth more rapidly than pyrite.
 - (f) Freeland group, Empire district, Colorado, Spurr: galena, siderite, and rhodocrosite decrease on lower levels.
 - (g) Siliceous lead ores in limestone, Weed: decrease in galena and silver in depth.

1. The ores are associated with intrusive rather than extrusive forms of igneous rocks.

2. The most varied types and largest number of ore deposits are associated with the periods of greatest intrusive action.

3. Within an igneous period, the successive magmatic differentiates furnish increasing proportions of vein-forming solutions.

4. The time sequence of ore deposition in any igneous phase follows a definite order: first, contact and border; second, internal segregations; and third, fissure and fault veins filled from deep-seated sources.

5. The character of vein filling shows progressive change within the limits of an igneous period, the main feature being a decrease in the proportion of iron.

6. Projecting and quickly cooled masses of the igneous rocks, which favor moderately rapid crystallization, and ejection of mineralizers, are the general loci of border types of ore deposits.

7. Deep-seated segregations are more acid than peripheral segregations.

8. The fissure ore deposits are situated in and above the high points of their accompanying igneous intrusions.

9. There is a geographic variation in the composition of vein filling, which increases with successive phases of a period of mineralization.

10. A definite vertical primary zoning exists in the fissure veins.

11. The horizon of ore precipitation migrates downward through successive phases of mineralization.

All in all, it is evident that the ore deposits are essentially a superficial phenomenon, connected fundamentally with crustal conditions in the batholith, and only found in the depths where these conditions are accidentally repeated.

APPENDIX A.—BIBLIOGRAPHY

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APPENDIX B

A. Andesite Period.

All of the types of ore deposits described in this paper are found in the following districts:

1. Emery.
2. Elliston.
3. S. Baldy.*
4. Elkhorn.
5. Radersburg.*

B. Granite Period.

(a) Segregations and disseminations are found in the following mining districts:

1. Helena.
2. Sheridan.
3. Elkhorn.
4. Whitehall.
5. Heddleston.
6. Red Rock Creek.

(b) Contact replacements are found in the districts listed below:

1. French Gulch.
2. Highland.
3. Argenta.
4. Bannack.
5. Blue Wing.
6. Philipsburg.
7. Blue Eyed Nellie.
8. Vipond.
9. Red Lion.
10. Georgetown.
11. Utopia.
12. Polaris.
13. Whitehall.
14. Moose Creek.
15. Melrose.

(c) The following are the districts characterized by fissure veins of the Granite Phase:

1. Pony.
2. Garnet.
3. Scratch Gravel.
4. Marysville.
5. Gould.
6. Heddleston.
7. Racetrack.
8. Pikes Peak.
9. Georgetown.
10. Norris.
11. Mammoth.
12. Unionville.
13. Argenta.
14. Elkhorn (S).*
15. Polaris.
16. Melrose.
17. French Gulch.
18. Confederate Gulch.
19. Colima.
20. Top O'Deep.
21. Clinton.

Probably also the following districts belong to this group:

22. Winston-McClellan Creek.
23. Beaver Creek.
24. Indian Creek.
25. Radersburg.*
26. Bannack.
27. Tidal Wave.

And scattered individual deposits of other districts.

2. *Aplite Phase.*

(a) Segregations and Disseminations.

1. Red Lion District.
2. Basin District.
3. Rimini.

(b) Contact Replacements.*

1. Bryant.*
2. Elkhorn.*
3. Philipsburg.*

(c) Fissures.

1. Helena.
2. Rimini.
3. Elliston.
4. Jack Mt.
5. Basin.
6. Comet.
7. Wickes.
8. Prickly Pear.
9. Little Boulder.
10. Warm Springs Creek *
11. Philipsburg.*
12. Elkhorn (S).
13. Oro Fino.
14. Butte.*
15. French Gulch.
16. Clancy.
17. Peterson Creek.

3. *Quartz-Porphyry Phase.*

(a and b) No segregations, disseminations, or contact replacements of importance.

(c) Fissures. Several ages of fissuring connected with the mineralization of this phase at Butte. Possibly also represented in the later reopening of the Granite Bi-Metallic Vein at Philipsburg. No other indications of this type of mineralization yet discovered.

C. *Rhyolite Period.*1. *Early Rhyolite Phase of Mineralization.*

(a) Disseminations, contact, and fissure impregnation found only near Rimini.

2. *Dasite Phase.*

(a) Contacts and Fissures.

1. Clancy.
2. Warm Springs Creek.*
3. Lowland Creek.

* In the above lists the asterisks mark doubtful cases, the S. Baldy district being marked because the age of this andesite is not known; the Radersburg district because while the veins are entirely within andesite the solutions may have had their source in the granite; the aplite contact replacements at Elkhorn, Hecla, and Philipsburg because the only criteria for their classification is the character of their vein fillings; the Butte district under aplite fissures because these fissures may be of the quartz-porphyry phase of mineralization entirely; and the Warm Springs Creek district under Rhyolite fissures because of some uncertainty as to the age of the veins.

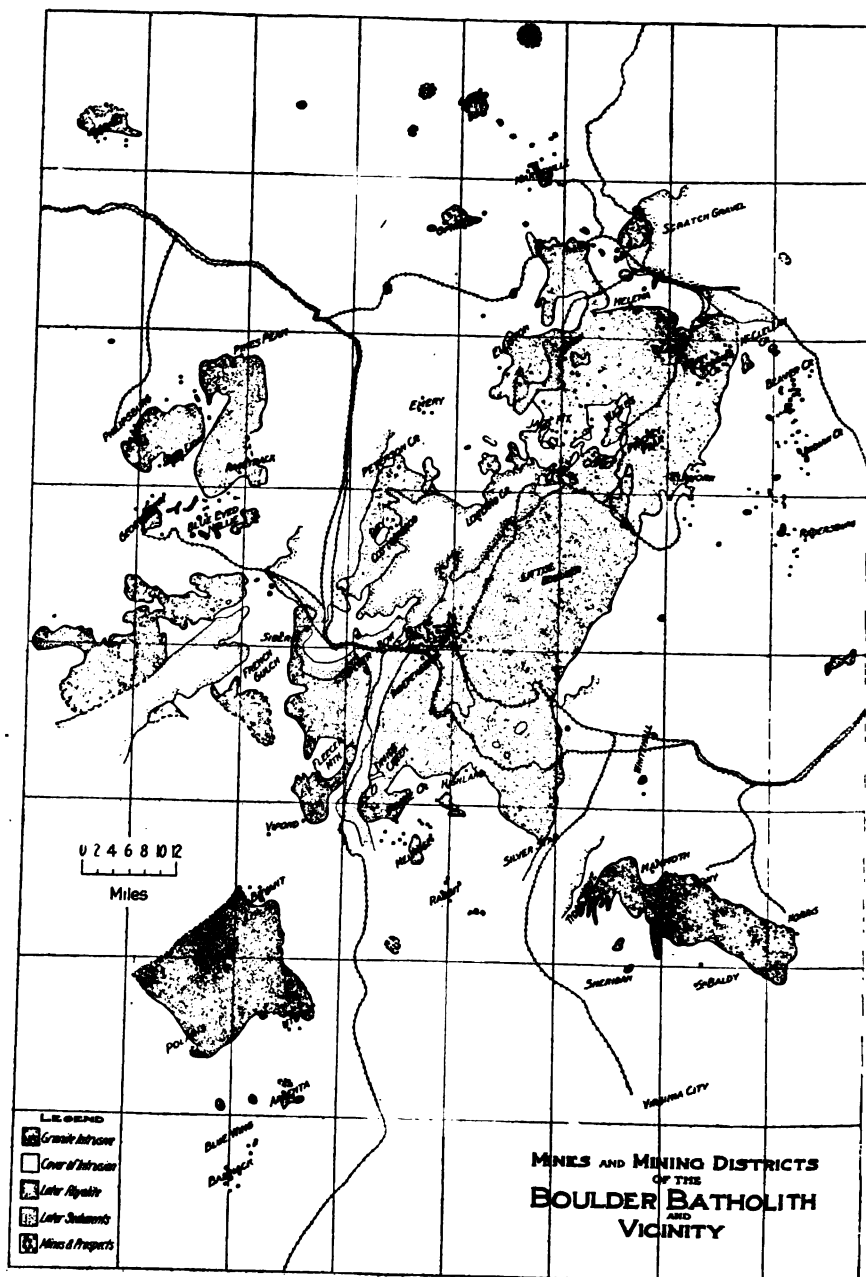


FIG. 1.—MAP OF GRANITE AREAS, MINING DISTRICTS AND MINES.

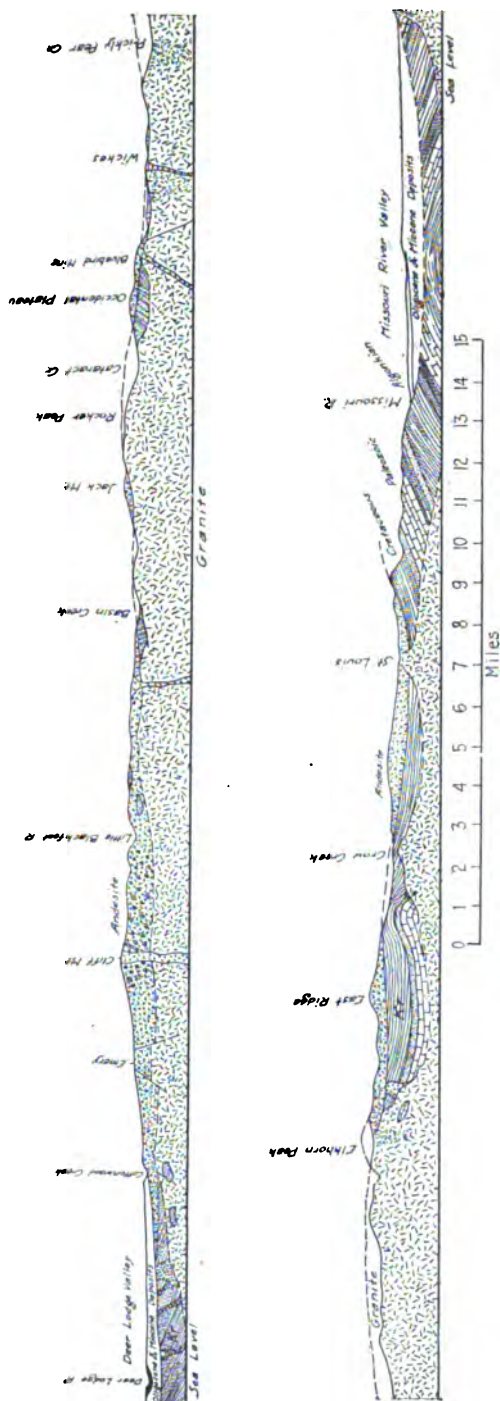


FIG. 2.—TYPICAL SECTION ACROSS THE BOULDER BATHOLITH.

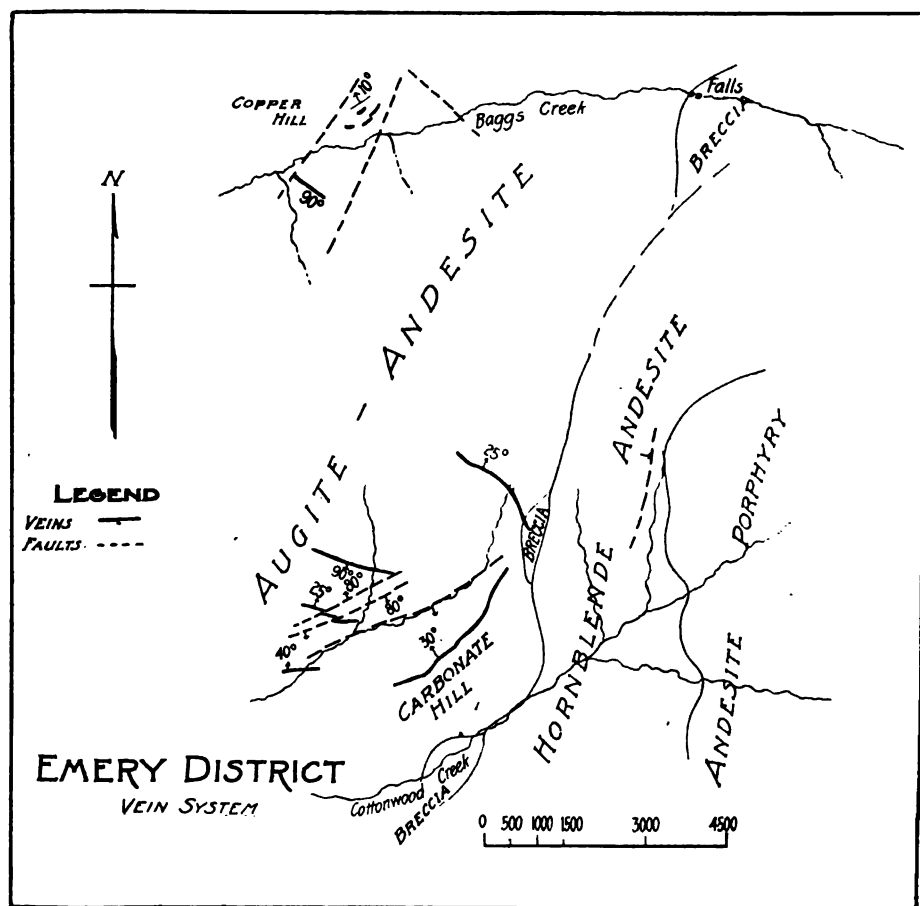


FIG. 3.—EMERY DISTRICT.

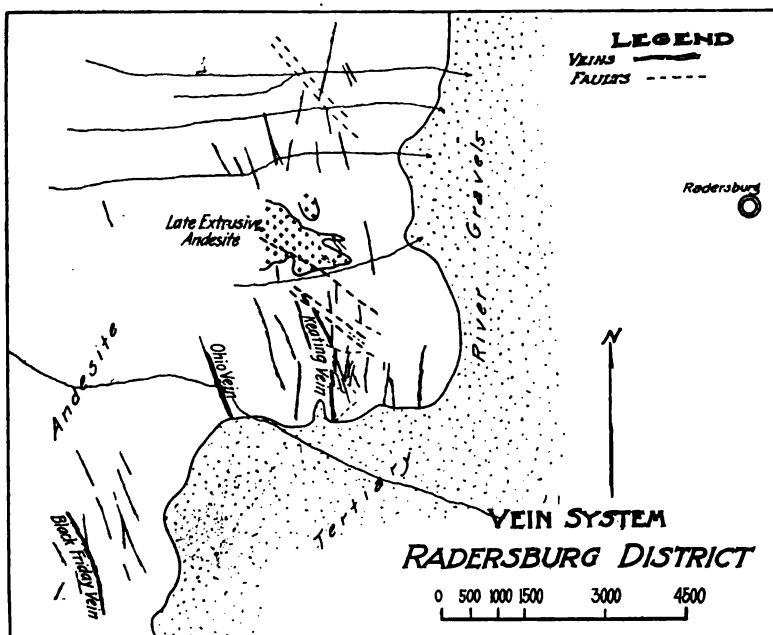


FIG. 4.—RADERSBURG DISTRICT.

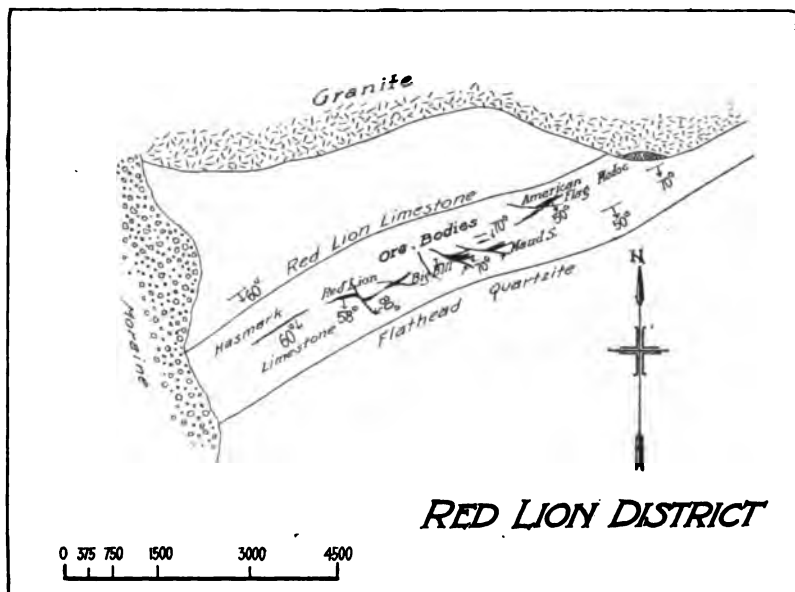


FIG. 5.

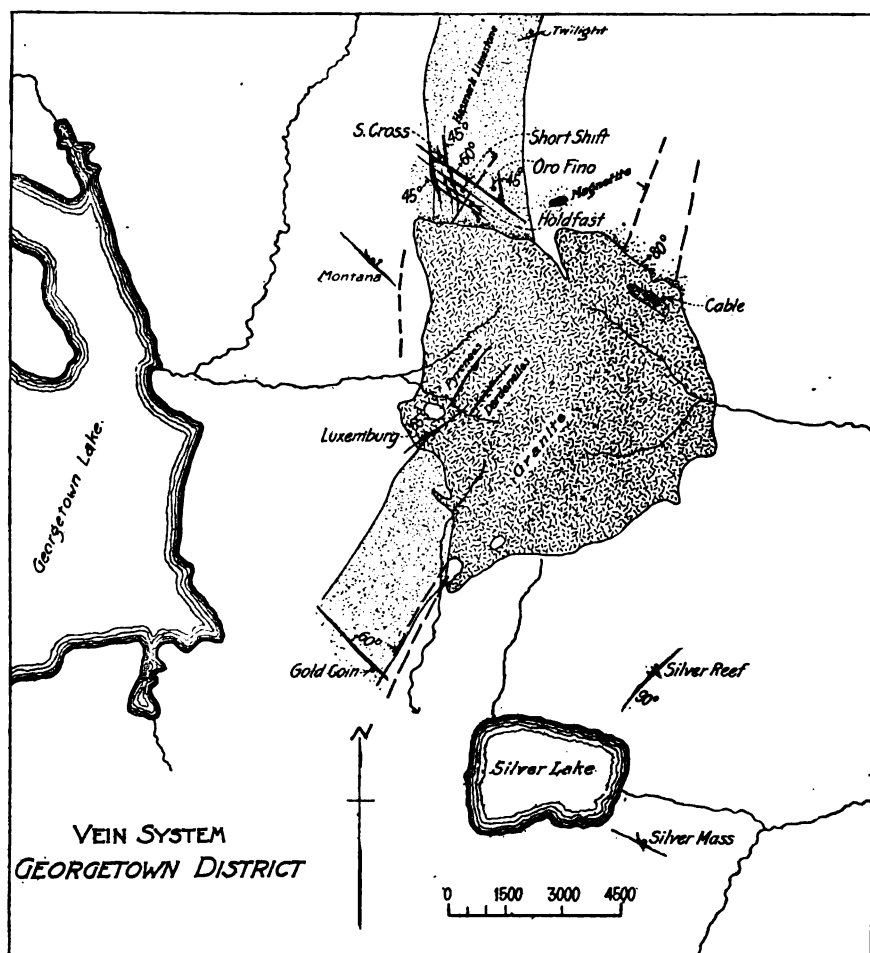


FIG. 6.—VEIN SYSTEM, GEORGETOWN DISTRICT.

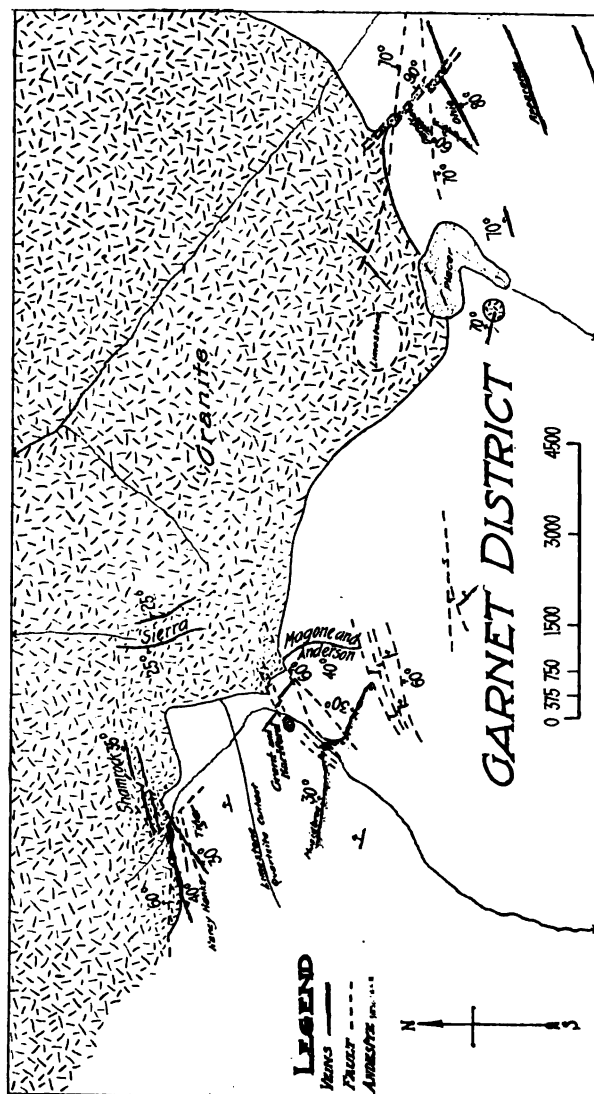


Fig. 7.

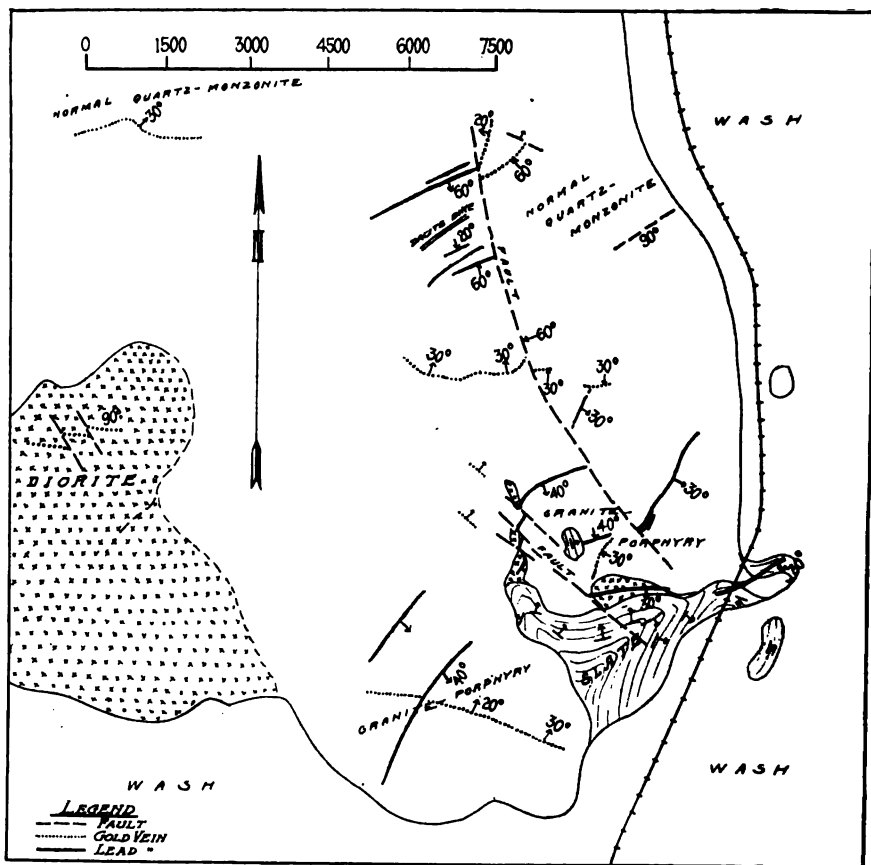


FIG. 8.—SCRATCH GRAVEL MINING DISTRICT, HELENA, MONT.

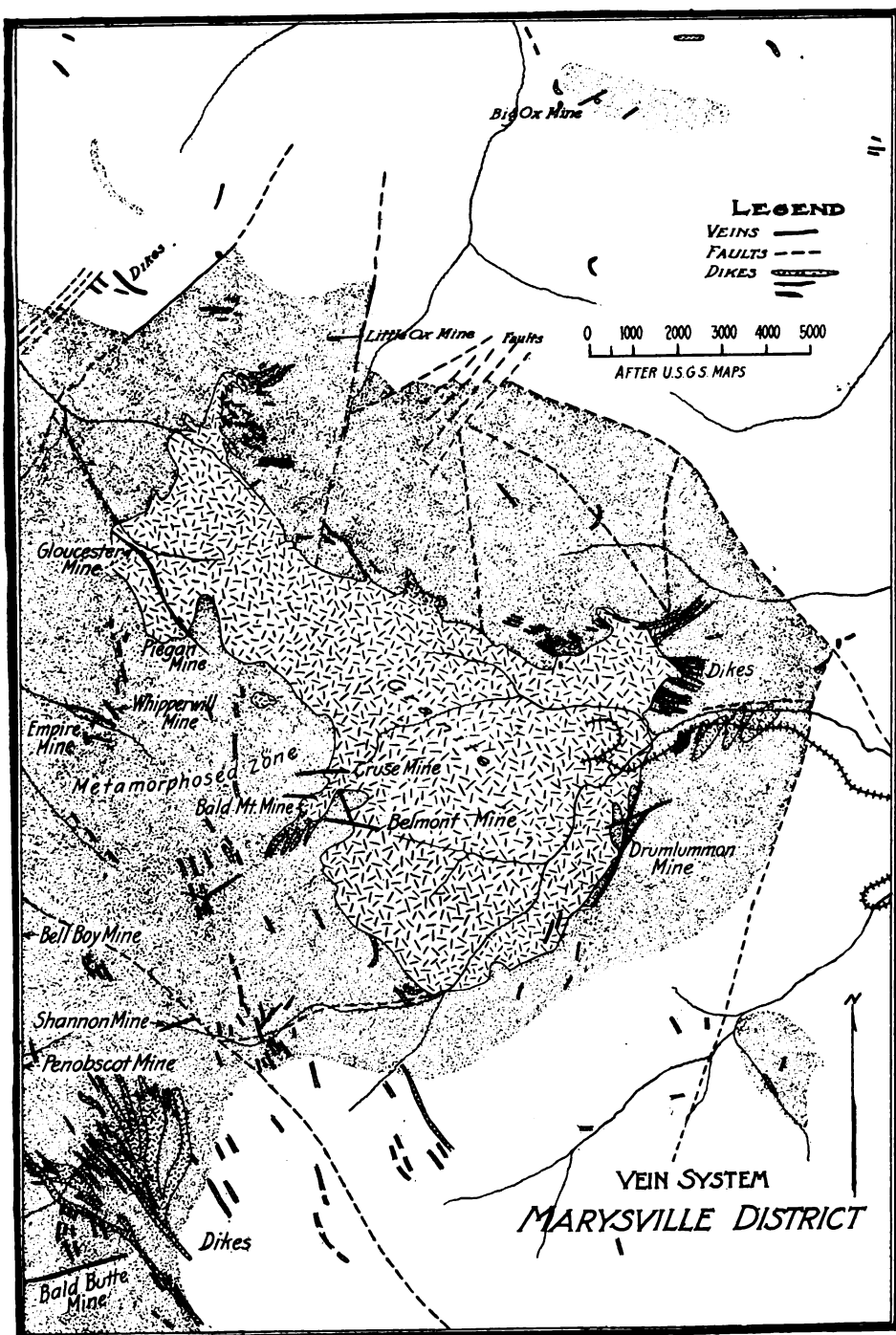


FIG. 9.—VEIN SYSTEM, MARYSVILLE DISTRICT.

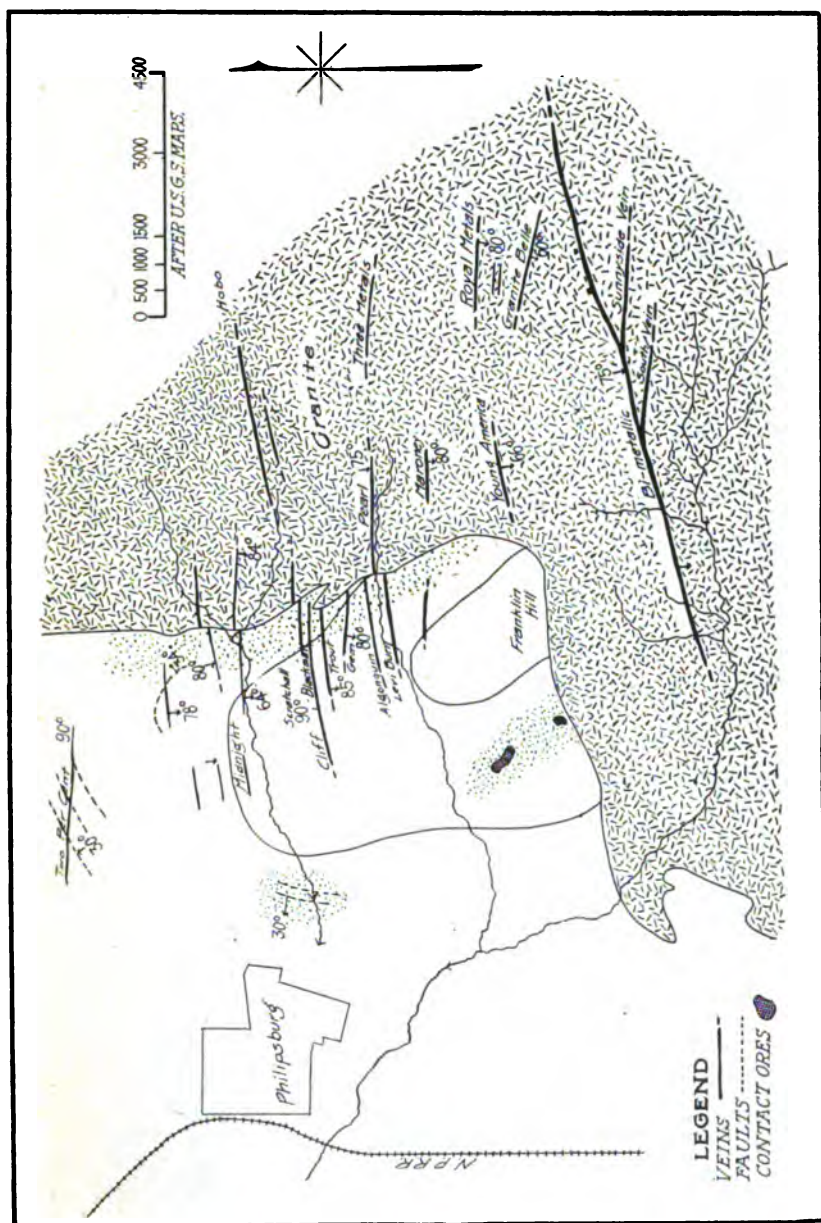


FIG. 11.—VEIN SYSTEM, PHILIPSBURG DISTRICT.

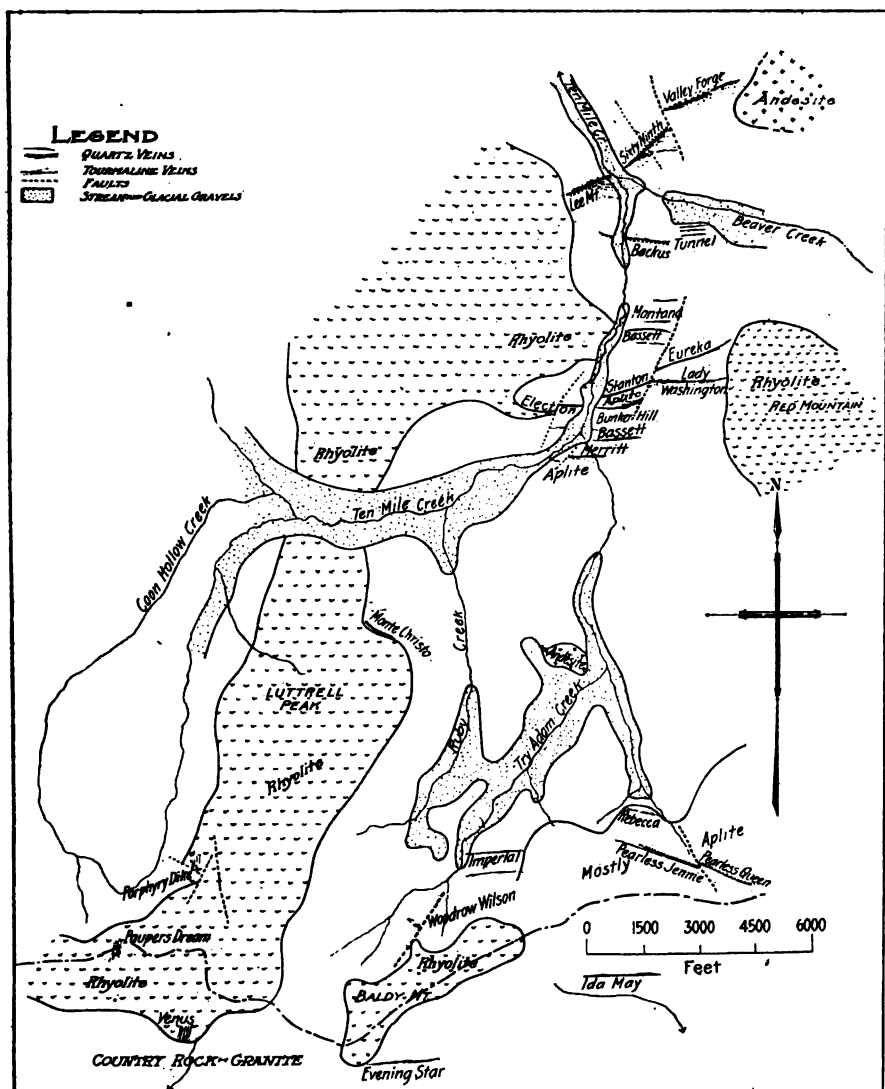


FIG. 13.—GENERAL MAP OF THE RIMINI DISTRICT, LEWIS AND CLARK COUNTY, MONTANA.

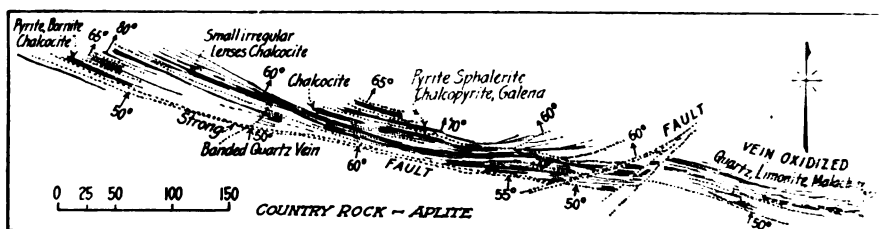


FIG. 14.—VEIN STRUCTURE CRYSTAL MINE, JACK MT. DISTRICT.

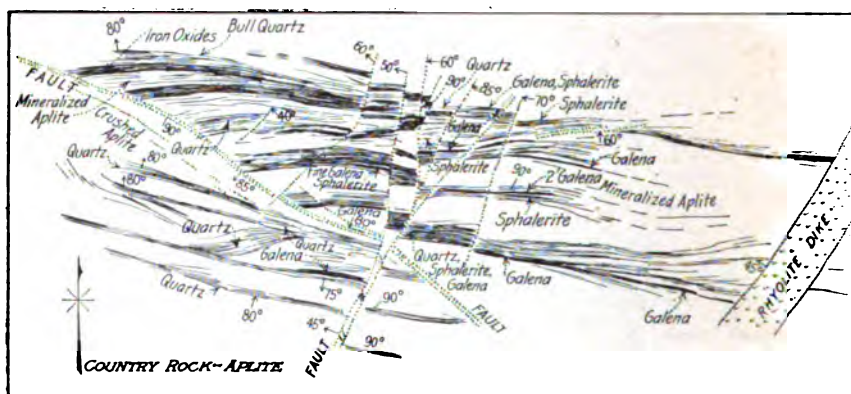


FIG. 15.—VEIN STRUCTURE, COMET MINE, BASIN DISTRICT.

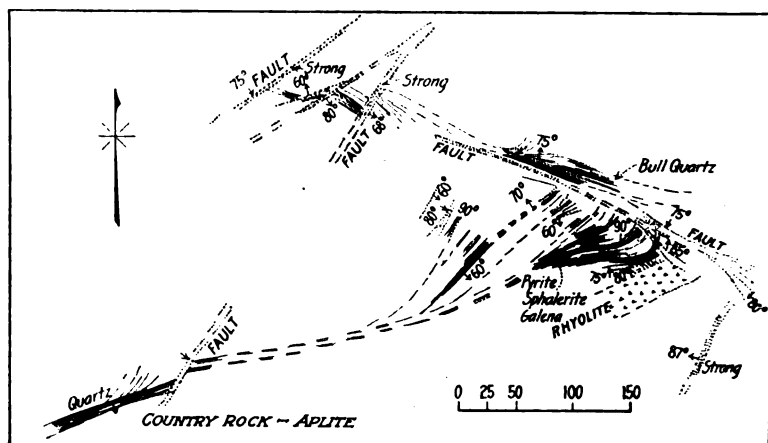


FIG. 16.—VEIN STRUCTURE, BALTIMORE MINE, BASIN DISTRICT.

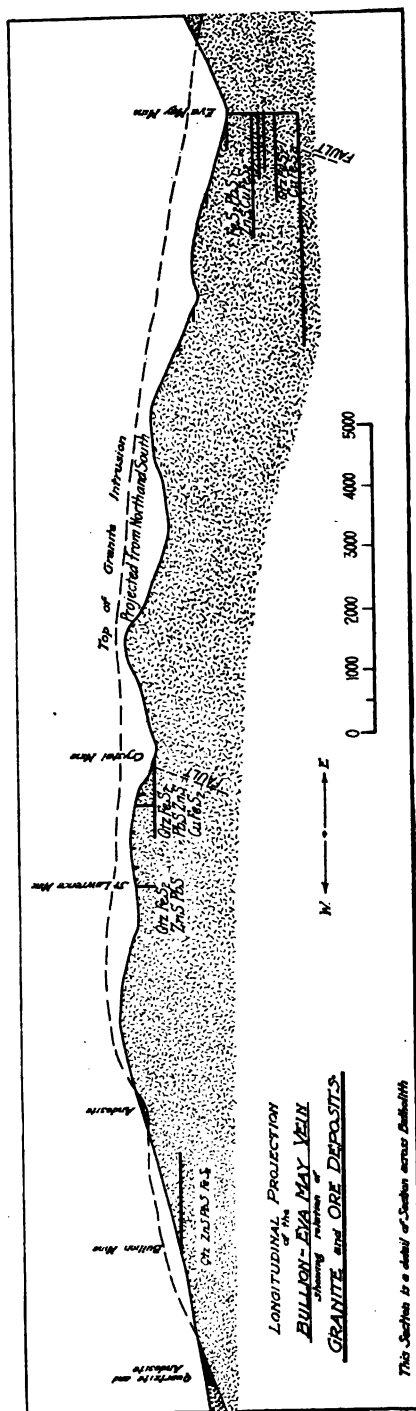


FIG. 17.—BULLION-EVA MAY VEIN WITH REFERENCE TO TOP OF GRANITE.

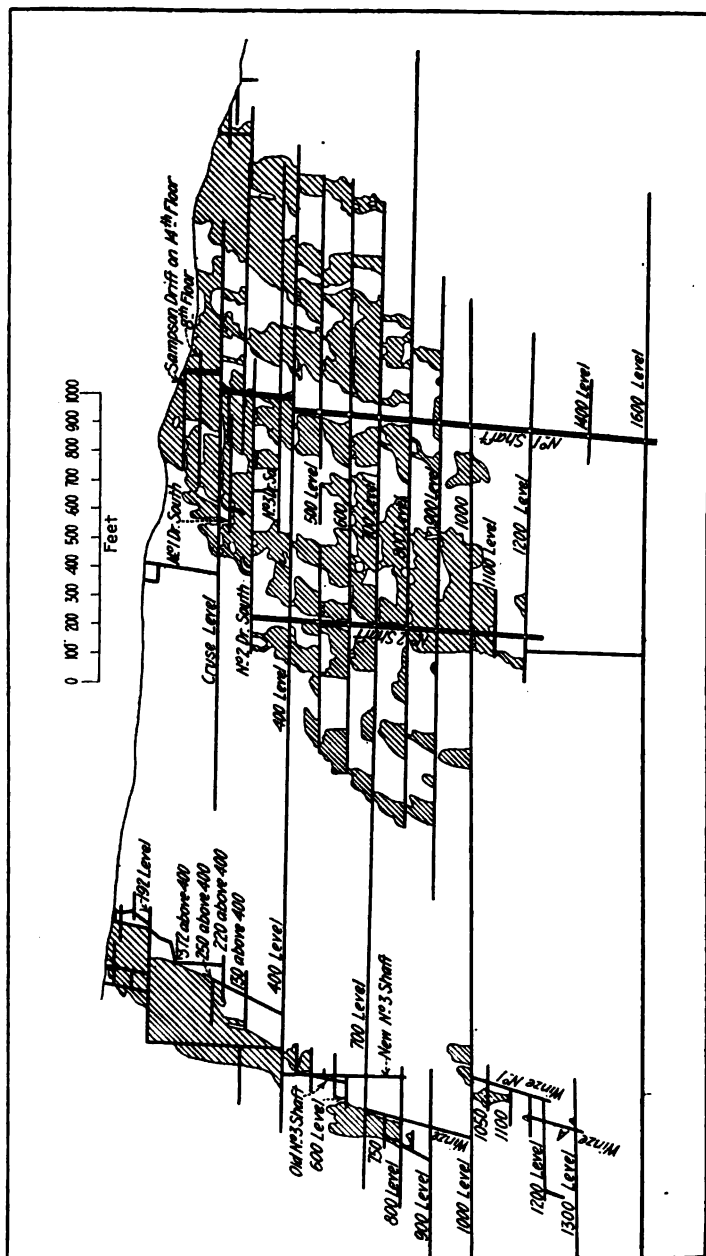


FIG. 18.—DRUMLUMMON MINE, LONGITUDINAL STOPE SECTION. (After C. W. Goodale, *Trans.*, 49, 269.)

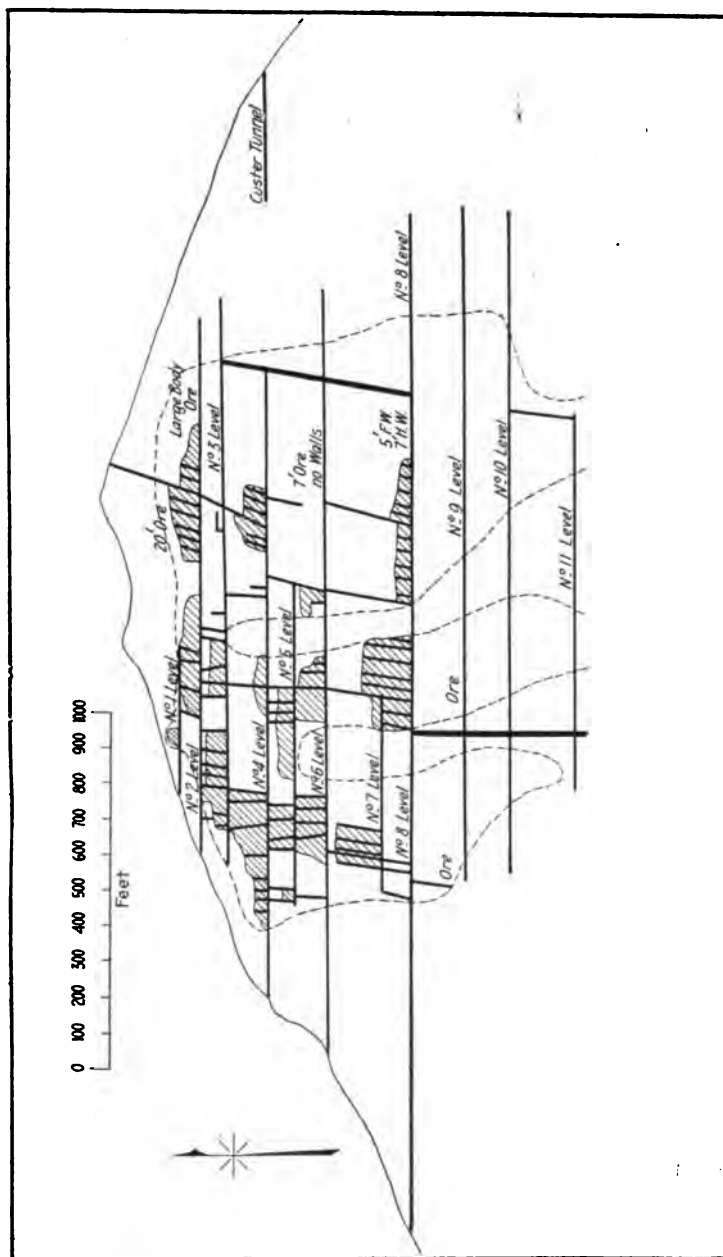


FIG. 19.—LONGITUDINAL SECTION, ALTA MINE, WICKES DISTRICT MINE.

(This section is projected on an E. W. Plane. Dotted lines show boundaries of ore shoots. These ore shoots and the lower levels are taken from a map of Oct., 1893, shortly before the mine was closed. The stopes are from a map of April, 1888. In Oct., 1893, the mine was practically worked out above the 800 level, but ore remained between the 800 and 1100 levels. A total production of 997,650 tons is recorded, and about \$2,500,000 was distributed in dividends. Later stope sections were destroyed by fire. The vein is developed 800 ft. below No. 8 level.)

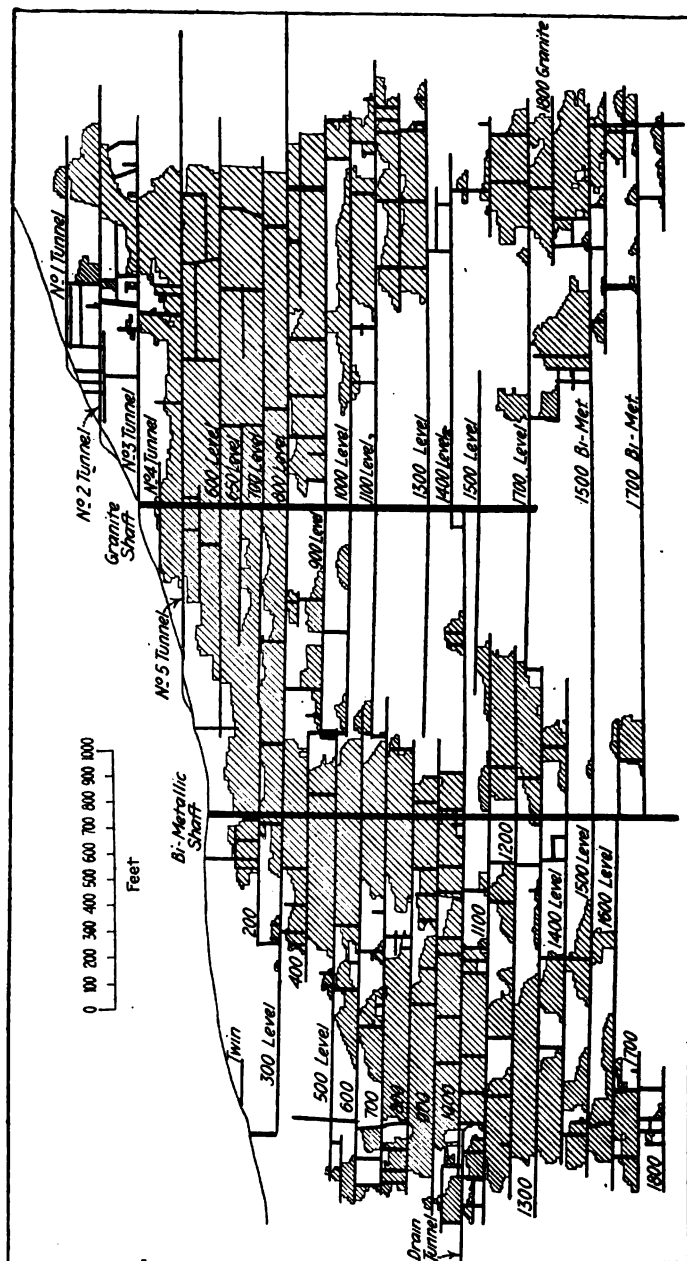


FIG. 20.—GRANITE-BI-METALLIC LONGITUDINAL STOPE SECTION.

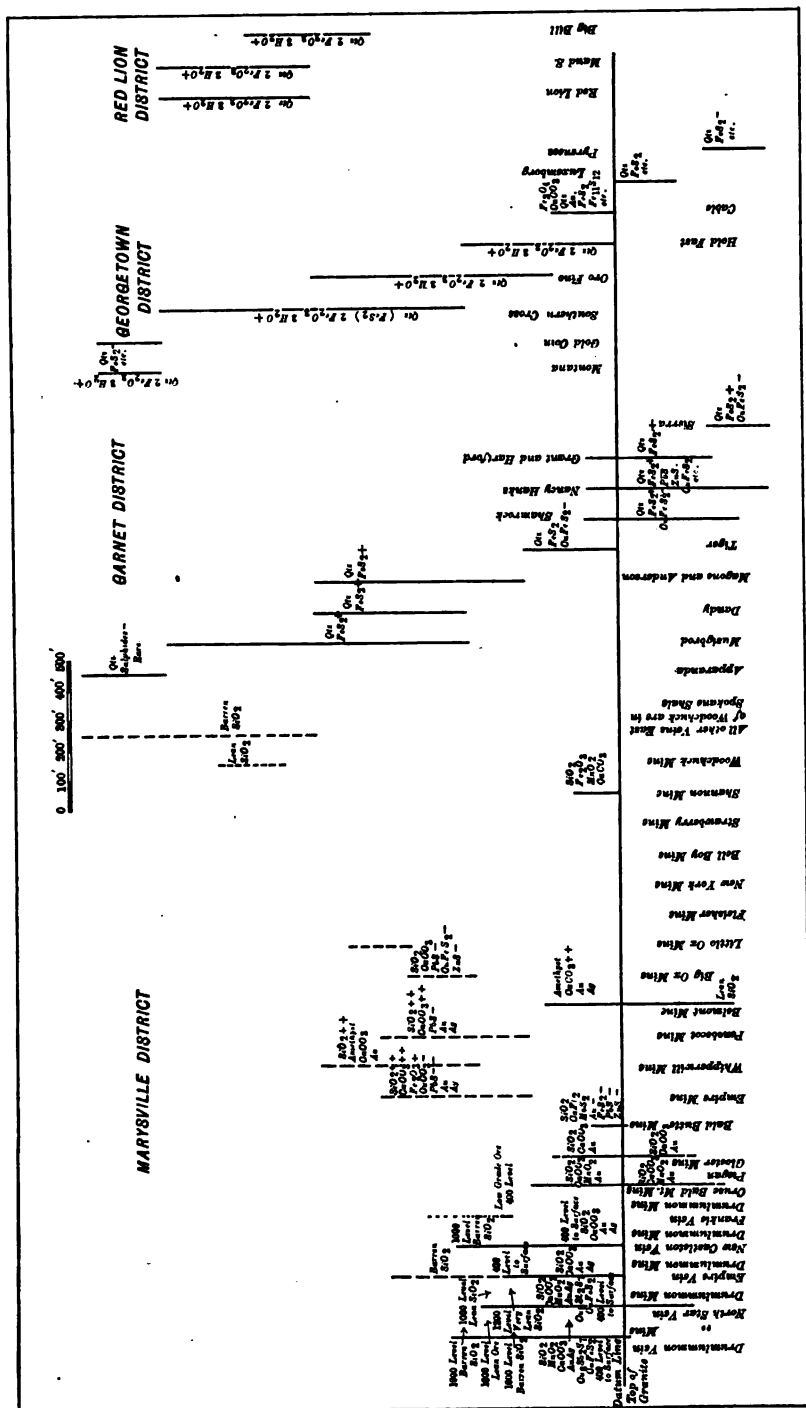


Fig. 21.—MINERAL DISTRIBUTION CHARTS OF GOLD DISTRICTS.

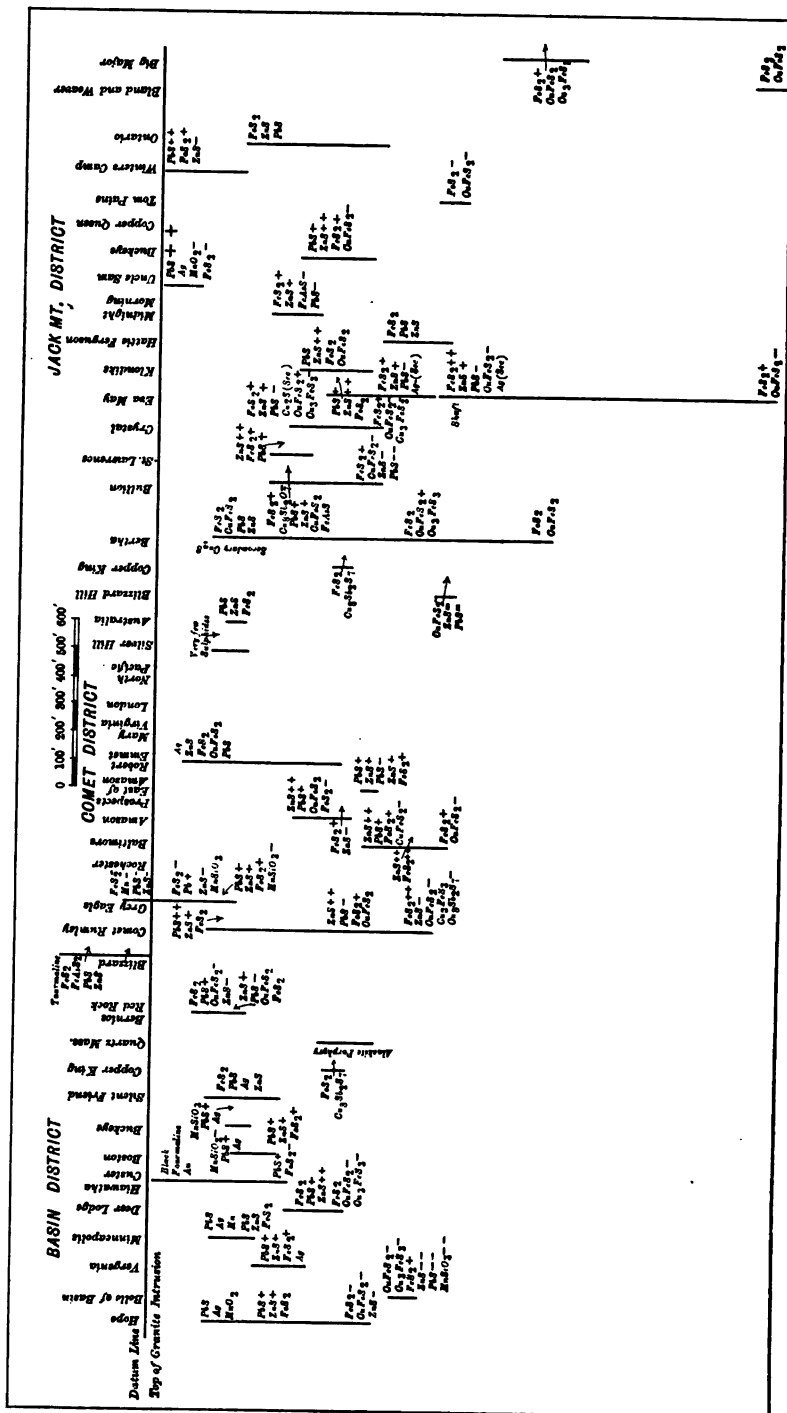


FIG. 22.—MINERAL DISTRIBUTION CHART, SILVER LEAD-ZINC DISTRICTS.

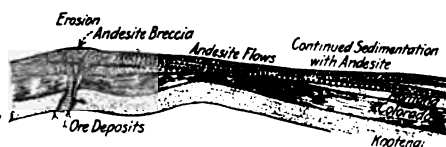
EQUILIBRIUM



FOLDING



1. MIDDLE CRETACEOUS

ANDESITE EXTRUSION
AND ORE DEPOSITION

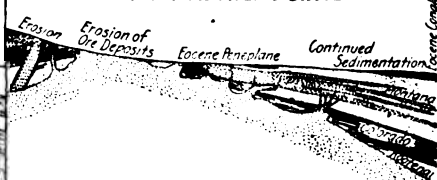
2. END OF MIDDLE CRETACEOUS

FOLDING & OVERTHRUST
FAULTING

3. UPPER CRETACEOUS

GRANITE INTRUSION & UPLIFTING
WITH ORE DEPOSITION AT HIGH
POINTS OF INTRUSION

4. UPPER CRETACEOUS

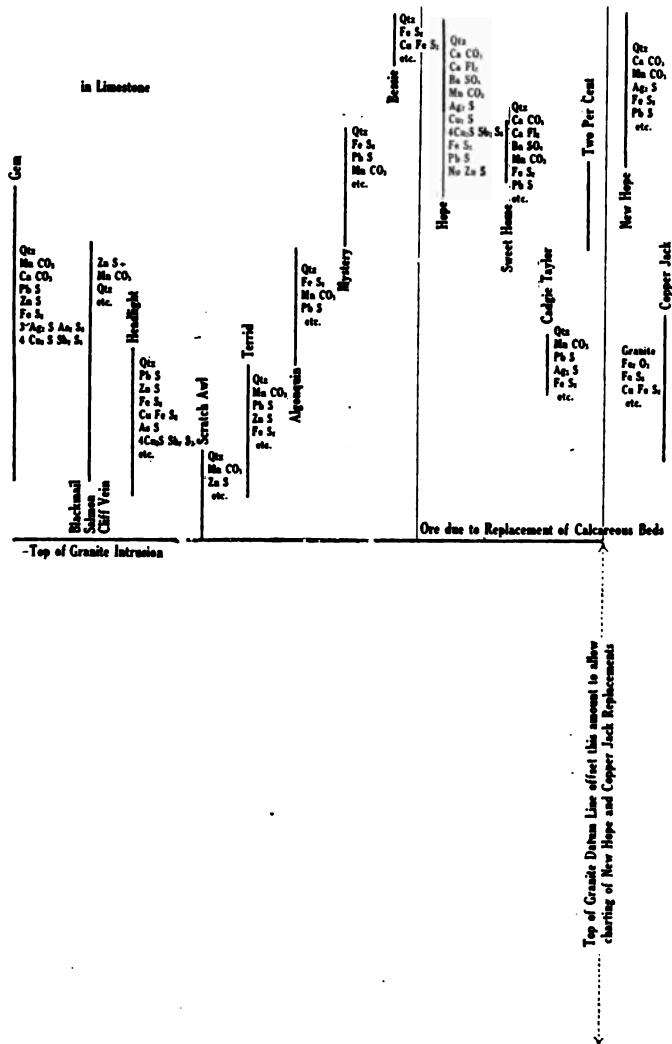
EROSION EXPOSING GRANITE
INTRUSION AT HIGH POINTS

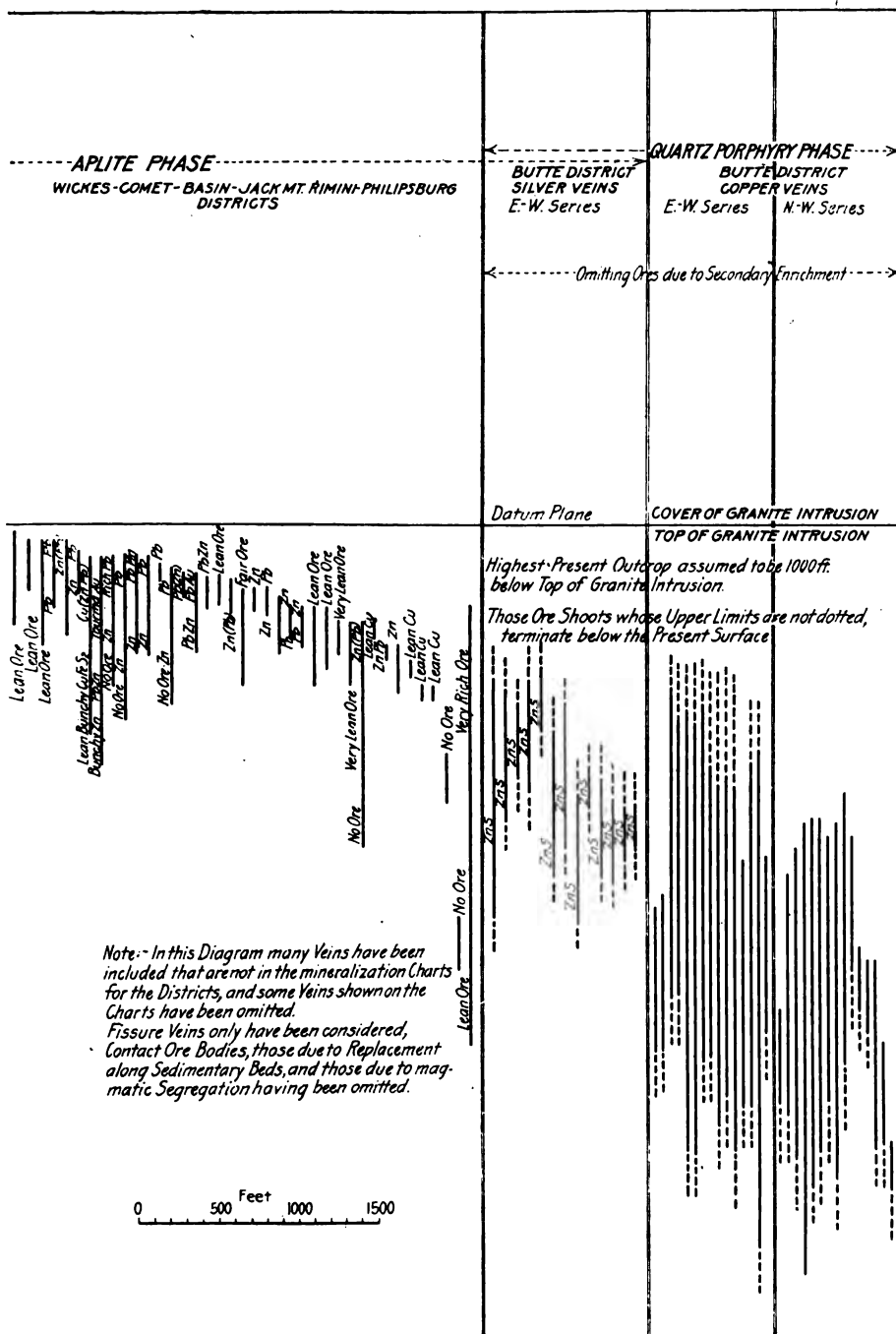
5. END OF CRETACEOUS

6. EOCENE

FIG. 23.—GRAPHIC REPRESENTATION OF GEOLOGIC HISTORY UP TO GRANITE STAGE ORE DEPOSITION.

(Idealized to show age relationship of igneous rocks and cretaceous period. Most of the ore deposits are in rocks of other ages.)





SURE VEINS, GRANITE PERIOD OF VEIN FORMATION.
(for mineralization of Butte veins.)



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Tests on the Hardinge Conical Mill

BY ARTHUR F. TAGGART,* NEW HAVEN, CONN.

(St. Louis Meeting, October, 1917)

INTRODUCTION

THE major portion of the work described in this paper was performed by R. W. Young,† a graduate student in the department of Mining and Metallurgy, Sheffield Scientific School, Yale University, working under a coöperative agreement between the Hardinge Conical Mill Co., the Sheffield Scientific School, and himself.

Since this coöperative scheme is at present in effect in the case of a considerable number of other students in the department and since it is the hope of the school that the privilege thus extended may be utilized even more freely in the future by mining and manufacturing companies, it may not be amiss at this point to give a summary of the general plan. Briefly it is as follows:

A graduate student, whose undergraduate work in this or other universities shows promise of ability to handle research work, is chosen by conference between the company and instructor involved. The aim of the company in the agreement then entered into is to obtain the solution of one or more of the technical problems with which it may be confronted, or, at the end of 1 or 2 years, to obtain as an employee a man especially trained in its work. As a means to accomplish one or both of these ends, the company furnishes the machine, apparatus, or material to be tested and pays the student during his graduate work a small salary, usually just sufficient to cover his living expenses, tuition and fees. The aim of the student is special training along a line in which he is particularly interested, the attainment of his advanced degree, and the chance to show to his future employer ability to handle such problems as may be presented to him. In return for the financial aid which he receives he agrees to devote at least half of his working time to the special problem submitted by his company. The other half is devoted to study of the collateral subjects required by the department for the granting of the degree which the student seeks. The student further agrees to enter the employ of the company in question at a wage not greater than that paid in like positions to recent graduates not specially trained and to remain with his employer at such wage for at least 1 year. If the student is to obtain a degree, the special work forming the basis of his investigation must be such as will involve real research and not mere routine manipulation. The subject is chosen by conference between the three parties to the agreement. The work is carried on under the direct supervision of the instructor involved. The school furnishes the general laboratory and library equipment essential to the pursuit of any extended investigation. In

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† Deceased, June, 1916.

return it is expected that the results of the investigation shall be, in part at least, available for publication, if they are deemed of interest to the profession.

MILL USED

In the laboratory work described in the following pages, a 4½-ft. Hardinge mill with three removable cylindrical sections, 16 in. each in length, was used. Fig. 1 shows the mill with three cylindrical rings in place. This combination allows a mill 4½ ft. by 0 in., 4½ ft. by 16 in., 4½ ft. by 32 in., or 4½ ft. by 48 in., as desired. The conical and cylindrical sections were built of cast iron, 1¼ in. thick and were lined with chrome-



FIG. 1.—HARDINGE MILL USED IN GRINDING TESTS.

steel lifting bars 2½ in. high, 3 in. wide and 16 in. long, set on 11-in. centers. The head bearing was adjustable in height, thus allowing the mill to be tilted any desired amount.

MATERIAL

A majority of the tests were made on quartzite and trap. The quartzite contained an appreciable amount of white mica in flakes 1 to 2 mm. (0.04 to 0.08 in.) diameter, which made it rather easy to crush in the coarser sizes but difficult to grind when the finer sizes were reached. The trap was a variety of diabase quarried locally for road metal. The other materials tested (see tests 230 to 236) were of a special nature and will be described more particularly in connection with the record of the work done upon them.

METHOD OF FEEDING

The material to be ground in any given test was weighed up and divided into lots, each lot being sufficient to furnish the feed for the mill for 5 min. In general, this lot was divided by eye into five portions, so that a portion could be dumped into the feed box every minute. This procedure, aided by the low capacity of the scoop feeder on coarse material, assured a practically uniform feed rate. When, as was the case with finer materials, the scoop tended to take up the material meant for a 1 min. portion in two or three revolutions, the method of feeding was so changed as to take away from the scoop the burden of regulating the feed for even such a short interval as 1 min. The importance of this insistence on regular feed will be seen in Fig. 10, which presents a comparison of the feed and discharge rates of the mill, dry crushing. In wet feeding the same methods of introducing the rock were followed. The water was introduced into the feed box from a calibrated orifice at the proper rate to give the desired moisture percentage and the result was checked by moisture samples of the discharge.

SAMPLING

Feed samples were taken, in every case, by the method of alternate shovels. Large samples were cut to insure accuracy. Samples of the product consisted of the whole discharge stream caught for varying intervals according to the feed rate. The interval for wet samples was rarely less than 1 min. For dry samples the interval was never less than 1 min. and in all cases where the feed rate was less than 1 ton per hour the sample of the product consisted of the whole discharge for an interval of 5 min.

SCREEN TESTING

Screen tests on feed samples were made in duplicate. The accuracy of the sampling was accepted as sufficient when cumulative graphs of the tests were closely coincident. Product samples were passed over the 6.680-mm. (0.26-in.) sieve. The total oversize on this sieve was then run through the coarser series. The undersize of the 6.680-mm. sieve was riffled down to not less than 200 grams and then run through the balance of the Tyler Standard Sieve Scale series of screens (1.414 ratio). The amounts of the aliquot parts of the whole sample remaining on these fine sieves and passing the last (0.074 mm.) were then calculated back into terms of the whole sample and the percentages given in Table 2 were calculated from the figures thus determined. Duplicate samples of the riffled undersize were run occasionally in order that frequent screen tests might not breed carelessness. In no case was the difference between duplicates greater than that to be expected in grading analyses.

MOISTURE DETERMINATION

Moisture determinations were made on products only, as the feed was, in every case, so dry as to be dusty. In all cases the weight of the solid plus water in the sample and the weight of the dry solid were determined by direct weighing and the percentage of moisture calculated from these figures.

POWER MEASUREMENTS

Belt drive was used for the testing work. The power transmission is not so efficient, of course, as direct drive through herring-bone gears and the latter installations will give higher relative mechanical efficiencies than those recorded in this paper. The watt-hour meter and the voltmeter and ammeter were read at 5-min. intervals. The watt-hour meter readings are the basis for the figures of power consumption used, the readings of the indicating instruments being used for purposes of check only.

OUTLINE OF TESTING WORK

Objects

The specific object of the work described in the following pages was the determination of a set of constants and characteristic curves for the conical mill which could be applied to any installation. The 4½-ft. mill is large enough to do any class of work for which the conical mill is suited, its only limitation being a question of capacity. It was hoped to cover the question of variation in capacity due to variation in diameter by a few tests on mills of other sizes. It has, however, been impossible to do this, and the writer can offer but a tentative rule based on figures collected by correspondence.

Plan of Work

The plan of the work was to start with some given set of conditions, for instance, a 16-in. cylindrical section, 4,006-lb. load of mixed balls, a trap rock feed of a given size, no moisture, mill level (Test 202); and, keeping these conditions constant, vary one other condition, (Tests 203 and 204) in this case the feed rate, and determine the effect of this variation on the character of the product, the horsepower, and the relative mechanical efficiency. By varying in similar manner the size of the feed, the kind of rock fed, the percentage of moisture, length of cylindrical section, slope of the mill, and the character and weight of the crushing charge, the effects of such variations on the performance of the 4½-ft. mill were determined.

In the collection of the aforementioned data various attendant phenomena of considerable interest were observed. Thus the distribution in

the mill of the various sizes of balls composing a mixed charge was accurately determined, the effects of slope and moisture percentage on the possible maximum crushing charge were observed, and the lack of agreement between the feed and discharge rates over any short interval (5 min. for example), resulting in practice in a pulsation in the flow to subsequent machines in a mill flow sheet, was studied. These results are presented in their proper places later. A complete series of power tests, totaling more than 100, was made to afford a basis for a formula giving the horsepower required by a conical mill of any size.

POWER TESTS

Description of Tests

Six series of tests were made to determine variations in power consumption with varying conditions of loading, one series for each of the following conditions:

Test Series No.	Length of Cylindrical Section, Inches	Condition of Pulp
1	16	Dry
2	16	Wet
3	32	Dry
4	32	Wet
5	48	Dry
6	48	Wet

In each series the first set of power readings was made with the mill empty. Successive sets of readings were then taken with ball loads starting at 500 lb. (226.8 kg.) and increasing by 500-lb. steps. With each 500 lb. of balls, 170 lb. of trap rock was charged in order to prevent excessive wear and hammer in the mill. In the dry tests loading was continued until the surface of the load was considerably above the axis of the mill, discharge being prevented by plugging the discharge end. In the wet tests enough water was fed to produce a slight discharge throughout the series, and the tests were discontinued when discharge of balls commenced. Rock was fed in these latter tests from time to time to balance the rock carried off in the discharge, but no exact balance was attempted and the degree of balance attained is not known. The duration of the tests for each condition of loading varied from 30 to 90 min. Power readings were taken every 5 min. and the run was continued until the power-time curve became a horizontal line. The early readings in any given test were considerably higher than the last due probably to cold bearings, slipping belts, etc., and were disregarded in making up the average power consumption for the run.

Fig. 2 is a graph covering all the power tests. The greatest variations from smooth curves occur near the end of the graphs for the 4½-ft. by 48-in. mill. These variations are due, in part at least, to an overloaded motor. Fig. 3 was plotted in an attempt to draw the curves for the different tests closely enough together to give a reasonable basis for an average curve upon which it would be possible to base an empirical form-

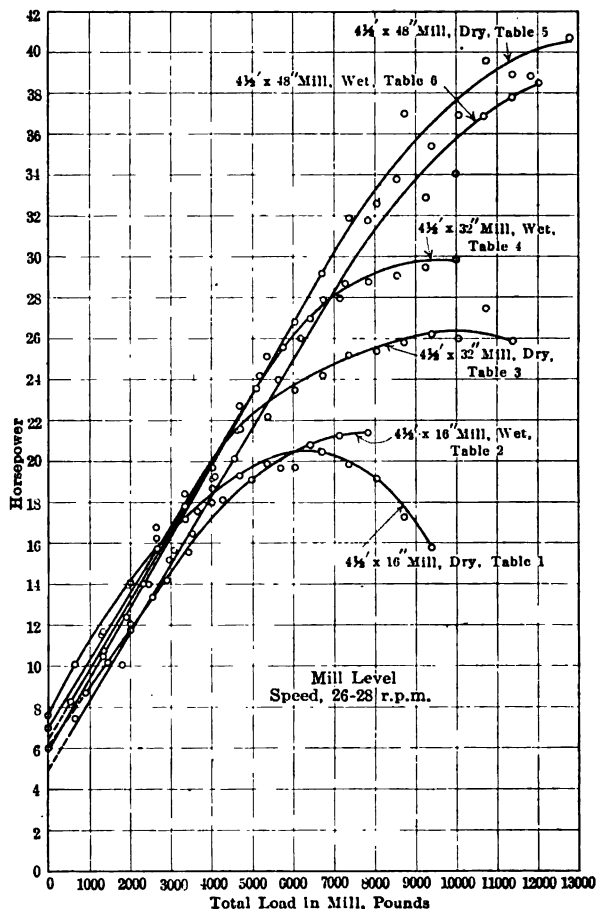


FIG. 2.—POWER CONSUMPTION OF 4½-FT. CONICAL BALL MILL AT HAMMOND LABORATORY.

ula for horsepower. As will be seen by reference to this figure, the curves are closely parallel throughout their respective lengths, the only graph departing seriously from parallelism with the others being that for the 4½-ft. by 16-in. mill, dry, series No. 1. It will be noted that the variation of this curve begins at the point where the charge in the mill rose above the horizontal axis. The mill was here working under unnatural

conditions, so that this variation will not affect a formula designed to cover working ranges only.

From this point two methods of procedure were followed, resulting in the following formulæ for the horsepower of a conical ball mill within working ranges.

$$(1) Hp. = 0.002(100D^2L^{0.132} + L) \quad (1)$$

$$(2) Hp. = \frac{D^{2.42}L^{0.086}}{6.53} + \frac{L}{1,000}(0.025m + 1.4) \quad (2)$$

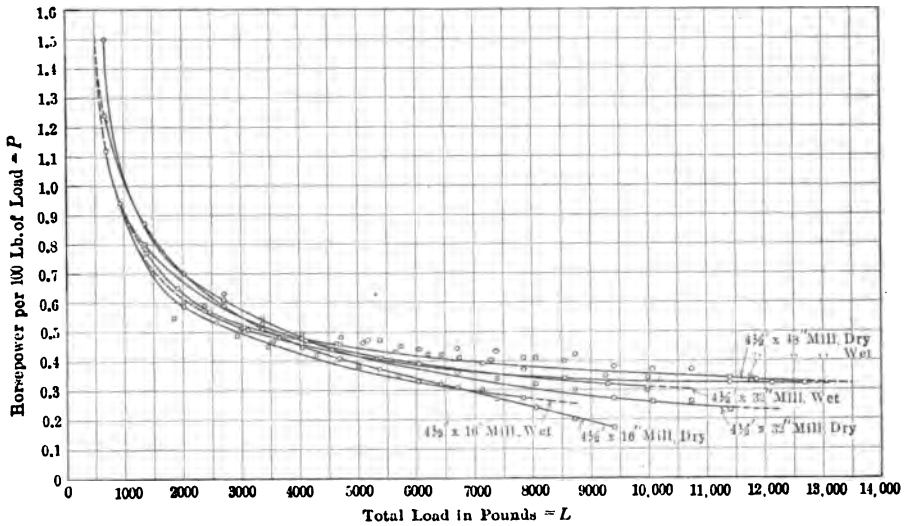


FIG. 3.--POWER CONSUMPTION PER 100 LB. OF BALL LOAD BY 4 1/2-FT. CONICAL BALL MILL WITH DIFFERENT CYLINDER LENGTHS.

where

D = the internal diameter of the mill in feet,

L = the total load in the mill in pounds,

m = the length of the cylindrical section in inches.

The first of these formulæ is of the nature of a preliminary trial and was developed from a free-hand average curve drawn through the curves on Fig. 3. It does not, therefore, give results which check throughout the range of operating conditions. Formula (2) was developed as outlined in the succeeding paragraph.

The first step in the determination of Formula (2) was to plot the average curve shown in Fig. 4 from the curves on Fig. 3. The points determining this curve were obtained by averaging the ordinates of the curves on Fig. 3 at the abscissæ 1,000, 2,000, etc. Arbitrary ordinates y and abscissæ x were then assigned to this curve and various functions

such as x^2 , y^2 , $x^{1/2}$, $y^{1/2}$, $\log x$, $\log y$, $\frac{x}{y}$, $\frac{y}{x}$, $\frac{1}{x}$, $\frac{1}{y}$, etc., were plotted against each other in different combinations in an attempt to straighten out the curve. The best approximation to a straight line was obtained by plotting $\log (y - 2)$ as ordinates and $\log (x)$ as abscissæ. The points thus obtained are shown on Fig. 5. The straight line drawn through these points was obtained by averaging ordinates and abscissæ. The equation for this line is:

$$\log (y - 2) = 1.213 - 0.914 \log (x) \quad (4)$$

From (4)

$$\log (y - 2)(x^{0.914}) = 1.213 \quad (5)$$

Taking antilogarithms of both sides

$$(y - 2)(x^{0.914}) = 16.33 \quad (6)$$

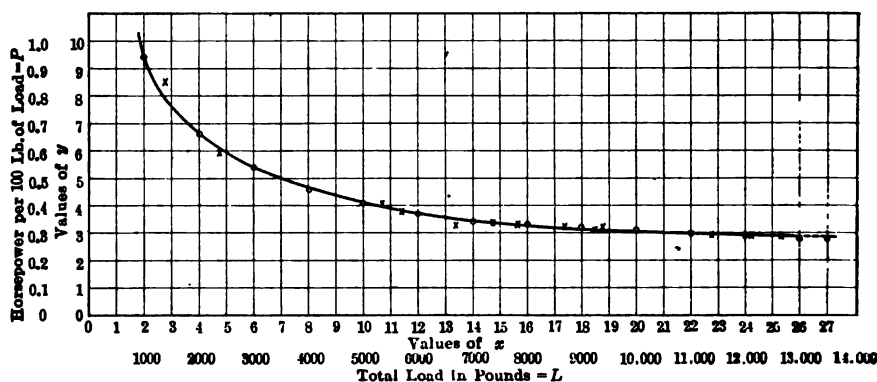


FIG. 4.—AVERAGE POWER CONSUMPTION OF 4½-FT. CONICAL BALL MILL.

Clearing

$$y = \frac{16.33}{x^{0.914}} + 2 \quad (7)$$

But from Fig. 4

$$y = 10P \quad (8)$$

where P = hp. per 100 lb. of load

$$x = \frac{L}{500} \quad (9)$$

Substituting these values for x and y in equation (7) and clearing

$$P = \frac{478}{L^{0.914}} + 0.2 \quad (10)$$

But

$$Hp. = \frac{LP}{100} \quad (11)$$

Then

$$Hp. = 4.78L^{0.086} + 0.002L \quad (12)$$

Horsepowers solved for by this formula for different loads are indicated by crosses (x) on Fig. 4. These check solutions show a close agreement with the average curve, but show in some cases as much as 30 per cent. departure from the horsepowers determined experimentally. The variations are, as might be expected, greatest for the 16-in. and 48-in. cylindrical sections, as the average curve of Fig. 4 departs most greatly from the curves for these cylinder lengths. In order to eliminate this variation, Formula (12) was written as follows:

$$Hp. = 4.78L^{0.086} + CL \quad (13)$$

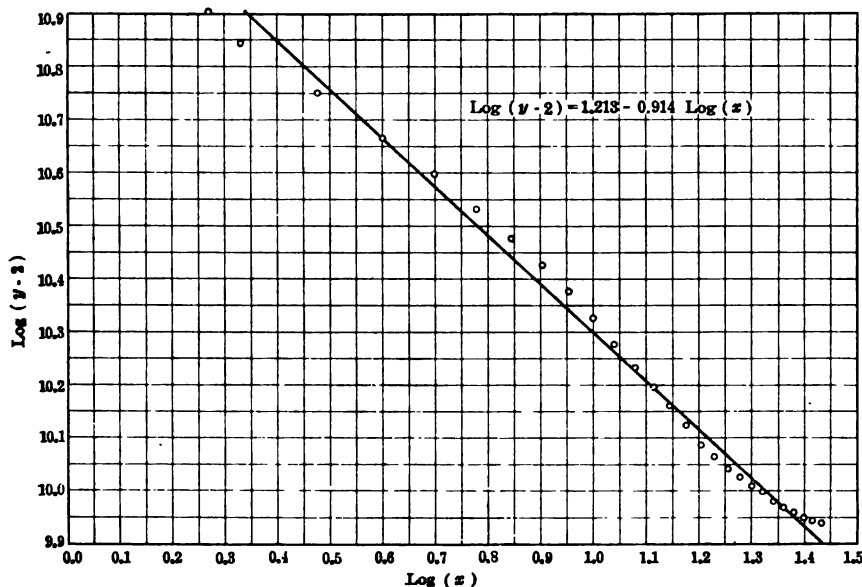


FIG. 5.

and average values of C were determined for different cylinder lengths by substituting known values of $Hp.$ and L corresponding to values of m from results of tests, series 1 to 6 inclusive. By this method the following corresponding average values of C and m were determined:

m	C
16	0.0018
32	0.0022
48	0.0027

The relation between these quantities can be expressed in the linear form

$$C = 0.000025m + 0.0014 \quad (14)$$

from which equation (13) may be rewritten as

$$Hp. = 4.78L^{0.086} + \frac{L}{1000} (0.025m + 1.4) \quad [15]$$

which gives values for the horsepower of the 4½-ft. ball mill that are accurate within a few per cent. The average curve for horsepower for mills of other diameters plotted with horsepower per 100 lb. (45.36 kg.) of load as ordinates and total load as abscissæ will be similar to the curve

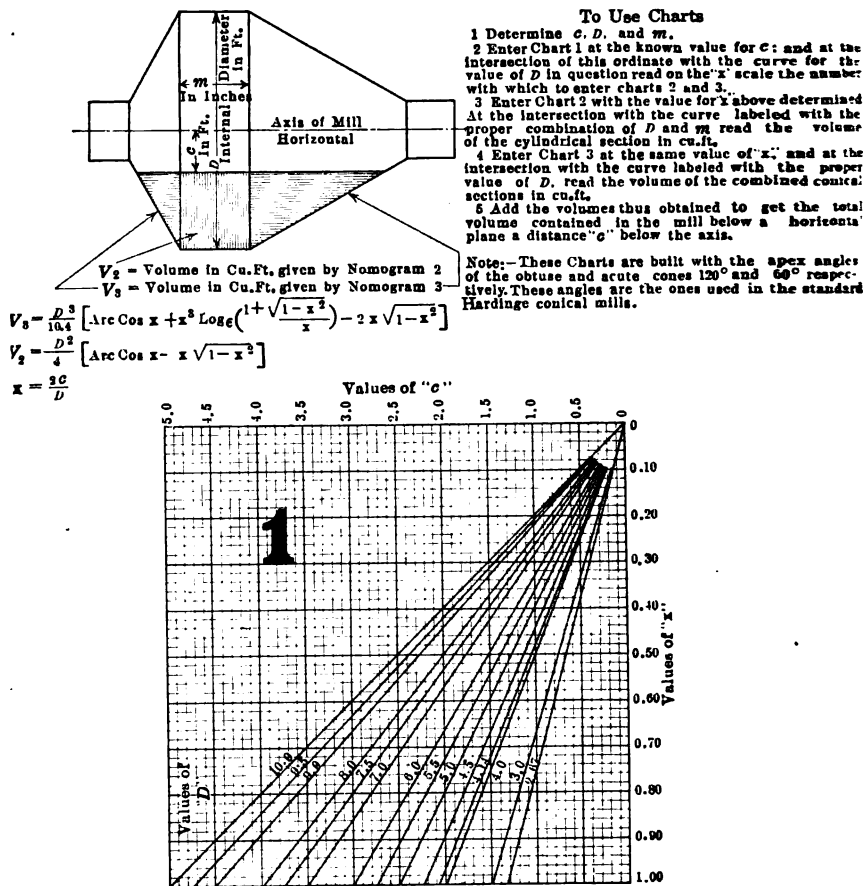


FIG. 6.

in Fig. 4 and, by changing the scale, can be made to coincide with this curve. If, then, corresponding values of x and L on this figure can be established for mills of several diameters and the proper substitutions made in equation (7) we will get a series of different numbers for the coefficient of the term $L^{0.086}$ in equation (15), corresponding to different internal diameters. The values of L corresponding to a given value of

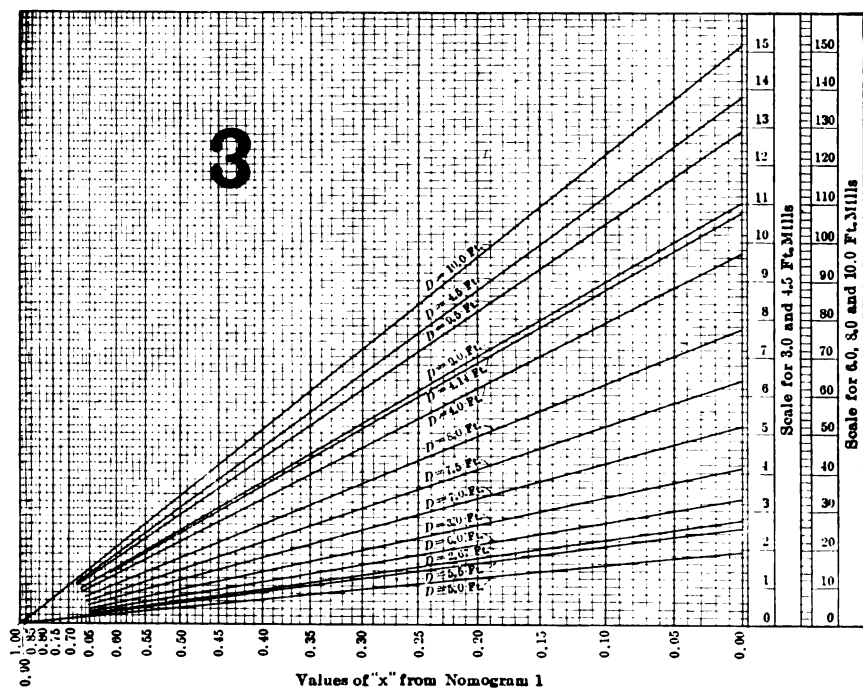
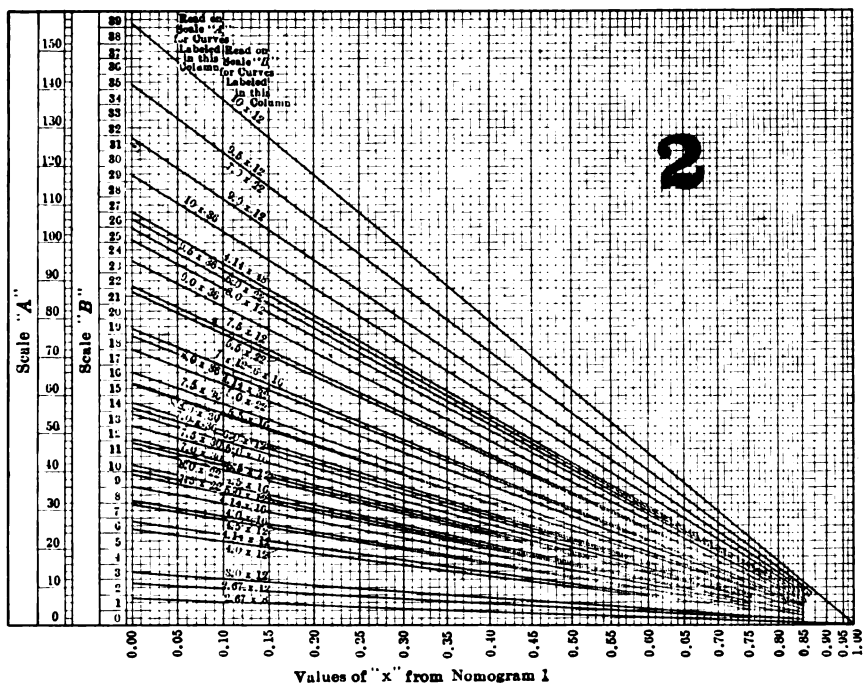


FIG. 6.

x , Fig. 4, will vary according to the volumes of the different mills. These values and the corresponding values of the coefficients of $L^{0.086}$ are as follows:

Internal Diameter of Mill in Feet	x	L	Coefficient of $L^{0.086}$
4.14	1	500	4.78
5.50	1	1,035	9.30
7.50	1	2,355	19.72
9.50	1	4,460	35.35

The logarithms of these coefficients of $L^{0.086}$ and the logarithms of the internal diameter of the mill, D , bear a linear relation to each other which is expressed in the equation

$$\log (D) = 0.413 \log (C) + 0.337 \quad (16)$$

From this equation

$$C = \frac{D^{2.42}}{6.53} = \text{coefficient of } L^{0.086} \quad (17)$$

Substituting this value for the coefficient of $L^{0.086}$ in equation (15) we have

$$Hp. = \frac{D^{2.42} L^{0.086}}{6.53} + \frac{L}{1,000} (0.025m + 1.4) \quad (18)$$

This formula gives values accurate within a few per cent. for the horsepower of the conical ball mill throughout the range of operating conditions.

For pebble mills with smooth lining, results obtained by the above formula should be multiplied by the factor 0.65; with a semi-smooth lining, 0.8; with a rough lining, 0.95. It must be noted, however, in the use of the formula, that the load should be calculated on the assumption that the mill is horizontal, as the reduction in load due to tilting does not produce a corresponding decrease in power consumption.

The charts given in Figs. 6 and 7 will be found useful in determining the value of L in the horsepower formula. The use of these charts may be best explained by following through a calculation for the horsepower consumed by an 8-ft. by 30-in. ball mill crushing rock of a specific gravity of 2.6 with a moisture content of 50 per cent., using a 30,000-lb. ball load, composed of 5-in., 4-in., and 3-in. balls. For this condition

$$D = 7.5.$$

c (Fig. 6) = 0. (The volume contained in a mill in operation is more than that contained in the same mill at rest, and the assumption that $c = 0$ is legitimate.)

Then from nomogram 1,

$$x = 0.$$

Enter nomogram 3 with this value for x and read on line $D = 7.5$, $V = 63.7$.

Enter nomogram 2 with $x = 0$ and read on line 7.5×30 , $V = 55.3$.

The working volume of the mill is, then, $63.7 + 55.3 = 119.0$ cu. ft.

The volume occupied by the balls is $30,000 \div 495 = 60.6$ cu. ft.

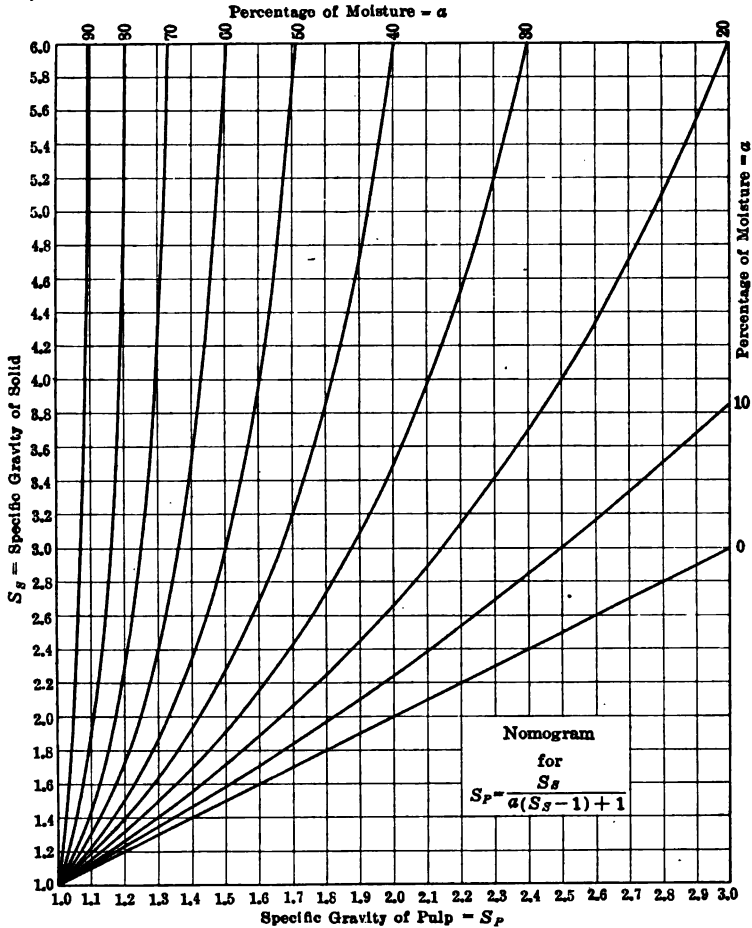


FIG. 7.

The volume occupied by the pulp is $119.0 - 60.6 = 58.4$ cu. ft.

The specific gravity of the pulp is determined from Fig. 7.

Enter at $S_S = 2.6$. At the intersection with the curve $a = 50$ per cent., read $S_P = 1.44$. The weight of the pulp in the mill is, then, $58.4(1.44)62.5 = 5,250$ lb.

$$L = 30,000 + 5,250 = 35,250.$$

$$Hp. = \frac{(7.5^{2.42})(35,250)^{0.086}}{6.53} + \frac{35,250}{1,000} (0.025(30) + 1.4)$$

$$= 125.2.$$

The economy in calculation to be gained from the use of Fig. 6 is not so apparent in the foregoing instance, where c was taken equal to zero, as it will be if the information sought is the crushing load which fills a given mill to within a given distance of the center, or the depth to which a given load will fill a mill of a given size. In the course of 2 or 3 years' work with the mill the writer has been confronted with a consider-

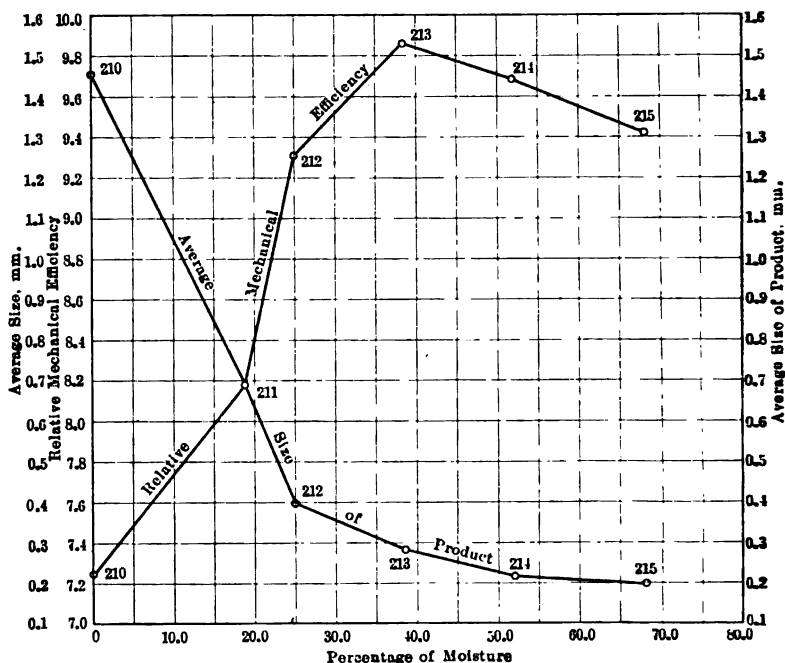


FIG. 8.—EFFECT OF MOISTURE ON CRUSHING EFFICIENCY AND AVERAGE SIZE OF PRODUCT.

able number of such problems and it is because of the saving in time effected in their solution by the use of the chart, that it is inserted here. In such calculations the weight of a cubic foot of steel balls may be taken as 250 lb. and the weight of a cubic foot of pebbles as 100 lb.

ANALYSIS OF OPERATING DATA

Thirty-five tests were run on the 4½-ft. mill to determine the effect of variations in operating conditions on the performance of the mill. As each test furnished some information that may be classified under several heads, it is not possible, without considerable repetition, to segre-

gate them. They are, therefore, presented in Tables 1 and 2 in the order in which they were performed and will be referred to by number in the subsequent discussion.

The indicator commonly used in this paper for comparing the character of the work done by the mill under any given condition with that done under some other condition is the figure in the last column of Table 1 headed R.M.E. (Relative Mechanical Efficiency). A detailed explanation of the development of this conception is given in the article,

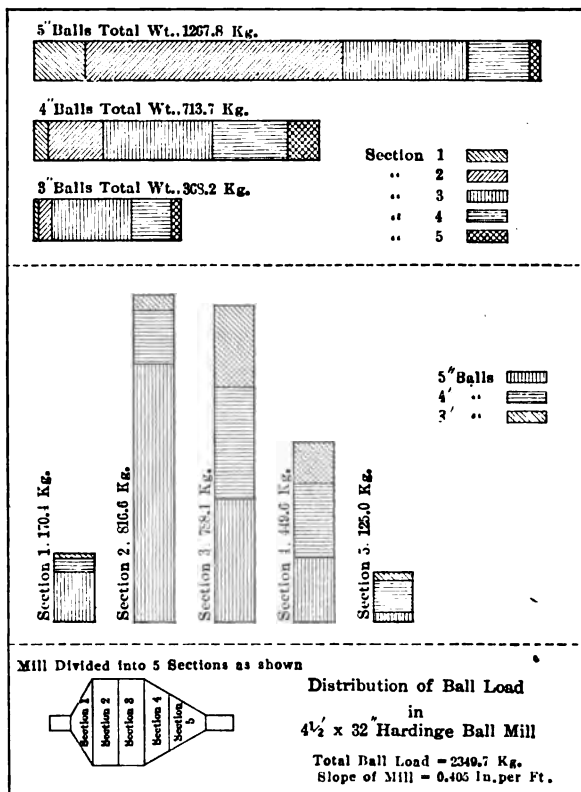


FIG. 9.

The Work of Crushing, *Trans.* (1914), **48**, 153. Briefly, it is expressed in the formula

$$\text{R.M.E} = \frac{(\text{Difference E.U. Feed and Product}) (\text{Tons per 24 hr.})}{\text{Hp.}}$$

in which the term "Difference E.U. Feed and Product" is a measure of the useful work done per unit of weight in reducing the material in question from feed size to discharge size, and is determined by screen analysis.

Thus the total energy units, E.U., in the feed sample, screen test 22-A, Table 2, is 34.32 and is obtained by summing the products of the percentages on the different screens by their corresponding ordinal numbers. The same method applied to screen test 22-41 gives 1,664.22 E.U. in the product of run No. 202. Then

$$\text{R.M.E.} = \frac{(1,664.22 - 34.32)12}{21} = 930.$$

In order to make this figure accord with commonly accepted figures of efficiency, the R.M.E. thus obtained is divided by 100, giving for test 202 a value of 9.3.

Rate of Feed

The effect of rate of feed on the relative mechanical efficiency of the conical ball mill is given in the two groups of tests 202 to 204 and 213, 216 to 219. In the first group, trap rock of an average size of 24.58 mm. (0.96 in.) was fed dry at the rates of 1,000 lb. (453.59 kg.), 1,500 lb., and 2,000 lb. per hour. The axis of the mill was horizontal and the ball load was a mixture of 5-in., 4-in., 3-in., and 1¾-in. balls in approximately the same proportions that would be found in commercial operation after the mill had settled down. The relative mechanical efficiencies, 9.3, 9.13 and 10.3 respectively indicate the result, confirmed in later tests, that the ratio of useful work done by the mill to power input increases with the feed rate. That there is, of course, a limit to this proportionate increase at the point of overload is shown in the second series of tests above mentioned. In this series the mill was tilted 2½ in., or 0.405 in. per foot, toward the discharge end. One result of this tilting was to decrease the ball capacity of the mill by about 1,200 lb. Quartzite of an average size of 9.90 mm. was fed with an average of about 38 per cent. moisture at rates of 1,500 lb., 3,000 lb., 6,000 lb., 9,000 lb. and 12,000 lb. per hour. The relative mechanical efficiencies corresponding to the above rates were 9.86, 17.30, 29.21, 43.50, and 41.10. In this series of tests the relative mechanical efficiency of the machine increases with the feed rate up to 4.5 tons per hour, beyond which we have an apparent condition of overloading. Table 3 gives the reduction in average size of particle in the different tests above discussed.

These figures present three different cases for consideration. Test 202 is in a class by itself, the machine is patently underfed for all purposes except that of producing a practically finished, fine, dry product at one passage through the machine. It is a surprising fact that in doing this kind of work the machine uses power so efficiently. Tests 203, 204, 218 and 219 compared with tests 213, 216 and 217 point the moral that for most efficient work it is not wise to attempt too great reduction at one passage through the mill. When the large amount of power consumed

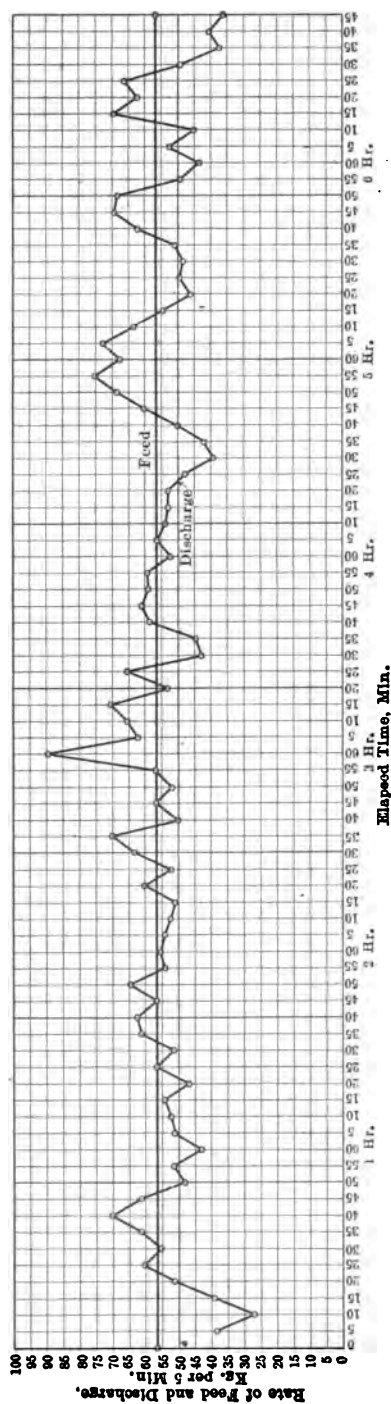


FIG. 10.—COMPARISON OF FEED AND DISCHARGE RATES OF CONICAL BALL MILL DRY CRUSHING.

TESTS ON THE HARDINGE CONICAL MILL

Test No.	Size of Mill	Charac-ter	Crushing Charge				Kind of Rock Crushed	Screen Test, No.		Feed Rate, Pounds per Hour	Percent- age of Non- sieve material	Elevation of Feed End, Inches	R.p.m.	Hp.	R.m.e.
			5-in.	4-in.	3-in.	1 3/4-in.		Feed	Product						
202	4 1/2 by 16	Balls	1,960	953	712	381	Trap	22-A	22-41	1,000	0.0	0	28.0	21.0	9.30
203	4 1/2 by 16	Balls	1,960	953	712	381	Trap	22-A	22-69	2,000	0.0	0	28.0	21.0	10.30
204	4 1/2 by 16	Balls	1,960	953	712	381	Trap	22-A	22-77	1,500	0.0	0	28.0	21.3	9.13
205	4 1/2 by 16	Balls	2,455	1,110	703	235	Trap	22-A	22-79	1,500	0.0	0	28.0	22.1	11.03
206	4 1/2 by 16	Balls	2,400	522			Trap	22-A	22-88	1,500	0.0	4 1/4	26.5	23.0	8.66
207	4 1/2 by 16	Balls	2,400	522			Trap	22-A	22-91	1,500	0.0	2 1/2	28.0	22.2	9.33
208	4 1/2 by 16	Balls	2,400	522			Trap	22-A	38-4	1,500	19.5	2 1/2	26.5	19.9	11.46
209	4 1/2 by 16	Balls	2,400	522			Quartzite	39-A	39-3	1,500	0.0	2 1/2	25.5	18.4	8.09
210	4 1/2 by 16	Balls	1,460	885	385		Quartzite	39-A	40-1	1,500	0.0	2 1/2	26.0	19.2	7.25
211	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	41A-1	1,500	18.8	2 1/2	27.0	17.7	7.18
212	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	41B-1	1,500	25.0	2 1/2	27.0	17.7	9.31
213	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	41C-1	1,500	38.5	2 1/2	27.0	17.7	9.86
214	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	41D-1	1,500	51.8	2 1/2	26.0	19.3	9.67
215	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	41E-1	1,500	68.2	2 1/2	26.0	20.1	9.42
216	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	42-1	3,000	37.4	2 1/2	26.0	18.5	17.30
217	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	42-2	6,000	37.3	2 1/2	26.0	17.7	20.21
218	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	42-3	12,000	38.3	2 1/2	26.0	18.5	41.10
219	4 1/2 by 16	Balls	1,460	878	481		Quartzite	39-A	42-4	9,000	38.9	2 1/2	26.0	16.9	43.50
220	4 1/2 by 16	Balls	1,772	1,067	711		Quartzite	39-A	44-1	1,500	31.7	1 1/4	26.5	20.1	8.45
221	4 1/2 by 16	Balls	2,125	1,284	855		Quartzite	39-A	44-2	1,500	36.5	0	28.0	20.1	9.06
222	4 1/2 by 16	Balls	2,125	1,284	855		Quartzite	45-A	45-1	1,500	35.4	0	25.0	20.9	10.20
223	4 1/2 by 16	Balls	1,457	874	480		Trap	45-A	46-1	1,500	41.7	2 1/2	26.5	20.1	10.71
224	4 1/2 by 16	Balls	696	417	203		Quartzite	39-A	47-1	1,500	47.0	2 1/2	26.5	15.3	9.83
225	4 1/2 by 16	Balls	1,412				Quartzite	39-A	47-2	1,500	40.0	2 1/2	27.0	13.7	10.72
226	4 1/2 by 32	Balls	2,785	1,572	810		Quartzite	39-A	49-1	1,500	39.4	3 1/2	25.0	26.6	7.15
227	4 1/2 by 48	Balls	3,765	1,822	1,067		Quartzite	39-A	50-1	1,500	40.4	0.405 in./ft.	24.5	33.5	5.88
228	4 1/2 by 48	Pebbles			3,125		Quartzite	51-A	51-1	1,500	42.2	0	27.5	16.9	5.37
229	4 1/2 by 48	Pebbles			3,125		Trap	52-A	52-1	1,500	37.6	0	26.5	16.4	1.80
230	4 1/2 by 48	Balls	1,604	1,479	2,160	70	Cocoonut shell	32-1	32-2	1,000	0.0	2 1/2	25.0	22.1	1.67
231	4 1/2 by 48	Balls	343	1,480	2,720	758	Cocoonut shell	32-2a	32-3	550	0.0	2 1/2	28.0	23.6	0.60
232	4 1/2 by 48	Balls	343	1,480	2,720	500	Cocoonut shell	32-4	32-7	550	84.5	0	28.0	31.4	1.66
233	4 1/2 by 48	Balls	343	1,480	2,720	500	Cocoonut shell	32-6	32-8	2,200	64.8	0	27.0	32.8	0.04
234	4 1/2 by 16	Balls		1,466	2,536		Newturt	5,321	5,331	300	0.0	0	27.0	15.8	0.48
235	4 1/2 by 16	Balls	1,027	951	815	720	Brass rollers	5,321	5,331	4,400	42.9	2 1/2	28.0	20.0	16.60
236	4 1/2 by 16	Balls	1,027	951	815	720	Brass rollers	5,321	5,331	1,850	36.7	2 1/2	28.0	21.0	10.30

TABLE 2.—*Screen Tests*

Ordinal Number	Screen Aperture, Millimeters	22-A	22-41	22-69	22-77	22-79	22-88	22-91
-1.0	38.100	2.15						
0	26.670	70.90		11.72	7.41	6.67	5.05	1.13
1	18.850	24.42		16.84	14.78	5.78	6.10	3.87
2	13.330	1.63		11.62	9.27	2.28	5.95	5.16
3	9.423	0.23		4.60	3.84	0.83	4.98	5.44
4	6.680	0.10		2.46	1.42	0.46	3.60	4.99
5	4.699	0.05		1.21	0.49	0.32	3.28	5.44
6	3.327	0.04		0.50	0.38	0.50	2.60	4.48
7	2.362	0.02	0.17	0.36	0.29	0.32	2.62	3.00
8	1.651	0.02	0.85	0.28	0.37	0.50	2.54	3.09
9	1.168	0.02	0.48	0.37	0.40	0.86	2.72	2.71
10	0.833	0.02	0.92	0.46	0.52	1.47	2.58	2.35
11	0.589	0.02	2.49	0.79	1.10	2.92	3.02	2.50
12	0.417	0.02	3.57	1.21	1.39	3.71	2.82	2.34
13	0.295	0.03	6.10	2.20	2.74	5.15	3.24	2.95
14	0.208	0.04	5.98	2.97	3.63	5.88	3.50	3.17
15	0.147	0.04	9.05	4.78	5.50	6.94	5.04	5.52
16	0.104	0.05	11.75	6.56	7.55	9.39	6.41	6.49
17	0.074	0.04	8.34	4.92	6.12	7.47	5.60	5.75
19	Through 0.074	0.16	50.30	26.15	32.80	38.55	28.35	29.62
	Aver. size of particle, mm.....	24.58	0.142	8.590	6.573	3.449	4.472	3.219
	Total energy units	34.32	1,664.22	936.00	1,114.80	1,387.64	1,138.47	1,183.64

TABLE 2.—*Screen Tests.*—(Continued)

Ordinal Number	Screen Aperture, Millimeters	38-4	39-A	39-3	40-1	41A-1	41B-1	41C-1
-1.0	38.100		0.66					
0	26.670	4.34	5.51	0.22	0.45			
1	18.850	3.22	16.36	0.76	1.44	0.03		
2	13.330	2.98	22.00	1.18	2.02	0.25		
3	9.423	2.32	11.18	1.00	1.63	0.45		0.01
4	6.680	2.30	7.46	1.16	1.37	0.72	0.11	0.02
5	4.699	1.82	5.49	1.38	1.83	0.70	0.28	
6	3.327	1.72	3.62	1.54	2.44	1.42	0.58	0.14
7	2.362	1.41	2.97	2.63	2.20	2.56	0.88	0.26
8	1.651	1.90	3.30	4.45	3.78	5.01	2.38	1.07
9	1.168	1.94	2.94	5.39	4.96	6.94	3.99	2.14
10	0.833	2.89	3.18	6.04	5.92	8.64	6.73	4.33
11	0.589	3.93	3.55	8.47	9.38	11.94	11.88	9.13
12	0.417	4.66	2.37	7.46	7.01	9.14	9.14	10.05
13	0.295	5.49	2.48	8.31	9.58	10.22	11.27	13.10
14	0.208	6.29	1.85	7.46	8.36	7.61	8.91	10.87
15	0.147	8.91	1.72	10.13	9.62	8.99	9.93	11.59
16	0.104	9.15	1.25	8.35	7.67	6.62	8.23	9.76
17	0.074	6.43	0.70	5.74	4.89	4.21	5.30	5.89
19	Through 0.074	28.30	1.41	18.33	15.45	14.55	20.39	21.64
	Aver. size of particle, mm.	2.896	9.900	1.052	1.456	0.688	0.397	0.288
	Total energy units	1,301.83	487.82	1,316.38	1,261.70	1,292.85	1,403.48	1,458.70

TABLE 2.—Screen Tests.—(Continued)

Ordinal Number	Screen Aperture, Millimeters	41D-1	41E-1	42-1	42-2	42-3	42-4	44-1
-1.0	38.100							
0	26.670					0.08		
1	18.850					0.76	0.58	
2	13.330					2.99	1.44	
3	9.423			0.02	0.04	3.38	2.55	
4	6.680			0.02	0.29	4.08	3.59	0.02
5	4.699				1.08	4.03	2.64	
6	3.327	0.11		0.29	2.47	5.40	3.67	0.16
7	2.362	0.05	0.06	0.83	4.56	6.96	5.96	0.41
8	1.651	0.33	0.15	2.45	8.22	8.70	8.02	1.27
9	1.168	0.75	0.48	4.34	9.30	8.06	8.10	2.76
10	0.833	1.93	1.58	6.92	9.86	7.79	8.80	5.41
11	0.589	6.22	4.68	12.37	12.35	9.88	11.37	10.50
12	0.417	8.91	7.98	10.15	9.18	6.99	8.40	10.96
13	0.295	12.20	12.57	12.40	9.72	7.73	8.42	12.45
14	0.208	11.27	12.48	10.55	6.91	5.91	6.57	9.35
15	0.147	13.21	14.73	10.88	8.10	5.93	6.74	11.73
16	0.104	11.50	11.87	8.48	5.72	4.24	4.84	9.37
17	0.074	7.29	7.35	5.12	3.78	2.44	2.84	6.14
19	Through 0.074	26.23	26.07	15.18	8.42	4.65	5.47	19.47
	Aver. size of particle, mm.....	0.215	0.196	0.388	0.768	2.129	1.628	0.316
	Total energy units	1,525.87	1,538.26	1,376.66	1,207.65	1,016.92	1,169.53	1,432.96

TABLE 2.—Screen Tests.—(Continued)

Ordinal Number	Screen Aperture Millimeters,	44-2	45-A	45-1	46-1	47-1	47-2	49-1
-1.0	38.100							
0	26.670		5.62					
1	18.850		14.72					
2	13.330		24.42					
3	9.423		15.92					
4	6.680	0.02	10.18	0.04	0.01	0.02	0.04	0.01
5	4.699	0.07	7.65	0.38			0.53	
6	3.327	0.08	4.92	0.08		0.78	1.21	
7	2.362	0.27	3.03	0.36	0.52	1.79	2.24	0.04
8	1.651	0.56	2.50	1.27	1.12	5.44	5.30	0.41
9	1.168	1.29	1.71	2.05	2.24	7.98	8.20	0.89
10	0.833	3.12	1.25	4.49	3.15	9.07	9.04	1.93
11	0.589	7.10	1.19	7.58	6.26	11.77	11.53	5.07
12	0.417	8.71	0.79	7.82	6.84	8.30	8.65	7.20
13	0.295	12.29	0.83	8.88	7.66	9.87	9.79	11.80
14	0.208	11.33	0.57	7.29	6.62	7.91	8.10	11.41
15	0.147	13.17	0.84	9.97	8.58	9.16	9.04	13.75
16	0.104	10.59	0.83	9.84	8.81	7.99	7.98	11.29
17	0.074	6.98	0.68	7.03	6.32	5.44	5.23	8.72
19	Through 0.074	24.42	2.35	32.92	41.87	14.54	13.12	27.48
	Aver. size of particle, mm.....	0.246	10.394	0.275	0.228	0.509	0.537	0.202
	Total energy units	1,500.73	384.29	1,525.27	1,580.21	1,324.53	1,305.35	1,543.11

TABLE 2.—Screen Tests.—(Continued)

Ordinal Number	Screen Aperture, Millimeters	50-1	51-A	51-1	52-A	52-1	32-1	32-2
-1.0	38.100						11.82	
0	26.670						45.47	2.88
1	18.850		0.14				20.97	12.58
2	13.330		0.39		0.02		12.67	25.14
3	9.423		0.76		0.06		3.96	21.01
4	6.680	0.02	1.19	0.02	0.18		1.88	15.26
5	4.699		3.60		1.78		0.65	6.46
6	3.327		5.15	0.36	1.33		0.41	3.34
7	2.362	0.04	5.44		1.60	0.06	0.48	2.02
8	1.651	0.43	8.47	0.25	2.31	0.12	0.22	1.33
9	1.168	0.60	8.47	0.32	2.20	0.23	0.22	0.94
10	0.833	1.34	8.79	0.38	2.43	0.52	0.22	0.75
11	0.589	3.89	11.01	1.08	3.60	0.67	0.25	1.37
12	0.417	5.89	8.66	2.12	3.79	0.58	0.19	1.15
13	0.295	10.76	9.06	5.29	5.79	1.42	0.15	1.14
14	0.208	11.65	6.82	9.78	5.71	3.48	0.09	0.88
15	0.147	14.90	7.49	17.18	9.69	11.44	0.09	0.76
16	0.104	11.58	5.10	17.65	9.63	15.41	0.11	0.92
17	0.074	9.69	2.93	12.26	8.37	13.60	0.05	0.62
19	Through 0.074	29.21	6.53	33.31	41.61	52.47	0.10	1.45
	Aver. size of particle, mm.	0.184	1.173	0.140	0.380	0.091	22.820	10.010
	Total energy units	1,583.89	1,130.23	1,635.04	1,563.10	1,735.92	83.00	390.15

TABLE 2.—Screen Tests.—(Continued)

Ordinal Number	Screen Aperture, Millimeters	32-2a	32-3	32-4	32-7	32-6	32-8	5321
-1.0	38.100							
0	26.670	4.67	0.09					
1	18.850	20.41	6.22					
2	13.330	40.80	27.25			6.00	0.80	
3	9.423	34.12	34.03	15.41		15.89	3.85	
4	6.680		18.23	27.77	0.20	23.27	13.32	0.13
5	4.699		5.35	20.86	0.81	12.88	9.79	0.60
6	3.327		2.15	16.75	1.18	14.72	18.83	2.10
7	2.362		0.69	8.23	2.31	8.91	7.99	5.81
8	1.651		0.17	4.40	3.85	4.19	8.01	19.42
9	1.168		0.07	2.39	6.11	2.13	6.03	23.27
10	0.833		0.07	1.34	9.38	1.53	5.70	25.11
11	0.589		0.09	1.15	14.94	1.39	7.46	17.89
12	0.417		0.11	0.33	14.78	0.86	4.71	4.33
13	0.295		0.18	0.43	12.63	0.84	4.63	0.80
14	0.208		0.18	0.24	8.01	0.57	2.79	0.18
15	0.147		0.36	0.14	6.36	0.73	2.49	0.12
16	0.104		0.53	0.19	5.10	1.02	2.47	0.09
17	0.074		1.34	0.13	3.27	1.73	1.42	0.09
19	Through 0.074		2.89	0.24	11.07	3.36	4.71	0.06
	Aver. size of particle, mm.	13.750	9.600	5.170	0.583	5.280	2.810	11.720
	Total energy units	204.37	382.60	527.30	1,269.69	600.74	853.40	940.75

TABLE 2.—*Screen Tests.*—(Continued)

Ordinal Number	Screen Aperture, Millimeters	5331	1	2	3	4
-1.0	38.100					
0	26.670				7.81	
1	18.850		1.14		8.78	
2	13.330		7.87	0.24	25.76	
3	9.423		18.60	1.00	26.31	1.32
4	6.680		19.05	3.36	17.42	3.88
5	4.699	0.07	13.22	5.18	9.19	6.23
6	3.327	0.38	13.92	6.37	2.21	5.14
7	2.362	1.44	9.21	6.91	0.68	5.48
8	1.651	5.28	4.81	6.75	0.48	4.13
9	1.168	8.12	2.75	7.00	0.24	3.06
10	0.833	19.48	2.21	7.90	0.15	2.93
11	0.589	20.70	2.00	8.73	0.14	4.07
12	0.417	15.98	1.41	8.94	0.11	5.13
13	0.295	12.25	1.16	7.64	0.12	7.14
14	0.208	6.89	0.78	6.10	0.11	7.21
15	0.147	5.02	0.59	5.75	0.11	7.68
16	0.104	2.12	0.52	5.21	0.12	8.45
17	0.074	0.96	0.23	1.56	0.07	4.99
19	Through	1.31	0.53	11.36	0.19	23.13
	Aver. size of particle, mm.	0.644	5.746	1.350	11.350	1.219
Total energy units.....		1,148.81	534.39	1,125.61	294.75	1,284.95

TABLE 3

Test No.	Feed Rate, Pounds per Hour	Aver. Size of Feed, Millimeters	Aver. Size of Product, Millimeters	Ratio of Reduction	R.M.E.
202	1,000	24.58	0.142	173:1	9.30
203	2,000	24.58	8.590	2.88:1	10.30
204	1,500	24.58	6.573	3.74:1	9.13
213	1,500	9.90	0.288	34.2:1	9.86
216	3,000	9.90	0.388	25.5:1	17.30
217	6,000	9.90	0.768	12.9:1	29.21
218	12,000	9.90	2.129	4.65:1	41.10
219	9,000	9.90	1.628	6.08:1	43.50

by one of these mills is considered, together with the fact that the power consumption is practically the same whether the mill is loaded lightly or heavily, it should be apparent that it will pay well to expend the small amount of power necessary for handling the pulp in a closed circuit and thereby gain increased efficiency of the mill.

This conclusion is practically in accord with that reached by the usual method of analysis. Taking 0.295 mm. (48-mesh) as a limiting size sought, Table 4 shows the relative number of mills and, therefore, the relative amounts of power necessary to crush quartzite, the screen test of which is

shown in S. T. 39-A, to pass a 0.295-mm. screen at the rate of 12,000 lb. per hour. The same relative figures will, of course, hold for any multiple of this desired capacity.

TABLE 4

Test No.	Feed Rate, Pounds per Hour	Total Feed Including Returns from 12,000 Lb. per Hour Original Feed, Pounds per Hour	Minus 0.295-Mm. Material in Feed, Pounds per Hour	Minus 0.295-Mm. Material in Product, Pounds per Hour	Minus 0.295-Mm. Material Produced, Pounds per Hour	Number of Mills Needed
213	1,500	20,000	104	896	792	13.0
216	3,000	23,920	208	1,507	1,299	8.0
217	6,000	36,400	416	1,978	1,562	6.0
219	9,000	45,000	624	2,382	1,758	5.0
218	12,000	51,600	832	2,778	1,946	4.3

This table is based on the assumption that the efficiency of reduction is the same on the smaller material returned to the mill as it is on the larger original feed. This is not quite true, but it is nearly enough true for the purposes of this argument.

Fig. 10 presents a fact which goes far toward explaining the irregular performance often met with in machines following a crusher of the ball- or tube-mill type. It will be noted that at the end of the $6\frac{3}{4}$ hr. operation the divergence between feed and discharge rate at any given minute is as great as at the beginning of the run, despite a careful, regular feed. The rising portions of the curve are accompanied by a progressively coarser product. At the peaks the screen tests show but little crushing. This irregularity in discharge rate and character of product is greater in dry crushing than in wet crushing, but it is also distinctly apparent in wet crushing. In most mill practice the irregularity is smoothed out by crushing in closed circuit, the circuit acting as a balance. Where no such balance occurs through other features of mill design, it will be wise to make special provision if the machines treating the discharge require a close adjustment.

Effect of Moisture Content

Fig. 8, summarizing tests 210 to 215 inclusive, Table 1, shows distinctly the effect of moisture on the crushing efficiency and average size of product of the conical ball mill. The true maximum of the efficiency curve probably lies somewhere between 40 and 50 per cent. moisture. The decidedly higher efficiency of wet crushing over dry crushing is confirmed in tests 207 and 208 where the relative mechanical efficiency rises from 9.33 to 11.46 due to the addition of 19 per cent. water, which is decidedly less than the most efficient water quantity. It will be noted, however, on referring to screen tests 40-1, 41A-1, 41B-1, 41C-1, 41D-1, and 41E-1, Table 2, that a progressively finer product is obtained by in-

creasing the amount of water in the feed and that the decreased relative mechanical efficiency in tests 214 and 215 is due to increased power consumption.

Moisture content has an effect on the weight of crushing charge that can be held in a mill. If the mill is charged to the limit when pulp of a given moisture content is being fed, a slight decrease in the moisture content will cause the discharge of a considerable quantity of balls or pebbles, as the case may be. The converse of this statement is, of course, also true.

Effect of Slope

The principal factors in mill operation affected by changes in slope are the ball load and the character of the product.

The effect on the ball load is best shown in tests 205 and 206. At the end of test 205 the mill, then level, contained a charge of 4,503 lb. consisting of 2,455 lb. of 5-in., 1,110 lb. of 4-in., 703 lb. of 3-in. and 235 lb. of 1¾-in. balls. At the end of test 206, which started with this load and was continued for several hours, the mill being set at a slope of 0.64-in. per foot, there had been forced out of the mill 55 lb. of 5-in., 588 lb. of 4-in., and all the 3-in. and 1¾-in. balls, leaving a total charge of 5-in. and 4-in. balls weighing but 2,922 lb. The ratio of weight of rock in the mill to weight of balls was also reduced. This latter fact considerably lessens cushioning and increases the amount of crushing done by impact as compared to that done by abrasion. The result is reflected in the increased efficiency and more granular product obtained, as noted later.

The change in power required to operate at higher slopes is in no way commensurate with what would be expected from the decrease in ball load (see tests 205 and 206). This fact should be borne in mind in using the formula given for horsepower.

The effect of changes in slope on the relative mechanical efficiency is so small that contradictory results due, no doubt, to unavoidable experimental inaccuracies, are shown. Thus tests 212, 213, 220 and 221 show a point of least efficiency at 1¼-in. slope with higher efficiencies at 2½-in. slope and no slope. It is the writer's opinion that the relative mechanical efficiency increases with increase in slope within operating limits, but that the change will in all cases be small. Tests 222 and 223 compared show higher efficiency and finer grinding at the greater slope. The finer grinding in test 223 is due to the higher moisture content of the pulp, rather than to the increase in slope. The progressively coarser grinding with increasing slope in the quartzite series, tests 212, 213, 220 and 221, is typical of the results to be expected in this direction. In dry grinding (see tests 205, 206, and 207) a decided change in the character of the product takes place with change of slope. The material discharged from the mill when grinding with the axis horizontal, contains

a large percentage of — 200-mesh material and a considerable percentage of the coarsest sizes with a decided minimum in the amount of the intermediate sizes. The product of the tilted mill, on the other hand, is more uniform. There is decidedly less coarse material and decidedly less dust, the bulk of the product lying in the intermediate sizes. This difference is undoubtedly due to the difference in the character of the crushing done in the two cases. With the mill horizontal a considerable proportion of the load is rock. This rock acts as a cushion to the falling balls in the mill so that crushing by impact is greatly lessened and crushing by abrasion forms an important part of the work done. In such crushing many of the large particles in the feed are reduced in size but slightly and pass out practically untouched, while such work as is effective produces very fine material. Thus we have the large percentages of very coarse and very fine ingredients in the product. On the other hand, when the mill is tilted the amount of rock that it contains at any time is small in proportion to the crushing load, there is little or no cushioning and the amount of crushing done by impact is large in comparison with that done by abrasion. Under such circumstances a granular product is to be expected.

Ball Load.—Varying the weight of the ball load affects the power consumption, fineness of grinding and relative mechanical efficiency. Power consumption increases with increase in the ball load, but the rate of increase in power consumption is not so rapid as the rate of increase of the ball load. Thus in tests 204 and 205 the ball load is increased 12.5 per cent. while the corresponding increase in power is but 3.8 per cent. In tests 222 and 223 an increase in ball load of 51.8 per cent. produces an increase in power consumption of but 4 per cent. It must be noted, however, that in the latter instance the increase in ball load is accompanied by a change in slope and that the mechanical efficiency of the power chain is unquestionably less when the mill is tilted than when it is horizontal. The increase in ball load in test 205 as compared with 204 causes reduction in average size of product, from the same feed, from 6.573 mm. (0.26 in.) to 3.449 mm. (0.14 in.) or 47.5 per cent. This material increase in the fineness of the product, with its corresponding increase in the mechanical value of the pulp, is sufficient to cause an increase of 20.8 per cent. in the relative mechanical efficiency of the machine, notwithstanding the increased power. As noted previously, the writer believes that the apparently contradictory result presented in tests 222 and 223, where the product of the lightly loaded mill is the finer, is due to the increase in percentage of moisture in the latter product and that with the same moisture percentage in both cases a result in agreement with the first case cited would have been obtained. It is, however, the writer's opinion further that the more lightly loaded mill, tilted and with a carefully aligned power chain, should show a higher relative mechanical

efficiency, due to a reduction in power consumed, which would more than compensate for any increase in average size of the product.

Effect of Difference in Size of Balls.—A comparison of test 209 with 210 and of test 224 with 225 shows that the larger the average size of ball in the crushing load (up to 5-in. diameter) the smaller the power consumption and the higher the relative mechanical efficiency. It is to be further noted, that a mixture of 5-in. and 4-in. balls crushes finer than a mixture of 5-in., 4-in., and 3-in. balls of equal weight, when the crushing is done dry and the average size of the feed particles is 9.900 mm. (0.39 in.). When the work is done in the presence of water, as in tests 224 and 225, the product when the ball charge is a mixture of 5-in., 4-in., and 3-in. balls is slightly finer (0.509 mm. as against 0.539 mm.) than when the charge consists wholly of 5-in. balls, but in test 224 the moisture percentage was 47.0 per cent. as compared with 40.0 per cent. in test 225. By reference to the section on Effect of Moisture, it will be seen that this result is probably due to the difference in moisture content and that at the same moisture content the 5-in. balls would crush finer than the mixed charge. In any case the difference in fineness in favor of the mixed load is so slight as to fail to justify charging a ball mill working on coarse feed with anything smaller than 5-in. balls. The writer inclines to the belief that the presence of small balls is a hindrance, and that periodical sorting of the charge accompanied by removal of the small balls (less than 3-in. diameter) will increase capacity, decrease power consumption, decrease the average size of the product and materially increase the relative mechanical efficiency.

Size of Feed.—Comparison between tests 208 and 223 apparently indicates that the ball mill works more efficiently on a coarse feed (24.58 mm. (0.96 in.) average size) than on a finer feed (10.394 mm. (0.41 in.) average size). In test 223 the percentage of moisture present, 41.7 per cent., is practically that determined most favorable, while in 208, but 19.5 per cent. of water was present in the feed. Notwithstanding this fact the relative mechanical efficiency in crushing the larger feed is 11.46 as against 10.71 in the case of the finer feed. There is very little difference in the power consumption. This conclusion must, however, be limited by a statement as to the rate of feed, viz., 1,500 lb. per hour. The reduction ratio is but 7.02 in the case of the larger feed as against 45.6 for the finer feed. In neither case was the mill fed up to its most efficient capacity. Comparing the results obtained here with those obtained in the rate of feed tests (213 and 216 to 219 inclusive) we may expect that by pushing the capacity in the case of the smaller feed until the reduction ratio is in the neighborhood of 7.0 that the relative mechanical efficiency will rise to about 40, while from the same series of tests it is obvious that lessening the ratio beyond this point in the case of the coarser feed, by increasing the feed rate, would result in lowering the

relative mechanical efficiency. When these facts are taken into consideration the smaller feed gives most efficient operation.

This conclusion cannot, however, be extended to finer and finer feeds, as is apparent when the pebble mill runs on trap and quartzite, 228 and 229, are compared with the ball mill runs on the same rocks, tests 213 and 223. In the latter tests, with feeds of approximately the same average size, the efficiencies varied by but 7.9 per cent., the trap showing the higher result. In the pebble mill tests the average size of the quartzite feed was 1.173 mm. and of the trap feed 0.380 mm. The corresponding relative mechanical efficiencies were 5.37 and 1.89, a difference of 65 per cent., all of which must be ascribed to the fineness of the feed.

Length of Cylindrical Section.—The effect of increasing the length of the cylindrical section in a ball mill is to reduce the relative mechanical efficiency. This is due to the fact that the ball load and power consumption increase with increased length much more rapidly than the fineness of the product increases. Thus, by reference to Table 1 we find that, all conditions being constant other than those noted above, an increase in power consumption amounting to 89 per cent. occurs with an increase in length of cylindrical section from 16 in. to 48 in., the corresponding decrease in average size of product is but 36 per cent., and there is a resulting decrease in relative mechanical efficiency of 40 per cent., the 16-in. mill being taken as the standard of comparison. Therefore, if the desired capacity of a plant is sufficient to justify the installation of more than one mill, additional mills placed in series, each making a relatively small reduction, will be more efficient than an installation which attempts a large reduction ratio in one mill by increasing the length of the cylindrical section.

Pebbles vs. Balls

Test 228 presents the pebble mill working at a reduction ratio of 8.3, which is close to the most economical ratio. Under these conditions the relative mechanical efficiency is 5.37. Test 227 presents a ball mill of the same cylinder length working on a coarser feed but making a reduction of 53.6 to 1. Even under this unfavorable condition the relative mechanical efficiency is 5.88. If the rate of feed is raised and the reduction ratio correspondingly lowered to a point comparable with the pebble mill, we may expect a relative mechanical efficiency much higher. It is obvious, then, that the ball mill is a more efficient crushing machine than the pebble mill. It is also obvious that it will grind as fine as the pebble mill, when the products of the two tests above cited are compared. The ball-mill product is 0.184 mm. average size, produced from a 9.9-mm. feed, while the pebble-mill product is 0.140 mm. average size produced from a feed only 1.173 mm. average size. Given a feed of the same size, the ball-mill product would have been finer.

Records were kept throughout of ball and pebble consumption, but the results were so contradictory, due to the relatively short duration of the runs, that they are not worth presenting.

Character of Feed

When the feed to a ball mill is a rock similar to an average ore, no great difference in efficiency is noticeable as between different kinds. Tests 221 and 222 give a comparison of grinding efficiencies on quartzite and trap of approximately the same average size. The reduction ratios in the two cases are 40.2 and 37.8 respectively, giving products 0.246 and 0.275 mm. average size. The relative mechanical efficiencies are 9.06 in the case of the quartzite feed and 10.20 in the case of the trap-rock feed. When, however, tough, soft materials such as cocoanut shells or sawdust are tested (tests 230 to 234 inclusive) the efficiencies fall off rapidly to figures ranging from 0.48 to 1.67. This means, of course, that a crushing device employing impact chiefly is not suitable for reducing such material.

An interesting and unexpected result is to be noted in tests 235 and 236. The brass ashes treated in these tests consisted of a mixture of unburned coal, coal ash, and a brittle slag containing metal shot. The coal ash and slag ground up with surprising ease, the coal was easily broken to an intermediate size and then seemed to float through on the surface of the load in the mill, while the metal particles were discharged with very little flattening or abrasion. Thus, due to the heterogeneous character of the feed, a higher efficiency was obtained than would be expected where one of the ingredients was so tough. When, however, it was attempted to grind slowly and pulverize metal, the relative mechanical efficiency of the machine fell to figures ranging from 0.04 to 0.71, confirming the comparison made in the first part of this section between rock and such tough materials as sawdust and cocoanut shells.

Capacity

As stated in the introductory part of this paper, it has not been possible to extend the series of tests to gain capacity figures on mills of different diameters. The writer has, however, some figures on the capacity of 6-ft. and 8-ft. ball mills which indicate that with a feed of average ore of 10 mm. average diameter, grinding wet to pass a 20-mesh (0.833-mm.) screen, the capacity will vary as a function of the cube of the nominal diameter of the mill. Approximate capacities for this duty for mills 4.5 ft. diameter and larger may be derived from the formula

$$C = 0.95D^3 - 65$$

where C = the capacity in tons per 24 hr. and D = the nominal diameter of the mill in feet. This formula should, however, be used with caution.

Distribution of Crushing Charge

At the end of one of the runs on the $4\frac{1}{2}$ by 32 mill, the balls were sorted as they were taken from the mill into heaps corresponding to the portion of the mill from which they were removed. For the purpose of this classification the charge was divided by theoretical vertical planes into five sections, as shown in the diagrammatic sketch, Fig. 9. The heaps taken from each of these sections were then sorted into sizes, with the result shown graphically in Fig. 9. It will be seen from this figure that there is a marked segregation of large balls in sections 1 and 2 at the head end of the mill. The segregation is, however, by no means complete, as is shown by the fact that the average size of ball in the mixture in sections 3, 4, and 5 is greater than 4 in.

CONCLUSIONS

1. In crushing average ores the character of the gangue has but little effect on the relative mechanical efficiency of the conical mill.
2. The mill is not suitable for grinding soft, tough materials.
3. The ball mill works more efficiently on material of intermediate (0.5 in. to 0.75 in. average) size than on either a coarser or a finer feed.
4. A greater ratio of reduction in average size of material can be expected with feed of an intermediate size than with a coarse feed.
5. Steel balls are much more efficient crushing media than pebbles.
6. Steel balls will grind as fine or finer than pebbles when working on the same feed.
7. Increase in the weight of the ball load, other conditions remaining constant, increases the ratio of reduction and the relative mechanical efficiency of the mill.
8. The power consumption increases with increase in the weight of the ball load, but this increase in power consumption is not in direct proportion to the increase in load.
9. Power consumption decreases with increase in the average size of the balls composing the crushing load up to an average size of 5 in.
10. A ball charge composed of 5-in. balls makes a greater reduction in size of particle at one passage through the mill than a mixed charge composed of 5-in., 4-in., and 3-in. balls.
11. The relative mechanical efficiency of the ball mill increases with the average size of ball in the crushing charge up to 5 in. average diameter.
12. The relative mechanical efficiency of the mill increases with the rate of feed to the point of overload.

13. Increase in the length of cylindrical section in the conical ball mill increases the reduction ratio at the expense of a marked decrease in the relative mechanical efficiency.

14. Increase in the slope of the mill axis decreases the ball load materially, but the corresponding decrease in power consumption is in no way commensurate.

15. In general, increase in slope tends to produce a more granular product with less very fine and less coarse ingredients than are present in the product of the mill set with the axis horizontal.

16. Increase in slope has but little effect on the relative mechanical efficiency.

17. Other conditions being constant, the relative mechanical efficiency of the mill is a maximum at between 40 and 50 per cent. moisture content in the feed.

18. The relative mechanical efficiency in wet crushing is decidedly greater than in dry crushing.

19. The increase in the percentage of moisture in the feed causes an increase in the reduction ratio.

20. Power consumption increases slightly with increase in the moisture content of the feed.

21. The rate of discharge and the character of the product of the mill fluctuate continually through rather wide limits. This fluctuation is greatest in dry crushing.

22. The conical mill should be operated in closed circuit with a sizing device which will return to it the oversize from its product. In this installation the rate of feed should be raised until the relative mechanical efficiency shows a maximum. When operating as a ball mill, the ratio of length of cylindrical section to diameter should not exceed 0.3. This will be a much more economical installation than one which seeks, by slow feeding or long cylindrical section, to obtain a finished product at one passage through the mill. In wet grinding the moisture content of the feed should be kept about 40 per cent. The slope should be adjusted to mill requirements, but for ordinary concentrating-mill practice should be about 0.4 in. per foot. The ball charge should be the maximum that the mill will hold and should be kept as large in average size as is possible without too great sacrifice of small balls.

ACKNOWLEDGMENTS

The writer is indebted to Percy F. Smith, Professor of Mathematics in the Sheffield Scientific School, for valuable assistance in the development of the formulæ, and to Professor Herbert L. Seward of the department of Mechanical Engineering for help with the nomograms.

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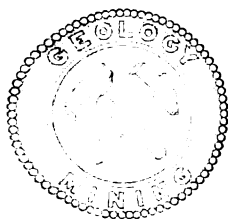
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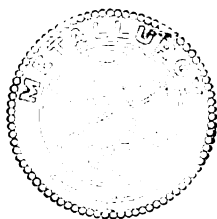
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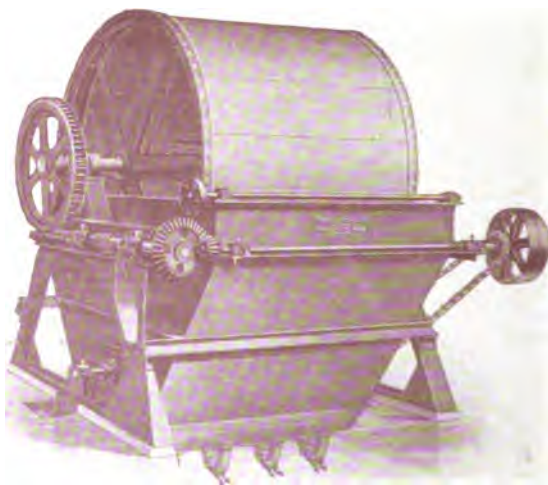
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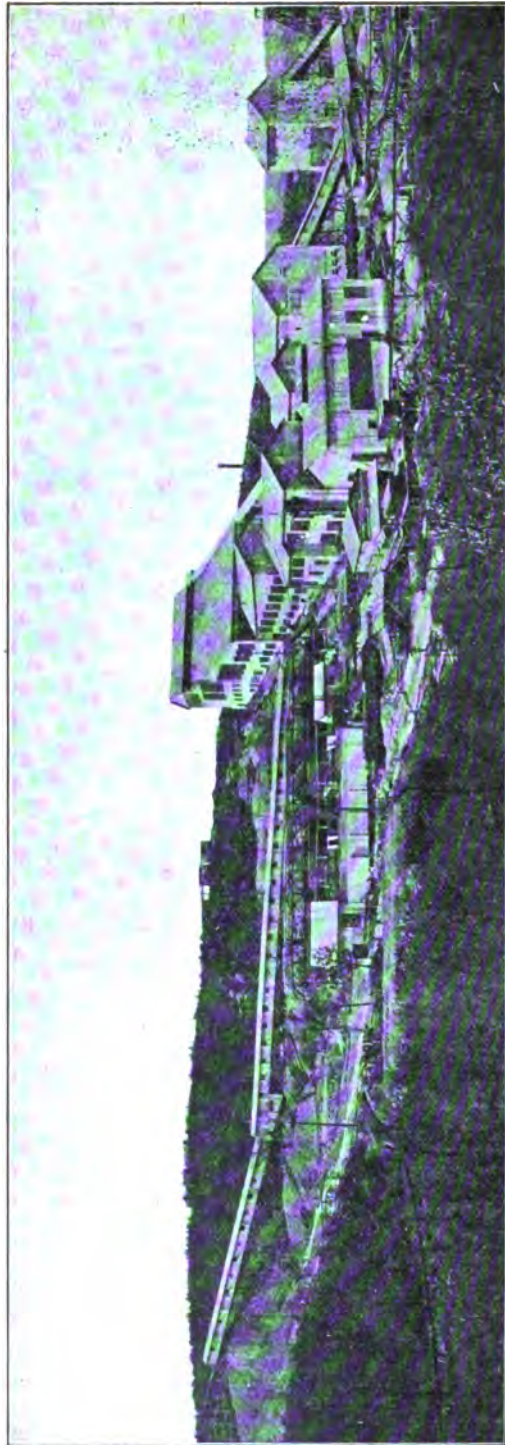
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BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 125

MAY

1917

Published Monthly by the American Institute of Mining Engineers at 212-218 York St., York, Pa., H. A. WISOTZKEY, Publication Manager. Editorial Office, 29 West 39th St., New York, N. Y., BRADLEY STROUGHTON, Editor. Cable address, "Aime," Western Union Telegraph Code. Subscription (including postage), \$12 per annum; to members of the Institute, public libraries, educational institutions and technical societies, \$5 per annum. Single copies (including postage), \$1 each; to members of the Institute, public libraries, etc., 50 cents each.

Entered as Second Class matter January 28, 1914, at the Post Office at
York, Pennsylvania, under the Act of March 3, 1879.

BULLETIN WANTED

The Institute would be glad to purchase from any Member a copy of the *Bulletin* for May, 1914.

ST. LOUIS MEETING

OCT. 8-13, 1917

The members of the Institute living in the lower Mississippi Valley extend a hearty and cordial invitation to their fellow-members to meet with them next October, in St. Louis.

They believe that it will be a meeting of singular interest, inasmuch as the district in question can show in active production almost all of the more useful metals and non-metals.

Perhaps too few know that the Southeast Missouri District, about 80 miles south of St. Louis, and to which a special trip is planned, is the largest lead district in the world. It was that before the European war, and now that the production of lead has greatly decreased abroad, and has correspondingly increased in the United States, this is one of the most interesting of all lead districts.

In 1913, the production of the world was 1,270,000 tons, the last world figures obtainable. In that same year the production of the United States

was 412,000 tons, and of Southeastern Missouri 133,000 tons. In 1916, the world's production having fallen off, the production of the United States was over 600,000 tons, and that of Southeastern Missouri 184,000 tons (32 per cent.).

In view of the great necessity for lead at present, and so far as we can foretell for the future, the possibilities not only in this known district but in contiguous areas are not only interesting but decidedly important.

The ore occurs not in veins, but disseminated through a great bed of magnesian limestone, more than 400 ft. thick in places, and when mined only carries an average of about 3 per cent. lead. It has taken nearly a generation to perfect the concentration plants, and the managers do not consider them perfect yet. Here this ore is crushed and cleaned until it will run about 70 per cent. lead. Some interesting developments in flotation are in progress at these mills, and nearly one-tenth of the clean ore shipped to the smelters is produced by flotation.

The resulting ore is smelted at the various plants of the different companies, located at Herculanum, Alton and Collinsville, with a combined capacity of 250,000 tons of lead per annum.

These smelters will all be visited on the occasion of the meeting.

The uses of the lead finally are interesting. Just now its tremendous importance as a war material overshadows the fact that probably one-half of the product goes into the form of white lead and other kinds of paint. Much is used as anti-friction metal and as pipe and sheet.

The following papers have been accepted for presentation at the St. Louis meeting, and others are in the hands of the Committee on Papers and Publications.

Methods for Determining the Capacities of Slime-Thickening Tanks.

By R. T. Mishler.

The Tayeh Iron-Ore Deposits. By Chung Yu Wang.

Exploration of Metalliferous Deposits. By W. H. Emmons.

Tests on the Hardinge Conical Mill. By Arthur F. Taggart.

Ore Deposits of the Boulder Batholith of Montana. By Paul Billingsley and J. A. Grimes.

Lead Mining and Smelting at Galetta, Ont. By William E. Newnam.

The Media Mill, Webb City, Mo. By H. B. Pulsifer.

Mine Models. By H. H. Stoek.

Coal Wastage. By Francis S. Peabody.

The Tredinnick-Pattinson Process. By William E. Newnam.

A New Silicate of Lead and Zinc. By P. A. van der Meulen.

The History and Legal Phases of the Smoke Problem. By Ligon Johnson.

Characteristics of Zinc Deposits in North America. Their Bearing on Origin and Tenure of Economic Life. By Frank L. Nason.

The Southern Extremity of the "Clinton" Gas Pools. By L. S. Panyity.

A Standard Screen Scale for Testing Sieves. Recommended for general adoption by a conference of representatives of various scientific and technical societies, government bureaus, and private firms, held at the U. S. Bureau of Standards.

The Replacement of Sulphides by Quartz. By H. N. Wolcott.

A Feasible Plan for Gaging Individual Wells. By R. H. Johnson and W. E. Bernard.

Proceedings of the Meeting of the Board of Directors, Mar. 23, 1917

Upon nomination of the President, the following Committee on Membership was appointed:

Karl Eilers, Chairman,
Arthur S. Dwight,
Lewis W. Francis,
Louis D. Huntoon,
Arthur L. Walker.

The following resolution was then unanimously passed:

WHEREAS, the integrity and life of this nation is dependent upon its ability to successfully defend itself against foreign aggression and attack, and

WHEREAS, it is necessary that the nation be properly prepared to the end that our untrained patriotic volunteers be not uselessly slaughtered and our country disgraced and defeated, be it therefore

RESOLVED: That the Board of Directors of the American Institute of Mining Engineers believe that compulsory universal military training should be at once instituted and that adequate, complete and immediate preparedness should be provided.

BE IT FURTHER RESOLVED: That, we believe that thorough and complete preparedness in the army, the navy and the industries of the country is vital; anything less is unsatisfactory and inefficient and may result in disaster to the nation.

Reports in writing were received from the Engineering Foundation, The Council of National Defense, the Treasurer and Finance Committee, the Trustees of the United Engineering Society, the Women's Auxiliary of the A. I. M. E. and the Library Committee.

Upon the recommendation of the Finance Committee, it was resolved to purchase paper for the *Bulletin* and *Transactions* equivalent to the Institute's publication needs, up to the middle of the year 1918.

The sum of \$100 was appropriated to the Puget Sound Section.

Upon the recommendation of the Library Committee, it was voted that, in the opinion of the Board, it is desirable to transfer the library of the Institute to the United Engineering Society.

A request from the New York Stock Exchange Committee on Stock Lists for suggestions for the listing of companies owning and operating petroleum and gas wells was referred to the Committee on Petroleum and Gas with a request that it report its recommendations to this Board.

Upon the recommendation of the Committee on Membership, members, associates and junior members were elected.

The resignations of twenty-six members were accepted, and the dues were suspended of three men who are engaged in active duty with the armies at the front.

The time of payment of one member was extended for one year, and seven members who had paid their arrears of dues were reinstated.

Upon recommendation of the Committee on Papers and Publications, the first prize of the President's Prizes of the year 1915 was awarded to Joseph J. Beeson.

The following resolution was passed:

RESOLVED: that, in view of the present acute condition of the paper industry and the indication that the present scarcity of paper will continue for at least twelve months longer, the Editorial Department of the

American Institute of Mining Engineers is instructed to condense as much as is compatible with clarity and literary merit, every manuscript accepted for publication; that the condensed papers be submitted to their authors and that, if there be thereafter any question as to their clarity or literary merit, such questions shall be referred back to the Committee on Publications.

Presentation of the John Fritz Medal to Professor Howe

The presentation of the John Fritz Medal to Professor Henry M. Howe will take place in the auditorium of the Engineering Societies' Building, 29 West 39th Street, New York, at 8.30 o'clock on the evening of May 10.

It is hoped that many members of the Institute will be able to attend this ceremony, and the committee desires to make special mention of the fact that ladies are cordially invited to be present.

The John Fritz Medal Book

The price of the John Fritz Medal book has been reduced to \$2.50. The Board of Award will refund the difference between this amount and the amount paid for the book to anyone who has already purchased a copy at a higher price. Address the Secretary of the John Fritz Medal Board of Award, Engineering Societies' Building, 29 West 39th Street, New York, N. Y.

Herbert C. Hoover as Food Controller

The paragraphs quoted, regarding Mr. Hoover's appointment as chairman of the newly organized food board of the United States, are taken from the editorial page of the *New York Times* for April 13:

"The country is fortunate in commanding the services of Herbert C. Hoover as Chairman of the newly established Food Board. This is a problem of first importance. Few realize the extent to which American food stuffs will enter into the decision of the war, and it is doubtful if it has occurred to many minds as yet that we shall have to deny ourselves what we have regarded as necessities in order to keep our European allies supplied.

"Mr. Hoover is one of the leading mining engineers of the country. At the outbreak of the war he put his private affairs aside to become the head of the American Commission for Relief in Belgium. He has had an experience that perhaps no other man ever had, covering the raising of money and the purchase and distribution of supplies on a large scale. He has been brought into close relations with the neutrals and the belligerents on both sides. By them all he is recognized as one of the best executives brought to the fore by the war. Mr. Hoover is in a position now to place abilities and experience at the service of his country. He will come back with comparative data that will enable him to measure resources and deficiencies accurately. This appointment gives renewed assurance of our Government's grasp of the problems of war and its resolve to apply business principles to their solution."

In this matter of food supply and food conservation, the work of each individual will count. The following letter comes from Washington:

DEPARTMENT OF COMMERCE
Washington

April 14, 1917.

To Commercial Organizations:

The war in which we are now engaged is a war of economic resources. It is absolutely essential to the successful prosecution of the war that everyone make his or her contribution to the economic welfare of the country. The production of food is a vital and present duty resting on every man and woman who can help it along. Without food workmen can not work, nor can armies fight.

The food supply of the country must be increased, and I urge upon you to coöperate in every way with the Department of Agriculture in its campaign to increase the crops of the country. Will you not take this subject up at once with the membership of your association, pointing out to them the needs of the situation and urge immediate action? I suggest that every organization should have a committee on the production of foodstuffs and that the assistance of women's organizations be enlisted in the campaign.

I need not point out that the planting season is at hand, and that any action to be effective must be taken at the earliest possible moment. In the United States, as in the warring countries of Europe, the effects of this struggle will be felt by everyone and economic preparedness will greatly lessen the burden that we must carry.

Very truly yours,
(Signed) William C. Redfield,
Secretary.

President's Prize Awarded to J. J. Beeson

The first prize of the President's Prizes has been awarded to J. J. Beeson for his paper entitled "Disseminated Copper Ores of Bingham Canyon." When this paper was written, Mr. Beeson was a student at Stanford University, and was president of the Stanford Geological and Mining Society.

The President's Prizes were established in 1915, funds being provided at that time for three prizes of \$50, \$30 and \$20 respectively, for the best essays, or other papers submitted in competition by Junior Members and Members of Affiliated Student Societies. The original intention was to award all three during the year 1915, but the committee has deemed it advisable to extend the time for the second and third prizes, and the Secretary of the Institute will be glad to hear from any member who wishes to compete for one of these prizes.

A competitor for the President's Prizes must be, at the time of submitting his essay, either a Junior Member or a Member of an Affiliated Student Society of this Institute.

The essays must be upon some subject related to Mining, Economic Geology, or Metallurgy. In making the awards, special weight will be given to originality in investigation and treatment; and to literary merit. As a general rule, the essays should not be less than 3,000, nor

more than 10,000 words. Graduation theses (with the permission of the authorities of the writer's institution, where necessary), or dissertations for higher degrees, are acceptable.

Emmons Memorial Fellowship

Announcement is made that applications for the Emmons Memorial Fellowship in Economic Geology will be considered, and an incumbent appointed prior to the first of June. Applications should be sent to Professor J. F. Kemp of Columbia University not later than May 15. The holder of this fellowship receives an income of \$1,000 and is expected to carry on an investigation in Economic Geology which will add a distinct contribution to the literature. The institution at which the study is pursued will be decided after conference with the members of the Committee of Awards. The character of the investigation will be decided after applications have been passed upon.

Forthcoming Meetings of Societies

Organisation	Place	Date 1917
American Electro-Chemical Society.....	Detroit, Mich.	May 2-5
American Waterworks Association.....	Richmond, Va.	May 7-11
American Institute of Electrical Engineers.....	New York City	May 18
American Society of Mechanical Engineers.....	Cincinnati, Ohio	May 22-25
American Iron and Steel Institute.....	New York City	May 25-26
Interstate Oil Mill Superintendents Association.....	Augusta, Ga.	May (last wk.)
Electric Power Club.....	Hot Springs, Va.	June 11-14
Society for the Promotion of Engineering Education.....	Evansville, Ind.	June 19-22
American Institute of Electrical Engineers.....	Hot Springs, Va.	June 25
American Society for Testing Materials.....	Atlantic City, N. J.	June 26-30
Mine Inspectors Institute of U. S. A.....	Indianapolis, Ind.	July 9-11
National Association of Stationary Engineers.....	Evansville, Ind.	Sept. 10-15
American Iron, Steel & Electrical Engineers.....	Philadelphia, Pa.	Sept. 10-14
American Institute of Metals.....	Boston, Mass.	Sept. 10-14
Chemical Industries, Third National Exhibition.....	New York City	Sept. 24-29
American Electro-Chemical Society.....	Pittsburgh, Pa.	Oct. 3-6
American Institute of Mining Engineers.....	St. Louis, Mo.	Oct. 8-13
American Gas Institute.....	Washington, D. C.	Oct. 16-19
American Institute of Architects.....	Philadelphia, Pa.	Dec. 26-29
American Society of Mechanical Engineers.....	New York City	Dec. 26-29
American Mining Congress State Chapter.....	Phoenix, Ariz.	Dec. 26-29

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members)

The following is a partial list of members and guests who called at Institute headquarters during the period Mar. 10, 1917, to Apr. 10, 1917.

Frederick R. Ahbe, Athens, Pa.
J. L. W. Birkinbine, Philadelphia, Pa.
Samuel D. Bridge, Monterrey, Mex.
J. R. Buchanan, Pasadena, Cal.
G. M. Colvocoresses, Humboldt, Ariz.
F. G. Cottrell, Washington, D. C.
H. N. Eavenson, Gary, W. Va.
H. T. Fisher, Englewood, N. J.
E. P. Fleming, Salt Lake City, Utah.
F. L. Garrison, Philadelphia, Pa.
W. M. Grant, So. Bethlehem, Pa.
J. F. Hanst, Negaunee, Mich.
H. D. Hibbard, Plainfield, N. J.
B. B. Hood, Chrome, N. J.
Henry M. Howe, Bedford Hills, N. Y.
Hennen Jennings, Washington, D. C.
Oscar Lachmund, Greenwood, B. C.

Frank M. Leland, San Francisco, Cal.
Edwin Ludlow, Lansford, Pa.
E. P. Mathewson, Toronto, Canada.
E. V. Matlack, Webster Grove, Mo.
E. C. Potter, Clearwater, Mont.
R. B. Stanford, Bluefields, Nicaragua.
R. S. Ransom, Jr., Newark, N. J.
M. L. Requa, San Francisco, Cal.
John A. Rice, El Paso, Tex.
W. E. Rice, Ringwood, N. J.
Lewis A. Riley, Philadelphia, Pa.
Hallet R. Robbins, Vancouver, B. C.
Allen H. Rogers, Boston, Mass.
A. A. Stevenson, Philadelphia, Pa.
Wm. Huff Wagner, Johnstown, Pa.
E. E. Watts, Denver, Colo.
Samuel S. Wyer, Columbus, Ohio.

H. W. Aldrich, after ten years of service with the Anaconda Copper Mining Co., at Anaconda, has resigned and accepted a position as general superintendent for the Ladysmith Smelting Works, at Ladysmith, B. C.

Charles E. Bunker, formerly superintendent of the Keystone mine, has been appointed superintendent of the Bunker Hill mine near Amador City, Cal.

Edgar A. Collins sailed for Yokohama and Vladivostock to take the position of manager of the Ridder mine of the Irtysh Corporation.

Dr. James Douglas has resigned as president of Phelps, Dodge & Co., and has been appointed chairman of the Board of Directors.

Walter Douglas has been elected president of Phelps, Dodge & Co.

William W. Elmer will be in Alaska for several months, engaged in engineering work.

Sir Robert Hadfield, through the Council of the British Institution of Mechanical Engineers, has offered £200 to provide a prize or prizes for a new and accurate method of determining the hardness of metals.

J. F. Hanst, formerly senior engineer on the staff of the Cleveland-Cliffs Iron Co., Ishpeming, Mich., has accepted a position with the Andes Copper Mining Co., Potrerillos, Chile.

Ott F. Heizer is now general manager for the Louisiana Consolidated Mining Co., at Tonopah, Nev., re-opening the old Tybo mine.

Rush M. Hess has resigned the superintendency of the Stoddard Mines Co. and will be in New York shortly.

S. A. Holman, for the past three years mine superintendent for the Balaklala Consolidated Copper Co., has been appointed manager, effective March 15th.

Henry M. Howe has been elected to the National Academy of Science.

Dr. Edward Keller has severed his connection with the Anaconda Copper Mining Co., with whom he was connected for many years as chemist and metallurgist.

Arthur Knapp, petroleum engineer, lately returned from the Russian oil fields, is at present engaged in managing an oil producing property in the United States. He has been appointed captain in the Engineering Corps of the Officers' Reserve Corps of the U. S. Army.

Frank C. Laurie has been appointed general superintendent of the Freeport and Tampico Fuel Oil Corporation, with headquarters in Tampico, Mexico. Mr. Laurie was formerly one of the managers for the Mexican Eagle Oil Co. in Mexico.

Jay Lonergan has resigned the position of professor of mining at the Peking Government University, China, and will resume active practice in the United States.

J. W. Maller, until recently in the New York office of the Ingersoll-Rand Co., has accepted a position with W. R. Grace & Co., and will be in charge of the mining machinery department with headquarters at Lima, Peru. He sailed for Lima in April.

E. P. Mathewson was presented with the medal of the Mining and Metallurgical Society of America by W. R. Ingalls at the recent meeting of the Canadian Mining Institute at Montreal, in recognition of his contributions to the advancement of metallurgy.

H. A. Morrison, who has been superintendent at the Silver Dyke mine, Sodaville, Nev., is now engaged in unwatering and opening up the Oak Hill mine, La Grange, Cal.

Jesse C. Porter and **John B. Stewart**, consulting mining and metallurgical engineers of New York, are now in charge of the Havana branch of C. L. Constant Co. (of New York) permanently established in the National City Bank (of New York) building, located at Calle Cuba No. 74.

E. C. Reeder has been appointed manager of the Chicago branch of the Clyde Iron Works, Duluth, Minn., with headquarters at 600 Fisher Building.

Dr. Louis D. Ricketts has been elected a life trustee of Princeton University.

E. K. Soper and **R. R. Goodrich** have been appointed consulting metallurgists for the U. S. Bureau of Mines, and will work in connection with the mining experiment station established by the Bureau at the University of Idaho, Moscow, Ida.

D. R. Thomas has resigned as manager for the Davidson Gold Mines, Ltd., South Porcupine, and is temporarily located in New York City.

Charles E. Van Barneveld is in charge of the Tucson station of the U. S. Bureau of Mines.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members)

Member, aged 33, technical graduate, B. S. and E. M., soon to complete contract in Spanish America, desires to get in touch with employers. Ten years mine, mill, cyanide and construction experience. Prefers executive position or assistant to executive. No. 352.

Member, University graduate, desires responsible position with good company. Married. Age, 37. Fourteen years' experience in Australia, Mexico and United States, milling, cyaniding and concentrating, gravity

and flotation. Last ten years has been employed as superintendent and assistant manager. Speaks Spanish. Best references as to character and ability. No. 353.

Member, technical graduate, married, age 29, with 7 years' experience in mining and construction, with good record in handling men. Desires position of responsibility. Can furnish best of references. No. 354.

LOCAL SECTION NEWS

PUGET SOUND SECTION

SIMON H. ASH, *Chairman*, I. F. LAUCKS, *Vice-Chairman*,
CHARLES SIMENSTAD, *Sec.-Treas.*, 425 Lyon Building, Seattle, Wash.
GLENNVILLE A. COLLINS, JOHN N. POTT.

The Puget Sound Section met at the Arctic Club, Seattle, Wash., on Jan. 20, 1917.

Sixteen members and five guests were present.

Dinner having been served, the meeting was called to order.

A letter from secretary of Associated Engineering Societies was read, requesting payment of this section's assessment to the quarterly budget, and upon motion it was voted that the Secretary-Treasurer be instructed to pay this assessment.

It was moved and carried that this section join with the Chemical Society in a meeting.

At the close of the business meeting, Mr. Asa C. Baldwin gave an illustrated lecture on the "Alaska Boundary Survey."

On Feb. 24, the Puget Sound Section again held a meeting at the Arctic Club, 16 members and three guests being present.

Chester F. Lee, as member of a committee representing this section in the joint council of the Associated Engineering Societies, reported that by lots it had been decided that the term of service of said committee would be as follows: Chester F. Lee, two years; George Watkin Evans, one year.

Following adjournment of the business meeting, Prof. Joseph Daniels, of the University of Washington, delivered a very interesting lecture on "Milling Practice of The Alaska-Castineau Mining Company."

CHAS. SIMENSTAD, *Secretary*.

NEW YORK SECTION

PERCY E. BARBOUR, *Vice-Chairman*,
A. D. BEERS, *Secretary*, 55 Wall St., New York, N. Y.
C. A. BOHN, *Treasurer*,
JOHN V. N. DORR, LEWIS W. FRANCIS.

A meeting of the New York Section was held at Whyte's Restaurant, on Mar. 9, 1917, preceded by a dinner. About sixty members of the Section were in attendance.

The speaker of the evening was W. S. Kies, Vice-President of the American International Corporation, who took as his topic "Possibilities for American Capital in Foreign Trade Development."

Mr. Kies showed the new opportunities and the new obligations that have been created for American enterprise by the great struggle going on in Europe. He pointed out the necessity for an organization to mobilize American capital and explained the way in which the American International Corporation is undertaking this work.

Following Mr. Kies, J. Morgan Clements, who is to leave shortly for the Far East to investigate the mineral resources of the Orient under the direction of the Bureau of Foreign and Domestic Commerce of the Department of Commerce, spoke briefly, outlining the work the Government is undertaking and saying that the Department desires to coöperate with American capitalists in building up trade as a result of this undertaking.

A. D. BEERS, *Secretary*.

CHICAGO SECTION

CHARLES H. MACDOWELL, *Chairman*, LUTHER V. RICE, *Vice-Chairman*,
HENRY W. NICHOLS, *Secretary-Treasurer*, 1645 E. 68 St., Chicago, Ill.
ALEXANDER K. HAMILTON, HENRY P. HOWLAND,
GEORGE P. HULST, FREDERICK T. SNYDER.

The Chicago Section met on Friday, March 2, at the Chicago Engineers' Club. After the dinner, Major Paul B. Malone, of the United States Army, told of the Officers' Reserve Corps and of many ways in which engineers can be of service in the preparations for the defense of the country.

R. J. H. DeLoach made an address illustrated by the stereopticon, on "The Bird, Our Fellow Citizen."

SOUTHERN CALIFORNIA SECTION

C. COLCOCK JONES, *Chairman*, PHILIP WISEMAN, *Vice-Chairman*,
ALVIN B. CARPENTER, *Secretary-Treasurer*, 508 Union League Building,
Los Angeles, Cal.
A. B. W. HODGES, R. A. PEREZ,
E. A. MONTGOMERY, WILLIAM F. STAUNTON.

This Section had the first of its four scheduled meetings of the season 1916-1917 on Nov. 27, 1916, but owing to the sudden illness of the Secretary, F. J. H. Merrill, on that date, and to his subsequent death, no report of this meeting was sent in.

The Section met for dinner at the Sierra Madre Club, about sixty members being present. After the dinner, the subject of the evening was announced as "A Resumé of the Arizona Meeting of the Institute."

Seeley W. Mudd, the leader for the evening, gave a general review of the trip with the Institute through Arizona, including a tribute to those who shared in the generous entertainment of the members. He then introduced the following men, who responded with descriptions of the various phases of mining and metallurgical work that was of special interest to those privileged to attend the main Institute meeting of the year.

Mr. Wartenweiler gave an interesting description of the mechanical features of the plants visited.

Mr. Penhoel covered the subject of flotation in a thorough manner, and in the discussion that followed, Mr. Crerar and others who have had experience with flotation practice gave data of great interest.

Alvin B. Carpenter followed with descriptions of the various methods of mining and caving systems used for the extraction of the ore in the mines at Bisbee, Ray, Miami and Inspiration.

Philip Wiseman and Albert Doer gave interesting general talks on the mines and the equipments at the Inspiration and Miami properties.

H. O. Chute, a chemical engineer and a guest, concluded the evening program with an instructive talk on flotation oils from wood distillation and the desires of the manufacturers to make their products to best conform with the requirements for flotation work.

After a brief social session, following the program, the meeting adjourned.

The second scheduled meeting of the Section was held in the main dining-room of the Sierra Madre Club. The subject of the meeting was "Petroleum" and the members of the four other National Technical Societies had been invited to attend. One hundred and fifty men sat down to the dinner, at the conclusion of which the Chairman, C. Colcock Jones, introduced Dr. David T. Day who spoke of the life work of Dr. Joseph A. Holmes and made an appeal for the support of the Dr. Joseph A. Holmes Memorial Fund, in commemoration of what Dr. Holmes had done to make mining safe.

Mr. Jones then introduced Ralph Arnold as leader of the meeting, and Mr. Arnold, after receiving the "buck" introduced the following speakers:

Captain John Barneson, of the General Petroleum Co., gave a most important address, "The Relation of Pending Legislation to the California Petroleum Industry." This address was a strong one and was received with enthusiasm. Owing to its political bearing, it is not appropriate for publication in the *Transactions*, but it has been widely published in the newspapers and magazines. Judge Pierce made some very pertinent remarks upon the same subject.

Ralph J. Reed, Chief Engineer for the Union Oil Co., addressed the meeting on "Problems in Petroleum Transportation."

Among many things of particular interest, Mr. Reed said that, within the past 20 years, the amount of oil transported in California has increased from 1,250,000 bbl. per annum to 93,000,000. With the exception of about 500,000 bbl. that is transported yearly by the railroads, all is carried in pipe lines. There are at present approximately 2,700 miles of main pipe-lines within the State. The cost of transportation ranges from 10 c. to 22½ c. per barrel through pipe lines, against about 38 c. by railroad transportation.

Coalinga to San Francisco by pipe line is 30 c. per bbl.

Coalinga to San Francisco by rail is 40 c. per bbl.

Pumping stations on pipe lines are placed 6 to 12 to 14 miles apart. The oil is heated to 170° at stations by exhaust steam. Pressure on pipe lines is about 800 lb. per sq. in. Average capacity of main lines is 1,000 to 1,200 gal. per hour.

W. W. Orcutt, Chief Geologist for the Union Oil Co., followed with a talk on "The Search for Future Oil Supply," that was of great interest. Apart from his subject, he said that the approximate cost in California of bringing in a well to production is 40 c. per barrel on the oil it will produce.

Ben Linsley, Petroleum Chemist of Taft, Cal., presented a compre-

hensive and interesting paper on the manufacture of gasoline, in which the various processes were described: The production of gasoline by distillation and from casing head gas; by absorption from natural gas, and by the cracking processes known as the Benton and Rittman processes. Dr. Day also spoke at length on the Rittman Process.

After interesting discussions of the various subjects the meeting adjourned.

ALVIN B. CARPENTER, *Sec'y.*

COLUMBIA SECTION

W. H. LINNEY, *Chairman,*

OSCAR LACHMUND, *Vice-Chairman*

LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154,
Spokane, Wash.

STANLY A. EASTON,

S. SHEDD.

Mining camps from Mexico to Alaska and from the Pacific coast to the Atlantic were represented by delegates at the 5th Annual Northwest Mining Convention which was held in Spokane, Wash., Feb. 19-24, 1917. About 300 delegates from outlying regions were present, this being fully 100 more than have been counted at any previous mining convention in Spokane.

Graham B. Dennis, president of the Northwest Mining Association, made the address of welcome. In part, he said:

"Heaven has made us the possessors of an area great enough for an empire, varied enough in climate and possibilities to compass the whole complex field of modern industry. In the development of this broad heritage, we are not only actors for today, but trustees for future ages. . .

"It is to foster this spirit of unity, to create and maintain strong bonds of mutual good will and confidence, that we founded the Northwest Mining Association. We support it because we believe it will give us strength; because we believe we shall learn from each other; because, in industry as in every other walk of life, personal contact and establishment of ties of friendship enrich and round out the characters of men."

One of the interesting lectures of the week was that on "Alaskan and Russian Gravel Mining," by Dr. Henry Mace Payne, consulting engineer for the Russian government and Consolidated Goldfields of South Africa. Dr. Payne's illustrations showed the different appliances in use in the countries mentioned.

The Resolutions Committee reported back to the Convention six of the seven resolutions that had been presented to it. These were adopted as submitted, with some minor changes.

The principal resolution recommended appointment of a committee of representative business and mining men to endeavor to secure location, in or near Spokane, of a modern electrolytic smelter and sampling and testing plant.

The next most important recommendation of the committee was embodied in a resolution requesting the Bureau of Mines to establish a Mines Experiment Station at the University of Idaho, Moscow, to take the place of the branch station recently placed there.

A resolution was also adopted recommending that a government highway be constructed from Stites and Grangeville, Idaho, into Elk City-Dixie-Buffalo Hump region, to provide transportation and encourage

development of the many promising placer gold and quartz properties of these districts.

A resolution was unanimously adopted calling upon the Idaho and Washington legislatures to take State educational institutions out of politics, thus ending controversies that periodically arise between the State College of Washington at Pullman and the University of Washington at Seattle, and the University of Idaho at Moscow and normal schools at different points in Idaho.

Resolutions were also adopted supporting House Bill No. 61, now pending before the Washington legislature, pertaining to mine leases and limiting the liability of companies and individuals owning properties under lease, and commending the work of the State Geological Survey and advocating that the scope of the department be extended to embrace other mineral districts not already given attention.

Mineral and machinery exhibits were not as extensive and varied as at former conventions, but nearly every camp of importance in Washington, Idaho, Montana, Oregon and British Columbia was represented by small but well-selected ore assortments.

As a part of the technical program of the 1917 Northwest Mining Convention, the Columbia Section held a session during the afternoon of Feb. 23.

With about 200 members and guests present, Chairman W. H. Linney called the meeting to order at 2 o'clock, when the following papers were read and discussed:

"Five Years of Progress in the Art of Ore Treatment." By F. A. Thomson.

"Experiments in Recovery of Tungsten and Gold in Murray District, Idaho." By R. R. Goodrich.

"Progress Report of Bunker Hill & Sullivan Smelter at Kellogg, Idaho." By M. H. Sullivan.

The second paper was made additionally interesting by samples of the ores and the progressive products which the author exhibited.

The third paper was illustrated by slides of sketches and plans and was followed by a lengthy discussion by several smelter representatives and mine managers.

No previous meeting has been so well attended, proving the value of the annual Mining Convention and participation of the various engineering societies who control the program, giving each society a period and leaving a day open to general business and discussion.

An executive session followed, at which two committees were appointed: A committee of three to serve on the executive committee of the Spokane Engineering and Technical Association, and a committee of two to act with the Secretary in passing on and suggesting eligible candidates for membership.

L. K. ARMSTRONG, *Secretary*.

AFFILIATED STUDENT SOCIETIES

The annual spring excursion of the College of Mines, University of Washington, is being made to the Coeur d'Alene region of northern Idaho. A party of 15 students and instructors is visiting the Bunker Hill and Sullivan mine, mill and smelter at Kellogg, and several mines and mills in the neighborhood of Wallace and Mullan.

Harrison W. Craver, New Library Director

We are pleased to announce the appointment of Dr. Harrison W. Craver, until recently librarian of the Carnegie Library in Pittsburgh, to be Director of the combined libraries of the American Society of Civil Engineers, the American Institute of Mining Engineers, The American Society of Mechanical Engineers, the American Institute of Electrical Engineers, and the United Engineering Society.



DR. HARRISON W. CRAVER.

Dr. Craver was born in Owaneco, Illinois, August 10, 1875. He was educated in the public schools of Carmi, Illinois, and Terre Haute, Indiana, and the Rose Polytechnic Institute, from which he was graduated in 1895, having specialized in industrial chemistry. The following year he spent in post-graduate chemical study with Dr. W. A. Noyes. In 1896 he became chemist to Kirkpatrick & Co., Limited, of Pittsburgh, and later to

the Shoenberger Steel Co., the Virginia Iron, Coal & Coke Co., and the Duquesne Reduction Co. During this period he acquired a practical knowledge of iron metallurgy, particularly of open-hearth steel manufacture.

In 1900, Dr. Craver first became associated with the Carnegie Library of Pittsburgh, being asked by the librarian, Dr. Edwin H. Anderson, now director of the New York Public Library, to undertake the organization of a technological department. To this work his attention was given until 1908, although interrupted by some returns to metallurgical work. Among these the most important was in 1902, when he became assistant superintendent of the Allegheny Steel Co., then in process of organization, and was placed in charge of their rolling mills. In 1908, he was elected librarian of the Carnegie Library of Pittsburgh, which position he has held since.

Dr. Craver's library experience has been unusually varied. The Technology Department, his first task, was a venture in a field hitherto untried, as no municipal library had attempted to meet the special wants of engineers and manufacturers. The methods of organization and the plan of service adopted have proved the model for the technology departments since inaugurated in most similar institutions in America. He has developed an unusual system of coöperation with the public schools and has been able, through the constant succession of bibliographic publications issued by the Library under his direction, to be of service to libraries and students throughout the world. The bibliographies on engineering subjects have been especially appreciated.

Among librarians his services to the profession have long been recognized. He has been a member of the Council of the American Library Association since 1909, a member of the Executive Board since 1913, and Chairman of the Finance Committee since 1914. At present he is also First Vice-President of the Association.

Dr. Craver has been a member of the Engineers' Society of Western Pennsylvania since 1900. He was for several years a director of the society and chairman of the publication committee. He is also a member of the American Chemical Society and of the American Association for the Advancement of Science.

Now that the libraries of the four Founder Societies are grouped under one roof, an opportunity occurs to mold them into one all-embracing Engineering Library and to render this of service to the greatest degree to the members of the Societies and the profession generally.

Dr. Craver will be given most cordial support by the Library Board to develop his ideas and bring this all-important problem to a successful issue.

LIBRARY

AMERICAN SOCIETY OF CIVIL ENGINEERS
AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

HARRISON W. CRAVER, Director

The Library of the above-named Societies is open from 9 A.M. to 10 P.M., except on holidays. It contains about 70,000 volumes and 90,000 pamphlets, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library, through its Library Service Bureau, can render valuable service through correspondence; letters requesting information will receive especial attention. The Library is prepared to furnish references and photographic copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information on the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

LIBRARY ACCESSIONS**PARTIAL LIST CLASSIFIED BY SUBJECTS****Mining, Metallurgy and Chemistry**

- ANUARIO DE MINERIA, METALURGIA, ELECTRICIDAD Y DEMAS INDUSTRIAS DE ESPANA.** 1916. Madrid, 1916.
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Only a limited number of each of the following papers is available for sale. They are, therefore, offered subject to prior sale.

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MEMBERSHIP

NEW MEMBERS

The following list comprises the names of those persons who became members during the period of Mar. 10, 1917, to Apr. 10, 1917.

- ADKINS, HARVEY S. Adkins & Denham, Engineers, Box 291, Hazard, Ky.
 ANDERSON, ARTHUR P. Leaser, Gibson Copper Co., Box 1978, Miami, Ariz.
 ARNOLD, EDWARD C. Min. Engr., Chief Engr., Federal Lead Co., Flat River, Mo.
 ASKIN, THOMAS B. H., Ball Mill Expert, Inspiration Cons. Copper Co., Miami, Ariz.
 BAIN, C. KREMER. Butte & Superior Mining Co., Butte, Mont.
 BAKER, WILLIAM A., JR., Vice Pres. & Sales Mgr., The Georges Creek-Parker Coal Co., 306 North Gay St., Baltimore, Md.
 BOOTH, GEORGE H. Mech. Engr., Inspiration Cons. Copper Co., Miami, Ariz.
 BRIDGMAN, G. TEMPLE, Asst. Consulting Mining Engr., Guggenheim Bros., 120 Broadway, New York, N. Y.
 BROOKS, CHARLES A. Mine Surveyor, Homestake Min. Co., Lead, So. Dak.
 BUTCHER, FREDERIC E., Secretary, Missouri Metals Corp., 1945 Railway Exchange Bldg., St. Louis, Mo.
 CHETNEY, PETER C., Min. Engr., Moctezuma Copper Co., Pilares-De-Nacozari, Sonora, Mexico.
 COLBERT, CLARENCE L., Supt. of Pachuca Smelter, Cia. de Real del Monte y Pachuca, Pachuca, Hgo., Mexico.
 CURRY, JOSEPH E., Secretary, Ariz. Chapter American Mining Congress, Bisbee, Ariz.
 DENNY, JOHN J. Met., Nipissing Mines, Cobalt, Ont., Canada.
 DIKE, CHARLES F., Mgr., Syndicate-Operating Montreal Mine, P. O. Box 390, Joplin, Mo.
 DOELLE, HENRY E., Supt., Lone Star & Washington Mine, The British Columbia Copper Co., Ltd., Greenwood, B. C., Canada.
 FAIR, FRED A. Pres. & Genl. Mgr., F. A. Fair Association, Boulder, Colo.
 FINUCANE, T. R., Genl. Mgr., The McKinley-Darragh-Savage Mines, Cobalt, Ont., Canada.
 FISHBACK, MARTIN. Mgr., Oriental Gold & Copper Co., Benson, Ariz.
 FORD, EDWARD H., Supt., Mt. Jessup Coal Co., Ltd., Peckville, Lacka. Co., Pa.
 FOX, JESSE B. Transitman, Calumet & Ariz. Min. Co., Box 951, Warren, Ariz.
 GAUL, JOHN C., Foreman, Berkeley Mine, Anaconda Copper Mining Co., Anaconda, Mont.
 GLEASON, VILLEROY, JR., Flotation Foreman, Anaconda Copper Mining Co., Box 842, Anaconda, Mont.
 GRANT, CLIFFORD G., Flotation Engr., Con. Interstate Calahan Min. Co., Wallace, Idaho.
 GRISWOLD, GEORGE G., JR., Min. Engr., Timber Butte Milling Co., Columbia Gardens, Butte, Mont.
 HARVEY, ALEXANDER S. Mine Operator, 610 So. Broadway, Los Angeles, Cal.
 HAWORTH, ERASMUS. Mining Geologist, Haworth & Haworth, Lawrence, Kansas.
 HENNEBACH, E. R. L. Min. Engr., Garfield Smelting Co., Garfield, Utah.
 HILL, JAMES M., Associate Geologist, U. S. Geological Survey, Washington, D. C.
 JACOBS, M. L. Supt. of Quarries, Bethlehem Steel Co., So. Bethlehem, Pa.
 JACOBSON, ROBERT C. Mining Chemist and Engr., P. O. Box 827, Kingman, Ariz.
 JOHNSON, JOHN D., Mine Surveyor, Alaska Treadwell-Mexican-United Gold Mining Companies, Treadwell, Alaska.
 KENNEDY, FRANK A. Civil Engr., John A. Savage & Co., Duluth, Minn.
 KOHLBERG, HARRY L. Box 691, Warren, Ariz.
 LANDON, ROBERT R. Bryan-Landon Co., Inc., Cebu, P. I.
 LAWRIE, GAVIN W. Supt. Concentrator, Balbach Smelt. & Ref. Co., Newark, N. J.
 MCBRIDE, ROY N., Smelter Representative at International Smelter for Miami Copper Co., Box 100, Miami, Ariz.
 McDONNELL, HARRY E., Asst. Supt. 5-10 Blast Furnaces, Ill. Steel Co., So. Wks., South Chicago, Illinois.
 MANDELL, AMBROSE J. 60 Jewel St., Forest Hills, New York, N. Y.
 MARKLE, DONALD. Min. Engr., 321 West Broad St., Hazleton, Pa.
 MECHESENEY, CHARLES A., Engineer, Philadelphia Co., 435 6th Ave., Pittsburgh, Pa.
 MUDD, HARVEY S. 1208 Hollingsworth Bldg., Los Angeles, Cal.
 NICHOLS, HENRY W. Concentrator Met., Magma Copper Co., Superior, Ariz.
 NICHOLSON, GEORGE E. 3512 Locust St., Kansas City, Mo.

- OGDEN, SAMUEL W. Supt., The Grasselli Chemical Co., East Chicago, Ind.
 PARK, HUGH. Mgr., Nipissing Min. Co., Ltd., Cobalt, Ont., Canada.
 PEARSON, JOHN L., Underground Mgr., Wallaroo Mines, Wallaroo & Moonta
 Min. & Smelt. Co., Ltd., Wallaroo Mines, So. Australia.
 RATHJENS, GEORGE W., Vice Pres., Twin City Brick Co.,
 140 W. Summit Ave., St. Paul, Minn.
 REYNOLDS, HENRY D. G. The Ridder Mine, Tomsk Government, Siberia.
 ROBERTS, DAVID J., Ore Buyer, El Paso Smelting Works, 651 Upson Ave.,
 El Paso, Texas.
 ROBINSON, WALTER B. Mine Mgr., Snake River Mines, Huntington, Oregon.
 SANDTNER, FRED, Mine Foreman, Calumet & Ariz. Min. Co., Box 3927, Lowell, Ariz.
 SANFORD, CHARD O. 209 So. Normandie, Los Angeles, Cal.
 SCOTT, JOHN T.; Asst. Chemist, Indiana Laboratories Co., Ruff Bldg., Hammond, Ind.
 SEGALL, JULIUS, Geologist. Hotel Hastings, Minneapolis, Minn.
 SELLERIER, CARLOS. 4A-Carpio-92, Mexico City, Mexico.
 SHERIDAN, LESLIE M., Asst. Chief Engr., Mexican Dept., American Smelt.
 & Ref. Co., 1108 Mills Bldg., El Paso, Texas.
 SHIMMIN, JOHN T. Butte Superior Mining Co., Butte, Mont.
 STEIN, PAUL, Testing Engr., Cons. Kansas City Smelt. & Ref. Co., El Paso, Texas.
 STERK, HARRY J., Foreman Zinc Roasters, Anaconda Copper Min. Co.,
 P. O. Box 926, Anaconda, Mont.
 STUART, QUIN W. Civil and Min. Engr., Edgewater Mine, Ensley, Ala.
 UREN, LESTER C., Instructor in Mining, University of California, Mining Bldg.,
 Berkeley, Cal.
 VAN EMAN, ANDREW G., Assayer, Chemist & Metallurgist, Custom Assay Office
 and Laboratory, 1119 Main St., Boise, Idaho.
 VEATCH, A. C. 7 Richmond Terrace, Whitehall, London, Eng.
 WALKER, ALEXANDER D., Mine Supt., Vindicator Cons. G. M. Co.,
 Independence, Colo.
 WALKER, STANLEY M., Mining Engr., 221-2 First National Bank Bldg., Denver, Colo.
 WATSON, GUY P., Smelter Met., Braden Copper Co., Rancagua, Chile, South America.
 WELLMAN, WALTER H. Supt., Needles Mining & Smelt. Co., Needles, Cal.
 WILLIAMS, BERKELEY, Pres., Virginia Lead & Zinc Corpn., 801 E. Main St.,
 Richmond, Va.
 WILLIAMS, THOMAS M., Min. Engr. Dover, N. J.
 WILSON, THADDEUS C., Flotation Engr., In charge of Flotation,
 Timber Butte Milling Co., Butte, Mont.
 WOODWARD, DON CARROLL, Administrador, Soc. de las Minas de Cobre de San Bartolo,
 San Pedro de Atacama, via Antofagasta, Chile, South America.
 ZWICKY, WILLIAM E., Pres. and Genl. Mgr., Cork-Province Mines, Ltd., Kaslo,
 B. C., Canada.

Associate Members

- BEDDOW, EDWARD W. Box 815, Warren, Ariz.
 GRISHAM, CHARLES A., Asst. Treas., St. Louis Smelt. & Ref. Co.,
 722 Chestnut St., St. Louis, Mo.
 NATHAN, FRED A., Store Mgr., New Cornelia Coöperative Merc. Co., Ajo, Ariz.
 POSKE, HARRY C., American Smelting & Ref. Co., 1108 Mills Bldg., El Paso, Texas.
 TEATS, ROSCOE. 2701 South 13th St., Tacoma, Wash.

Junior Members

- ARMSTRONG, CLIFTON T., Student. 347 Manhattan Ave., New York, N. Y.
 CHENEY, GEORGE M., Student. Colorado School of Mines, Box 398, Golden, Colo.
 DAVY, WILLIAM M., Min. Engr. and Chemist, Andes Exploration Co. of
 Maine, Casilla 290, Arequipa, Peru.
 FREUDENBERG, WALTER H., Student. Missouri School of Mines, Rolla, Mo.
 JAMES, FLOYD D., Student. Missouri School of Mines, Rolla, Mo.
 LYONS, ROBERT P., Student. Missouri School of Mines, Rolla, Mo.
 MILLER, ELTON A. Miner, Wisconsin Zinc Co., Birbeck Mine, Galena, Ill.
 RICE, JAMES W., Student. University of Nevada, Reno, Nevada.
 RODGERS, ALAN M. 31 Mine St., Calumet, Mich.
 SCHMIDT, FREDERICK L., Student. Lehigh University, So. Bethlehem, Pa.
 WEIMER, EARL J., Student. Missouri School of Mines, Rolla, Mo.
 WILEY, HOWARD, Senior, Mining Class of Columbia University, New York, N. Y.

Total Membership, Apr. 10, 1917. 6,186

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

Applications Lacking Endorsement

Applications for membership have been received from Mr. Drewitt, Mr. Lehman and Mr. Trench, whose records are given below. These applications lack the necessary number of endorsers, but since these candidates live at some distance from the headquarters of the Institute, their records are published here in order that any members who are acquainted with them may be advised of the circumstances and may have an opportunity of writing to the Secretary endorsing these candidates.

George Edward Drewitt, Mt. Morgan, Queensland, Aust.

Proposed by A. A. Boyd.

Born 1895–1900, Higher Grade School, Gillingham, England. 1903–09, Electrical and Mechanical Engrg., Gillingham Tech. College (evening). 1909–10, Min. Engrg. course, South Kensington, London, England. 1900–09, Engr. and draftsman, Medway Iron Works, Rochester, England. 1910, Engr., Maschinen-patrik, Lubeck, Germany. 1911–13, Asst., West African Mines. 1913, Reorganizing cement works and coal mines and reporting on properties, The Victorian Portland Cement Co., Richmond, Vict., Aust. 1912–13, Reporting, Pacific Phosphate Co., Ocean Island, Gilbert Group, Central Pacific. 1913–14, Reporting on mines and mills and supt. erection of mills.

Present position: Engrg. Dept., Mt. Morgan Gold Mines Co., Ltd.

Robert Hamilton Trench, Rangoon, Burma, India.

Proposed by A. W. G. Bleck, John Cadman.

Born 1880, Liverpool. 1902, Univ. of Cambridge, England, 2d class Mechanical Science. 1902–04, British Westinghouse Electric & Mfg. Co., Trafford Park, Manchester. 1904–08, Asst. Mgr. and Mgr., London & Pacific Petroleum Co. 1908–12, general reporting and executive work on properties in Russia, West Indies, East Indies, Salina, United States, Messrs. Thompson and Hunter, London.

Present position—1912 to date: Genl. Mgr., British Burmah Petroleum Co.

Alexander Dudley Mackay, Waratah, Tasmania.

Proposed by J. D. Millen.

Born 1888, St. Lawrence, Queensland. 1903–06, Launceston Grammar School. 1907–09, University of Tasmania, B. Sc. 1910–12, University of Melbourne, B. M. E. 1916, Degree of M. M. E. 1912–13, Asst. to chief mining inspector, Mines Dept. of Victoria. 1913–16, Chief Assayer, engaged in research work, Mt. Bischoff Tin Mining Co.

Present position: Chief Assayer, Mt. Bischoff Tin Min. Co.

The following persons have been proposed during the period Mar. 10, 1917, to Apr. 10, 1917, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Ciji Arakawa, Omori Mine, Nima-gun, Shimake-ken, Japan.

Proposed by K. Ide, T. Inouye, B. Katsura.

Born 1887, Aichi-Ken, Japan. 1893-1905, Common School. 1905, Entered the High School at Kyoto. 1908, Entered the Mining and Metallurgical Department of the Kyoto Imperial University. 1911, Graduated from the University. 1911, Employed as a Mining Engr., Fujita Mining Co.,

Present position: Mining Engr., Fujita Mining Co.

Stanley L. Arnot, Plymouth, Amador Co., Cal.

Proposed by S. R. Brown, A. Burch, Arthur B. Foote.

Born 1889, Marklesville, Cal. 1905-09, Eldorado Co., High School, Placerville, Cal. 1909-13, Univ. of Cal. College of Mines, B. S. 1913-15, North Star Mines Co., Grass Valley, Cal. 1915 to date, Plymouth Cons. Gold Mines Ltd., Plymouth, Cal. Present position: Engr., Plymouth Cons. Gold Mines, Ltd.

Verne Wallace Aubel, Newcastle, N. S. W.

Proposed by David Baker, H. F. Noyes, G. D. Delprat.

Born 1888, Mercer Co., Pa. 1902-06, New Castle, Pa., High School, Commercial Course. 1906-10, Penna. State College, Min. & Met. Engrg. 1908, Miner, Golden Cycle Min. Co., Victor, Colo. June 1-Sept. 1, Timberman, Portland Gold Min. Co., Victor, Colo. 1910-12, Stovetender, Blower, Stockhouse Foreman, Lackawanna Steel Co., Buffalo, N. Y. 1912-14, Blower, Night Foreman, Blast Furnace Dept., Jones & Laughlin Steel Co., Woodlawn, Pa. 1914-15, Mgr., Millersburg Fdy. & Mach. Co., Millersburg, Ohio. 1915 to date, Shift Foreman, Blast Furnace Dept., Broken Hill Prop. Co.

Present position: Night Supt., Broken Hill Prop. Co. Iron & Steel Works.

Leon Maxwell Banks, Metcalf, Ariz.

Proposed by W. G. McBride, F. W. McLeon, R. M. Prouty.

Born 1890, Haskel, Tex. 1903-07, M. T. H. S., Denver, Colo. 1907-12, Colo. School of Mines, E. M. 1907-11, Summers; Mining, Mucking, Assaying, etc., San Juan, Colo. Feb. to Dec., 1911, Concrete Foreman, Constructor A. C. Co., Ltd., Concentrator, Morenci. 1912-13, Foreman Moving Mach. A. C. Co., Ltd., Morenci. 1913, Mining A. C. Co., Ltd., Morenci and Coronado. 1913-16, Mine Inspt. A. C. Co. Mines. March to July, 1916, Leasing, Chloride, Ariz. July to Oct., 1916, Supt. Atlas Mine, Ouray, Colo.

Present position: Agent, Grant Leasing Co., Metcalf, Ariz.

Herbert F. Black, Pittsburgh, Pa.

Proposed by William Kelly, W. J. Olcott, F. H. Armstrong.

Born 1879, Pittsburgh, Pa. 1886-97, Pittsburgh Public and Pittsburgh High School. 1911, Special night course, University of Pittsburgh, on Metallurgy of Iron and Steel. 1897-1912, inclusive, Ore Dept. Carnegie Steel Co., Pittsburgh, which dept. handled all the ores, iron, manganese, etc., for the U. S. Steel Corporation. Directly associated with D. G. Kerr, Vice-President U. S. Steel Corp. Also with James Gayley, who formerly held that position. 1913 to present time, In charge of Raw Material Division of Cambria Steel Co., which division now embraces all companies of the Midvale Steel & Ordnance Co.

Present position: Assistant to President, The Midvale Steel Co., Pittsburgh, Pa.

Frank O. Breeding, Bluefields, Nicaragua, C. A.

Proposed by E. E. Carpenter, R. Hawxhurst, Jr., C. Sullivan.

Born 1888, Palouse, Wash. 1907-10, University of Oregon. 1910-12, University of Nevada. 1912, West End Mill, Tonopah, Nev. 1912-14, Rawhide, Nev. 1915, Hilltop, Nev. 1915, Buckhorn, Nev. 1916, San Dimas, Durango, Mexico. 1916, Wonder, Nev. 1917, Bluefields, Nicaragua, C. A.

Present position: Assayer for Eden Mining Co.

Patrick John Brennan, Jenkins, Ky.

Proposed by G. M. Gillette, L. B. Abbott, J. G. Smyth.

Born 1865, Kilkenny, Ireland. 1887-92, Private pupil in Min. Engineering and Surveying under Mr. William Veassly, Mgr., Tyldesley Colliery Co. at Manchester, England; Student at Migan School of Mines, England; Student in Mechanical Engrg. under Mr. C. M. Percy, Migan, England; Passed exam. at Manchester Univ. and secured first class certificate as colliery mgr. 1892-94, Mgr. of Birchenhead Collieries, Ashton in Wakefield, Manchester, Eng. 1895-97, Mgr., of Midland Coal & Coke Co., Stake-on-Trent, Eng. 1898-1906, Mgr., Great Faintin Collieries of the Stafford Coal

& Iron Co., near Stake-on-Trent, Eng. 1906-08, Mgr., Northern Colliery, Hanley, Eng. 1908, Came to U. S. A. and became Supt. of Coketon Plant of Davis Coal & Coke Co.

Present position: Genl. Supt. of the Elkhorn Division, Consolidation Coal Co., Inc.

Thomas F. Breslin, Cebu, Cebu, P. I.

Proposed by H. I. Smith, E. C. Lee, M. E. Wadsworth.

Born 1885, Summit Hill, Pa. 1900, Finished Summit Hill High School at this date.

Graduated, June, 1907, from Pa. State College with degree of B. S. in Mines and Metallurgy. 1906, Summer vacation as Rodman on New York, N. H. & Hartford at New Rochelle, N. Y. 1907-08, Chainman and Rodman on East River division, N. Y. C., for Pa. Tunnel & Terminal Co. 1908-09, Asst. Engr., Lehigh Valley Coal Co. at Wilkes-Barre, Pa. 1909-11, Survey and Chief of Party in Bureau of Lands, Manila, P. I. 1911 to date, In business for myself as Mining Engineer and Surveyor.

Present position: Consulting Engineer.

Jorge Brödermann y Vignier, Havana, Cuba.

Proposed by P. Ortega, J. I. Corral, J. R. Villalon.

Born 1889, Havana, Cuba. 1902-06, Trade School of Havana. 1910, Graduated Civil Engr., National University, Havana, Cuba. 1913, Graduate Architect, National University, Havana, Cuba. 1910-15, 5 years as Civil Engr. in Dept. of Public Works, Havana, Cuba. 1915-16, 1 year with the Dimas Mining & Smelting Syndicate.

Present position:

Richard Henry Brown, Jr., Joplin, Mo.

Proposed by D. M. Armstead, W. E. McCourt, J. D. Robertson.

Born 1881, Tyler, Tex. 1899, Graduated, Tyler, Tex., High School. 1903, Graduated, Virginia Military Institute with 3d stand in Engineering. 1903-08, International Steam Pump Co., New York office and St. Louis office. 1908-14, Reeves & Skinner Mach. Co., St. Louis. 1914, Ingersoll Rand Drill Co., St. Louis Office of Ingersoll Rand Co., N. Y.

Present position: Ingersoll Rand Drill Co., St. Louis, Mo.

George K. Burgess, Washington, D. C.

Proposed by H. L. Smyth, Albert Sauveur, H. M. Boylston.

Born 1874, Newton, Mass. 1888-92, Newton High School. 1892-96, Massachusetts Institute of Tech., S. B. 1898-1900, University of Paris, Sc. D. 1901, Sc. D. 1896-98, Asst. in Physics, M. I. Tech. 1900-01, Instructor in Physics, Univ. of Mich. 1901-03, Instructor in Physics, Univ. of Cal. 1903-13, Asst. Physicist and Associate Physicist, Bureau of Standards. 1913, Physicist and Chief Division of Metallurgy, Bureau of Standards.

Present position: Chief, Division of Metallurgy, Bureau of Standards.

Federico Garcia Caceras, La Fundicion, Peru, S. A.

Proposed by D. F. Hewett, H. G. Ferguson, J. T. Singewald, Jr.

Born 1889, Lima, Peru. 1905-11, Escuela de Ingenieros de Lima, where I completed my studies. 1911-12, Transitman with the North Western Railway of Peru. 1912 to date, Megociacion Minera Fernandini as Asst. Chemist, Surveyor and Met.

Present position: Chemist in charge of the laboratory.

Frederico F. Cardenas, Saltillo, Mexico.

Proposed by E. L. Porch, Jr., W. B. Phillips, S. H. Worrell.

1908, B. S. in Agriculture. 1909, M. of S. at University of Wisconsin, Madison, Wis. 1910-13, Irrigation work at Monclova, Coah. 1914-15, With Southern Seating & Cabinet Co. at Jackson, Tenn. and Dallas, Tex.

Present position: Genl. Mgr. of the Cia Minera Nazareus & Alicaute, S. A.

John L. Clarkson, East St. Louis, Ill.

Proposed by J. D. Robertson, H. S. Gulick, W. E. McCourt.

Born 1889, Kirkville, Iowa. 1907, Graduate of the Albia High School, Albia, Iowa. 1908, One year Liberal Arts, Iowa Univ. 1908-12, four-year mining course, Iowa State College, Ames, Iowa, B. S. in Mining. Sept. 1, 1913-Sept. 1, 1915, Mining Engineer for Superior Coal Co., Gillespie, Ill. Sept. 1, 1915-Sept. 1, 1916, Mine Inspector and Efficiency Engineer for Superior Coal Co. Sept. 1, 1916-Feb. 15, 1917, Asst. to Chief Engineer, Superior Coal Co.

Present position: Assistant to the President and Chief Engineer of the St. Louis and O'Fallon Coal Co.

Joseph Samuel Conpal, New York, N. Y.

Proposed by H. W. Hardinge, Harlowe Hardinge, R. B. T. Kiliani.

Born 1883, Lawrence, Mass. 1903-07, Mass. Inst. of Technology, B. S. in Mining Engineering. 1906, Horseshoe Mining Co., Fay, Nev. 1907, U. S. Mining, Smelting

& Refining Co., Cal. 1908, Treadwell Mines, Alaska, 1909, U. S. Smelt. Min. & Ref. Co., N. J., 1910-12, Providencia Mining Co., Parral, Chihuahua, Mex. 1912-14, Santa Lucia Mining Co., Oaxaca, Mex. 1915 to date, consulting mining work.

Present position: consulting mining work.

Charles Lawrence Dake, Rolla, Mo.

Proposed by A. L. McRae, C. R. Forbes, H. A. Buehler.

Born 1883, Chaseburgh, Wis. 1905-07, State Normal School, River Falls, Wis. 1907-12, University of Wis. 1911, A. B., 1912, A. M. 1910-12, Student Asst. Univ. of Wis. 1912-13, Williams College, Williamstown, Mass. 1913 to date, Missouri School of Mines. Summers: 1908, Cobalt District for C. K. Leith, Madison, Wis.; 1909, Baraboo Iron District, Alvin Rohn, Baraboo, Wis.; 1910, Wisconsin Geological Survey; 1911, Algoma Central Ry., Sault Ste. Marie, Ont.; 1912, Penakee Iron Range, Madison, Wis.; 1913-14-15, Missouri Bureau of Geology and Mines; 1916, Wyoming Geological Survey.

Present position: Asst. Prof. of Geology, Missouri School of Mines.

George E. Dalbey, Omaha, Neb.

Proposed by W. T. Page, J. O. Betterton, W. E. McCourt.

Born 1886, Chicago, Ill. 1908-10, 3 years in the Kansas City University Medical Dept. 1910 to date, American Smelt. & Refining Co., Omaha, 5 years as Analytical Chemist, 2 years as Electrometallurgist.

Present position: Metallurgist.

Thomas L. Davey, Bluefields, Nicaragua, Central America.

Proposed by E. E. Carpenter, C. Sullivan, R. Hawxhurst, Jr.

Born 1881, Virginia City, Nev. 1898, High School, San Jose, Cal. 1903, I. C. S. M. M. course. 1899, Mucker, carman and carchecker, Utica Mine, Angels Camp, Cal. 1901-02, Carman, cage tender, miner, Elkhorn Mine, Elkhorn, Mont. 1902-04, Miner, timberman, shift-boss, "Original," "Stewart," "Anaconda," "Lexington" Mines, Butte, Mont. 1904-06, Miner, timberman, shift-boss, De Lamar Mines, De Lamar, Idaho. 1906-07, Timberman, shift-boss, Portland, Goldcoin Mines, Victor, Colo. 1907-08, Underground foreman, Ashante Goldfields, West Africa. 1908-09, Night foreman, Yampa Mine, Bingham Canyon, Utah. 1909-14, Mine foreman with A. S. & R. Co. at various mines in Mexico. 1914-15, Mine Supt., Yellow Jacket, Belcher & Crown Point Mines, Virginia City, Nev. 1915-17, Mine foreman, Eden Mining Co., Bluefields, Nicaragua, Central America.

Present position: Mine Foreman, Eden Mining Co.

Franklin Kaercher Day, Fairmont, W. Va.

Proposed by Frank A. Ray, J. P. Hutchins, W. E. Thorne.

Born 1881, Mt. Pleasant, Pa. 1899, Graduate of Hazelton, Pa., High Schools. 1899-1903, Graduate of Lafayette College, Easton, Pa., C. E. Degree. 1903-04, Great Northern Rail Road Co., Construction work between Columbia Falls and Rexford, Mont. Jan. to Apr., 1905, Estate of A. S. Van Wickle (Min. Engr.), Coleraine & Evans Collieries, near Hazelton, Pa. April, 1905-1906 Min. Engr., Philadelphia & Reading Coal & Iron Co., Ashland, Pa. May, 1906, to Sept., 1916, Division Engr., The Consolidation Coal Co., Fairmont, W. Va.

Present position—Sept., 1916 to date: Min. Engr., The American International Corp., Petrograd, Russia.

Dean Reid Dove, Magna, Utah.

Proposed by R. C. Gemmell, R. H. Hawley, R. A. Conrads.

Born 1890, Topeka, Kansas. 1896-1904, Grade Schools, Denver, Colo., Joliet, Ill., and Topeka, Kan., 1904-09, Manual Training H. S., Topeka, Kan., Manual Training H. S., Denver, Colo.; Westport H. S., Kansas City, Mo. 1909, Entered Colo. School of Mines. 1913, Graduated from Colo. School of Mines, E. M. July, 1907 to Sept., 1908, Colo. and Southern Ry. Co. Engineering Dept. Summer, 1909, Same as above. Summer 1910, summer 1911, Union Leasing Co., Cripple Creek, Colo., mucker, trammer, helping timber and helping on a machine. May, 1913, to date, Laboratory, Magna Plant, Utah Copper Co., Garfield, Utah.

Present position: Chief Chemist.

John F. Dugan, Butte, Mont.

Proposed by G. M. Fowler, M. H. Gidel, F. A. Linforth.

Born 1885, St. John, N. B., Canada. 1906, E. M. Degree from Montana State School of Mines. 1906-14, Engr. for Anaconda Copper Mining Co. 1914-17, Foreman for Anaconda Copper Mining Co.

Present position: Foreman, Nettie Mine, Butte, Mont.

Benjamin Anton Enebo, Hurley, N. M.

Proposed by D. C. Jackling, J. M. Sulley, H. G. S. Anderson.

Born 1879, Norway. 1896-1900, Machinist Apprentice at Adals Brug Hede-marken, Norway. 1900-02, Skiens Fjordens Mekaniske Fagskole Porsgrund, Norway. 1904-05, Machinist, Utah Copper Co., Bingham, Utah. 1905-06, Draftsman, Union Iron Works, San Francisco, Cal. 1906-09, Draftsman, Utah Copper Co., Salt Lake City, Utah. 1909-13, Machinist, Mill-operator and Draftsman, Steptoe Valley Smelting & Mining Co., McGill, Nev. 1913-15, Draftsman, Utah Copper Co., Salt Lake City, Utah. 1915-17, Construction Engr., Chino Copper Co., Hurley, N. M.

Present position: Construction Engr.

Thorold F. Field, Duluth, Minn.

Proposed by W. Gohring, J. C. Greenway, E. C. Congdon.

Born 1884, Dobbs Ferry, N. Y. 1888-93, Private lessons. 1893-98, Montclair, N. J., Public Schools. 1898-1909, The Hill School, Pottstown, Pa. 1901-05, Univ. of Minnesota, School of Mines, Degree E. M. 1911-12, Graduate work at Harvard. Summer, 1900, Laborer in Daly West Mine, Park City, Utah. May to Dec., 1905, Employed by F. W. Smith, Bisbee, Ariz. 1905-1909, Geological Dept., Calumet & Ariz. Min. Co., Bisbee, Ariz. 1909-16, Mining Engr., employed by Chester A. Congdon, Duluth, Minn. 1916 to date, Mining Engr., employed by Mr. Congdon's Estate.

Present position: Mining Engr. in employ of Estate of Chester A. Congdon.

Umekichi Fukazawa, Los Angeles, Cal.

Proposed by G. H. Clevenger, D. M. Folsom, H. W. Young.

Born 1890, 65 Terai Gumma, Japan. 1908, Graduate of Gumma High School, Gumma, Japan. 1911, Graduate of Sendai Tech. College in Met. with degree of Togugioshi. 1916, Graduate in Met. at Leland Stanford University, with degree of A. B. 1911-13, Research Met. for Fuzita Mining Co., in the Kioto Imperial Univ., Osaka, Japan. 1913-14, Asst. Instructor in Kioto Imperial Univ., Osaka, Japan.

Present position: With Furukawa Mining Co., Korea.

William Gibson, Port Pirie, South Australia.

Proposed by G. C. Riddell, J. G. Poage, John Jobson.

Born 1875, Dunedin, New Zealand. 1901-04, Otago Univ. School of Mines, Dunedin, New Zealand. Associate courses in Mining, Metallurgy, Geology and Assaying. Associate in Mining, Otago School of Mines. Bachelor of Science of Mining, Engineering University of New Zealand. 1905-June, 1906, Assaying and Refining, Waihi Gold Mining Co., Ltd., New Zealand. June-July, 1906, Battery Supt., Omaha Gold Mining Co., Ltd., Thames Dist., New Zealand. July, 1906-June, 1907, Amalgamated Waihi Grand Junction Gold Co., Ltd., New Zealand. 1907-May, 1911, Director, School of Mines, Waikino, N. Z. 1911-May, 1914, Director, School of Mines, Karangahake, New Zealand. May, 1914 to Nov., 1916, Asst. Geologist, New Zealand Geological Survey. 1916 to date, Metallurgical Engineering, Broken Hill Associated Smelters Proprietary, Ltd., Port Pirie, South Australia.

Present position: Metallurgical Engineering, Broken Hill Associated Smelters Proprietary Ltd., Port Pirie, South Australia.

Willard S. Girvin, Waterbury, Conn.

Proposed by W. H. Bassett, D. R. Hull, G. C. Stone.

Born 1893, Buffalo, N. Y. 1912-16, University of Michigan, B. Ch. Eng. 1916 to date, Asst. Metallurgist, American Brass Co., Waterbury, Conn.

Present position: Asst. Met., American Brass Co.

L. B. Grindlay, Youngstown, Ohio.

Proposed by J. W. Deetrick, J. N. Reese, J. E. Perry.

Born 1878, Warren, Pa. 1898, Warren High School. 1902, Mechanical Engineering, Pa. State College. 1902-04, Draftsman, Cambria Steel Co. & Pa. R. R. Co. 1904-07, Special Apprentice, Nor. Pac. R. R. 1907-08, Testing Dept., Pa. R. R. 1908 to 10, Inspecting and Experimental Work, New York, New Haven & Hartford R. R. 1910-13, Inspector and Engr. of Tests, Open Hearth Works, Republic Iron & Steel Co. 1913 to date, Inspecting Engr., Republic Iron & Steel Co.

Present position: Inspecting Engineer, Republic Iron & Steel Co.

Esdrae Ludwick Gruver, New York, N. Y.

Proposed by S. H. Ball, A. H. Rogers, E. N. Skinner.

Born 1879, Reliance, Va. 1898, Shenandoah Normal College Graduate, Scientific Course, Reliance, Va. 1904-17, Secretary to and associated with A. Chester Beatty. 1916-17, Vice Pres. Esperanza Min. Co.

Present position: With A. Chester Beatty and Vice Pres. Esperanza Min. Co.

Cyril Wayth Gudgeon, Waratah, Tasmania.

Proposed by C. E. Coote, R. P. Roberts, L. V. Waterhouse.

Born 1884, Cromwell, N. Z. 1904, Graduate Scranton School of Mines, Mining. 1908, Graduate Scranton School of Mines, Met. 1909, Special course, Otago Univ. School of Mines. 1910, Transvaal, Mine Manager's Certificate, by exam. 1912, Tasmanian Mine Manager's Certificate, by exam. 1913, New Zealand, Mine Manager's Certificate, by exam. 1905-09, Practical experience mining work in various New Zealand Mines. 1909-10, Mine Sampler, Progress Mine, New Zealand. 1910-11, Mill Mgr., Celene Mill, Transvaal. 1910-11, Shift Boss, Simmer & Jack. 1911-12, Chief Surveyor & Draughtsman, Tasmania G. M. Co., Tasmania. 1912-14, Genl. Mgr., The Gold & Schultze Propy Co., Ltd., New Zealand.

Present position: Genl. Mgr., Mr. Birchhoff extended Tin Min. Co.

Gerhard William Gunderson, Crosby, Minn.

Proposed by W. Wearne, R. L. Wahl, B. A. Mizen.

Born 1889, Duluth, Minn. 1910, Univ. of Wisconsin Academic course. 1911-12, Louisiana State Univ. Special Course in Chem. Mechanical Engrg. and Mathematics. Three years in college but no degree. 1909-10, Clerk, Inland Steel Co., Hibbing, Minn. 1913-14, Chemist, Oliver Iron Mining Co. at Hibbing, Minn. 1915-16, Chief Chemist and Foreman, Iron Ore Concentrator for Inland Steel Co. at Crosby, Minn.

Present position: Mine Foreman and Chemist, Inland Steel Co.

R. F. Harrington, South Boston, Mass.

Proposed by Frank W. Durkee, A. C. Lane, W. M. Sanders.

Born 1890, Chicago, Ill. 1909-13, Tufts College, Medford, Mass., B. S. in Chemistry. 1913-17, Employed as chemist and metallurgist by the Hunt-Spiller Manufacturing Corp. and engaged in both control and research work along metallurgical lines.

Present position: Metallurgist and Chemist.

George Mitchell Henderson, Weston, Mass.

Proposed by B. A. Robinson, J. H. Polhemus, A. P. Rogers.

Born 1882, Newton, Mass. 1902, Newton High School. 1906, Mass. Inst. Tech. 1906, S. B., 1906, Engr., Oliver Iron Min. Co. 1907, Engr., Tramp Cons. Min. Co., 1908, Homestake Slimes Plant. 1909-11, Engr., Ray Cons. Copper Co. 1912, Engr., Arizona Copper Co. 1913-14, Ray Cons. Copper Co. 1915-16, Examination work in South America.

Present position: Consulting Mining Engr.

Earl Charles Henry, Mineville, N. Y.

Proposed by H. Comstock, George C. Foote, A. E. Hodgkins.

Born 1890, Port Henry, N. Y. 1903-07, Port Henry High School. 1907-11, Rensselaer Polytechnic Institute, Degree of Civil Engineer. Dec., 1911 to May, 1913, Engineer, Witherbee, Sherman & Co. May, 1913, to Dec., 1916, Supt. of Harmony Mine, Witherbee, Sherman & Co. Dec. 1, 1916 to date, Chief Engineer, Witherbee, Sherman & Co.

Present position: Chief Engineer, Witherbee, Sherman & Co.

William L. Hogg, Crimora, Va.

Proposed by G. A. Packard, C. E. Locke, C. R. Hayward.

Born 1883, Boston, Mass. Morgan Academy. 1902-06, University of Glasgow, Scotland. 1908-10, Utah Copper, Copperton Plant. 1910-11, Tonopah Min. Co. 1911, Tonopah Extension, Tonoaph, Nev. 1912, Guanajuato Development Co. 1912-15, Boston-Mexico Mining Co. 1915, Crimora Manganese Corp.

Present position: Chief Engr. and Genl. Mgr., Crimora Manganese Corp.

Rush Norman Hosler, Harrisburg, Pa.

Proposed by E. W. Hess, Walter Fahringer, H. M. Kanarr.

Born 1874, Fishing Creek, Pa. 1896, Graduate of Bloomsburg State Normal, Bloomsburg, Pa. 1897-98, On preliminary and locating survey on railroad. 1898-1900, Transimtan for Lehigh & Wilkes-Barre Coal Co., Audenrie, Pa. 1900-02, Construction Engineer, Okla. McAllister Coal Co. and Wilburton Coal Co. 1902-12, Chief Engineer for Morrisdale Coal Co., Morrisdale, Clearfield Co., Pa. 1912-14, Chief Engineer for Rochester & Pbg. Coal & Iron Co., Jefferson & Clearfield Coal & Iron Co., Pittsburg Gas Coal Co. 1914-16, Developing Engineer and Supt. for Davis Coal & Coke Co., Thomas and Elkins, W. Va.

Present position: Supervisor of Coal Mine Schedule Rating for State of Pennsylvania.

Joseph C. Houston, So. Porcupine, Ont., Can.

Proposed by H. P. DePencier, C. E. Rodgers, C. D. Kaeding.

Born 1873, Londonderry, Ireland. 1897-1902, Different positions at different mines in the West. 1902-05, Examination work on Iron. 1905-06, Mgr., O'Brien Mine, Cobalt. 1906-09, Mgr., Right of Way Mine, Cobalt. 1909, Mgr. Bonsal Mine, Gowganda. Sept., 1909 to July, 1910, Examination work in Mex. and Canada. 1910, Examination work in British Columbia. Apr., 1911 to Aug., 1912, Mgr. St. Anthony Mines, West Ontario, Canada. Bal. of 1912, Examination work in Ontario. Jan., 1912 to Jan., 1916, Mgr. Schumacher Gold Mines, Porcupine.

Present position—1916 to date: General Superintendent, Dome Mines Co., Ltd.

Victor H. Hughes, Tulsa, Okla.

Proposed by F. J. Fohs, V. H. McNutt, W. E. McCourt.

Born 1885, Fort Scott, Kans. 1908, Degree B. S. in Mine Engineering, Missouri School of Mines. 1912, Degree Engineer of Mines, Missouri School of Mines. 1908-14, with Missouri Geological Survey. 1912-14, Asst. State Geologist of Missouri. 1914 to date, Member of firm of Valerius, McNutt & Hughes, Consulting Geologists, Tulsa, Okla.

Present position: Consulting Geologist.

Herbert Alexander Kursell, Zmeinogorsk, Gt. Tomsk, Siberia.

Proposed by J. P. Hutchins, R. M. Geppert, F. A. Ray.

Born 1886, Revel, Russia. 1906, finished the Gymnasium of St. Ann's in Petrograd. 1906-07, Served in the Life Guard Dragoons, Officer, 1907. 1907-11, Engineering School in Riga, Russia. Mining Academy in Freiberg, Germany. Acquired degrees of Mining and Metallurgical Engineer, 1911. 1915, Acquired the rights of Russian Mining Eng. in Tomsk, Siberia. 1911-14, Employed by the Société Anversoise pour la recherche des mines au Katanga—a company formed by de Barry, Vandervelde Anverse, the Metallbank, Frankfort—first as engineer, later as manager of a prospecting expedition to the Belgian-Kongo. 1914-17, Employed by the Russian Mining Corporation Ltd. 1914, Assistant Geologist for H. W. Turner of San Francisco. 1915, Geologist for Altai concessions. 1916, Manager of the Zmeinogorsk concession.

Present position: Manager of the Zmeinogorsk Mines.

T. W. Lloyd, Birmingham, Ala.

Proposed by James Bonnyman, Robert Hamilton, E. E. Ellis.

Born 1885, Jefferson County, Ala. 1905-07, special mining engr. work at Ala. Polytechnic Institute, Auburn, Ala. 1907-08, Routine engineering work, T. C. I. & R. R. Co., Pratt City. Summer and fall 1908, looking after diamond drill operations of T. C. I. & R. R. Co. Winter, 1908-09, preliminary work for new plants at Blue Creek, T. C. I. & R. R. Co. 1909-10, routine engineering work at Pratt Mines of T. C. I. & R. R. Co. July, 1910 to date, in charge of coal leasing operations of T. C. I. & R. R. Co.

Present position: Asst. engr. in charge of coal leasing operations of T. C. I. & R. R. Co.

Glenn D. Macy, Copper Mt., B. C., Can.

Proposed by L. Lachmund, F. S. Norcross, Jr., R. G. Holmes.

Born 1889, Yorkshire, Ohio. Public School, Dayton, Ohio. 1902-06, Steele High School, Dayton, Ohio. 1906-07-08-09, Ohio State Univ., Columbus, Ohio. Summer months, 1904-05-06-07-08, Genl. Construction work with R. E. Kline & Co., Dayton, Ohio., Santa Fe Ry., Amarillo, Texas, and State Surveyor, Amarillo, Texas, as rodman, instrument man, draughtsman and computer. 1909-12, Asst. Engr., Columbus Gas and Fuel Co., Columbus, Ohio. 1912-14, Resident and Track Engr., Kettle Valley Ry., Penticton, B. C. 1914-15, Private practice, Oroville, Wash., Mineral Surveying and genl. min. work. 1915, With British Columbia Copper Co., Copper Mt., B. C.

Present position: Chief Engr., British Columbia Copper Co.

John P. Meyer, St. Louis, Mo.

Proposed by W. E. McCourt, E. J. Walsh, C. G. Ewing.

Born 1864, St. Louis, Mo. 1877-80, Washington Univ., St. Louis, Mo. 1914, Pres. Philipsburg Min. Co., Philipsburg, Mont. 1916, Pres. Granite-Bi-Metallic Cons. Mining Co., Philipsburg, Mont.

Present position: Pres., Granite-Bi-Metallic Cons. Mining Co.

James Macdonald Mitchell, Flushing, N. Y.

Proposed by J. S. Shedden, H. N. Spicer, Dan S. McAfee.

Born 1875, Flushing, N. Y. 1895, Columbia, B. A. 1909, Mich. College of Mines, M. E., B. S. 1903-04, United Verde Copper Co., 1906-07, Houghton Alaska Exploration Co. 1909-10, Chino Copper Co. 1910-11, East Canada Smelting Co. 1911-12, Canadian Copper Co. 1912-14, Sage Foundation House Co. 1915-16, Homestake Min. Co. 1916-17, St. Nicholas Zinc Co.

Present position: St. Nicholas Zinc Co.

Charles Woodman Morse, Anaconda, Mont.

Proposed by Frederick Laist, Louis V. Bender, Albert E. Wiggin.

Born 1874, Grass Valley, Cal. 1897, Univ. of Cal., B. S. Draftsman, Henshaw Bulky Co., San Francisco. Assayer and Asst. Supt., Victor Gold Mines. Supt. of the Dewey Mine, Grangeville, Idaho, for Massiner & Dickinson, Seattle, Wash. Mine Examination work in California, Mexico and Charge of American Mine, French Gulch, Shasta Co., Cal. El Tivo Mine, Sonora, Mexico, for Horace Chase & Wm. Fors, San Francisco. Technical Division, E. I. Du Pont. Co., Wilmington, Del. Land Classification work for the U. S. Govt.

Present position: Concentrating Engineer, Washoe Smelter.

Neal M. Muir, San Francisco, Cal.

Proposed by J. E. McIntyre, F. H. Probert, L. C. Uren.

Born 1883, Helena, Mont. 1899-1903, Central High School, St. Paul, Minn. 1903, University of Minnesota. 1904-05-07, Mich. College of Mines, B. S. 1906, Asst. Engr., Gt. Northern R. R., Garneil, Mont. 1907, Asst. Supt. Tunnel Construction, A. Guthrie & Co., Lavina, Mont. 1908, Min. Engr., Mint Gold & Silver Min. Co., Pearl, Ida. 1909, Assayer, Utah Apex Min. Co., Bingham Canyon, Utah. 1910-11, Foreman, Chemist, Supt. Utah Leasing Co., Bingham Canyon, Utah. 1912, Asst. Supt., San Antonio Copper Co., San Antonio de la Huerta, Sonora, Mex. 1913, Examination work in California, Oregon, and Arizona for the United States Min., Smelt. & Ref. Co. and other interests. Supt., Ariz. Belmont Min. Co., Silverbell, Ariz. 1914, Supt. Genesee Valley Copper Co., Genesee, Plumas Co., Cal. 1915, Examination work in Nevada for R. E. Cranston and other interests of San Francisco. 1916, Examination work, and Supt. Starlight, Faxie & Jarbidge Gold Mines, for Hanford Interests at Jarbidge, Nev.

Present position: Min. Engr., Genl. Practice.

Alfred C. North, Bluefields, Nic., C. A.

Proposed by E. E. Carpenter, C. Sullivan, R. Hawxhurst, Jr.

Born 1884, Riverside, Cal. 1902, Riverside High School. 1910, University of Cal., College of Mining, B. S. 1905-09, Off and on summer vacations, etc., and one full year as shoveler, timberman, specimen and blasting boss, Empire Mine, Grass Valley, Cal. 1909-10, Surveyor for W. Y. Richardson, Tonopah, Nev. 1911-13, sampler, shift-boss, surveyor for Butters Davisadero Co., La Union, Salvador, C. A. 1914-15, Mine Foreman, San Luis Mg. Co., San Dimas, Durango, Mex. 1915-16, Mine Supt., Bright Star Mine, Shasta, Cal. 1916, Mining Engr., with Eden Mg. Co., Nic., C. A.

Present position: Mining Engr., Eden Mining Co.

Torgus Hans Oaas, Butte, Mont.

Proposed by C. W. Goodale, John Gillie, B. H. Dunshee.

Born 1888, Romsdalen, Norway, Europe. 1895-1903, Public Schools, Merrill, Wis. 1903-07, High School, Merrill, Wis. 1907-08, Bergen Trade School, Bergen, Norway. 1909-10, High School, Butte, Mont. 1910-12, Notre Dame University as mining student. 1908-09, Machinist for Ordalen Grube Atkie Bolog Lökkens Verk. 1912, 6 mon. chem. at Ill. Steel Co.'s plant at Gary, Ind. 1912-13, Miner, Anaconda Mining Co., Butte. 1913, 6 months, Boss Trammer, Anaconda Min. Co. 1913-16, Shift Boss, Anaconda Min. Co. 1916, Asst. Safety Engr., Anaconda Min. Co. 1917, Safety Engineer, Anaconda Min. Co.

Present position: Safety Engr., Anaconda Copper Mining Co.

Edward L. Ohnsorg, Federal, Ill.

Proposed by Rudolph Porter, Herman Garlichs, W. E. Newnam.

Born 1870, St. Louis, Ill. 1888, Grad. St. Louis High School. 1888-92, Special instruction under J. P. Gazzam in genl. chemistry and met. 1888-90, Asst. Chemist, St. Joseph Lead Co. 1890-91, In control of the parting plant and electrolytic copper refinery of the St. Louis Smelting & Ref. Co. 1891-92, Genl. Foreman of the St. Louis Smelt. & Ref. Co. 1892-99, Chemist and Met., St. Joseph Lead Co.'s Smelter. 1899-1904, In control of St. Joseph Lead Co.'s Smelter. 1904-16, Chemist and Met., Federal Lead Co., Federal, Ill. 1916-17, Supt., Federal Lead Co.

Present position: Supt., Federal Lead Co.

James Eugene O'Rourke, Carterville, Mo.

Proposed by H. I. Young, J. N. Houser, H. B. Henegar.

Born 1882, West Rutland, Vt. 1897, Three years in West Rutland, Vt., High School. 1897-1903, With Vt. Marble Co. at West Rutland, Vt., as blacksmith apprentice, channeler operator and machinist apprentice. Mar., 1903-Dec., 1903, Machinist with Rutland Florence Marble Co., at Fowler, Vt. Dec., 1903-Feb., 1905, Master Mechanic with Norcross-West Marble Co., at Dorset, Vt. Feb., 1905-Sept., 1905, Master Mechanic with S. Pearson & Son, Contractors, East River Tunnel of Pennsylvania R. R. Sept., 1905-Feb., 1915, Rock drill and channeler demonstrator, Sales Engineer and Office Manager with Sullivan Machinery Co. of Chicago, Ill. Sept., 1905 to date, Eff. Engineer with American Zinc, Lead & Smelting Co., St. Louis, Mo.

Present position: Eff. Engineer with American Zinc, Lead & Smelting Co.

John B. Orynski, Bluefields, Nic., C. A.

Proposed by E. E. Carpenter, R. Hawxhurst, Jr., C. Sullivan.

Born 1886, San Antonio, Tex. 1914, B. S. in College of Mining, University of Cal. 1908-09, Refinery, El Progreso Mining Co., El Triunfo, B. C., Mex. 1914, 6 mos. as rodman and utility man with U. S. G. S.; Yosemite Valley. 1915, Prospecting in Baja, Cal., Mexico. 1912, 3 mos., millman, amalgamator with the Pittsburgh Silver Peak Gold & S. Min. Co., Blair, Nev. 1917, Refineryman with Eden Mining Co., Bluefields, Nic.

Present position: Eden Mining Co.

William C. Peyton, New York, N. Y.

Proposed by J. H. Janeway, G. C. Stone, M. F. Chase.

Born 1868, California. 1887, University of California, B. S. 1887-99, Asst. Supt. of the Santa Cruz plant of the California Powder Works. In charge of the manufacture of black and military powder. 1889-1911, Pres. of Peyton Chemical Co. in Cal. 1914-17, Practising engineering in association with A. M. Hune.

Present position: Practising Engr. in association with A. M. Hune.

Arthur C. Pope, Tiltonville, Ohio.

Proposed by J. E. Thomas, G. S. Brooks, L. G. Duncan.

Born 1890, Baraboo, Wis. 1912, B. S. in Chemical Engr., Univ. of Wisconsin. 1912-16, Dept. Asst., Mineral Point Zinc Co., De Pue, Ill.

Present position: Supt. Tiltonville Plant, Prime Western Spelter Co.

Donald A. Powell, Columbus, Ohio.

Proposed by J. W. Maller, E. J. Hall, Wm. Campbell.

Born 1890, Columbus, Ohio. 1896-1903, Public Schools of Columbus. 1903-07, High School of Columbus. 1907-1910, School of Mines of Ohio State University. 1910-11, Engineering Staff, Colorado Fuel & Iron Co., Trinidad, Colo. 1911-12, Engr. Staff of the Madera Co., Ltd. and the Mexico N. W. R. R. at Madera, Chihuahua. 1912-13, Draftsman, Jeffrey Mfg. Co., Columbus, Ohio. Feb.-July, 1913, Asst. Engr., Tezuitlan Copper Co., Pueblo, Mexico. 1913-14, Shift boss, El Oro Mining & Ry. Co., El Oro, Mexico. 1914-15, Engr., Cinco Mines Co. 1915-16, Genl. consulting office in Guadalajara, Mexico.

Present position: Supt. of Construction Sea Wall Case Coast Protection Corporation.

A. E. Remington, Shafter, Tex.

Proposed by H. W. Young, W. F. Dietrich, G. H. Clevenger.

Born 1886, Watertown, N. Y. Graduated from the Watertown High School. 1913, Graduated Stanford Univ. (mining and Geol. Dept.) Degree of A. B. Underground work at Bunker Hill and Sullivan and Last Chance Mines, Idaho, and Ludwig Copper Mine, Nev. May, 1913-Dec., 1913, Esmeralda Option of Babilonia Mine, Nicaragua, sampler and assayer. Jan., 1914-June, 1914, Cyanide man at Leonessa Mill, Matagalpa, Nicaragua. Includes mapping, sampling, reports, ore reserves, mine contracts, and at times assaying.

Present position: Mining Engineer with the Presidio Mine.

John Rhodes Roberts, Minneapolis, Minn.

Proposed by C. S. Longyear, H. M. Roberts, J. E. Hodge.

Born 1888, Superior, Wis. Common, High School. 1913-14-15, Special student in Geology, Univ. of Wisconsin. 1908-12, Magnetic Surveys, Cuyuna Range (Private Parties). 1913-14, Wisconsin Geological Survey. 1915, Western Copper Camps. 1916, Employed by John E. Andrus (capitalist).

Present position: In charge mineral lands of John E. Andrus.

Orson F. Riser, Hurley, N. M.

Proposed by D. C. Jackling, J. M. Sully, H. G. S. Anderson.

Born 1882, Salt Lake City, Utah. Public school, High school and 2 years University of Utah. 1900-02, Cons. Mercur Gold Mines Co., Mercur, Utah, Assay Office and Refinery. 1903-04, De Lamar Gold Mining Co., De Lamar, Nev., Foreman of one shift cyanide and of mill. 1905, Electrician and repairman, Utah Copper Mill, Bingham, Utah. 1906, Mill operator, Newhouse, Utah. 1907, Engineer Power Plant, Utah Copper, Bingham, Utah. 1908-09, Engineer and Asst. Chief Engr., Utah Copper Co., Garfield, Utah. 1910, Shift Foreman, Utah Copper Mill, Bingham, Utah. 1911, Engr., International Smelting Co., Tooele, Utah. 1912-17, Fine Crushing Foreman, Concentrating Foreman, General Foreman, Chino Copper Co.,

Present position: Genl. Mill Operating Foreman, Chino Copper Co.

Tony Harry Rosenkranz, Butte, Mont.

Proposed by Frank A. Linforth, Reno H. Sales, Earl B. Young.

Born 1892, Dorchester, Wis. 1909-14, Student at Wash. State College, geology. 1914, B. S. degree in geology from Wash. State College. 1914-15, Three months post-graduate work in geology, Wash. State College. 1913-16, During summers, field assistant for U. S. Geological Survey under J. T. Pardee. 1915, Aug.-June, 1916, Asst. Testing Engr. for Anaconda Copper Mining Co. Aug., 1916 to date, Asst. Geologist for Anaconda Copper Mining Co.

Present position: Assistant geologist.

Nathaniel Harry Ruby, Portland, Ore.

Proposed by Chas. Butters, T. A. Rickard, F. D. Pagliuchi.

Born 1887, Austria. 1906, Univ. of Chicago. 1909, Mich. College of Mines, E. M. 1910, Miner, Calumet & Arizona Mining Co. 1911-13, Assayer, sampler, surveyor, American Smelting & Refining Co., American Smelters Securities Co. in Mexico. 1914, Supt., San Francisco de Bornia Mining Co. Velardina, Dgo., Mex. 1914, Engineer, Butters Salvador Mines Ltd., Salvador.

Present position: Engineer, Butters Salvador Mines Ltd.

Julian Schueler, Peoria, Ill.

Proposed by H. A. Brassert, P. A. Newton, Walther Mathesius.

Born 1889, Detroit, Mich. 1903-07, Bradley Polytechnic Inst., Peoria, Ill. 1907-11, Univ. of Ill., Urbana, Ill. 1908-11 (summers), Univ. of Ill. for Dr. Grindley. 1911 to present time, Keystone Steel & Wire Co., Peoria, Ill.

Present position: Chemist, Keystone Steel & Wire Co.

Clifford Ernest Covell Smith, Toronto, Ont., Can.

Proposed by Philip N. Moore, Arthur Thacher, H. A. Wheeler.

Born 1871, Fairfield, Ont., Can. 1892, Matriculated from Brockville Collegiate Institute, Brookville, Can. 1893-98, Attended Queens Univ. at Kingston, Can. Registered in Arts and School of Mining and College of Applied Science for five years. 1914, Was appointed to the Board of Governors of above school of Mining and College of Applied Science for term of 1914-18. 1916, Was elected a Councillor of Queens Univ. term of 1916-22, as representative of graduates. Am a member of Canadian Mining Institute and am serving third term of election as a Councillor. 1915, Made a member of the Institution of Mining & Metallurgy, London, Eng. 1915, After various mining engineering work, I was appointed, in 1915, Canadian mining engineer of The New Jersey Zinc Co. This particular Canadian work was finished in 1908 but connection has continued in other Canadian interests.

Present position—1914 to date: Consulting Engineer and Assistant to the President of the Coniagas Mines, Ltd., St. Catherines. Manager of all extension work of that company.

Charles T. Smith, Fredericktown, Mo.

Proposed by Ambrose E. Ring, Howard G. Washburn, J. Frank Thompson.

Born 1876, Glendale, Ky. 1892-95, Vanderbilt Training School, Elkton, Ky. 1895-98, Davis Military School, Winston, Salem, N. C. 1904-15, El Oro Mining & Railway Co., El Oro, Mexico, Cashier, Accountant and Acting Business Manager.

Present position—1916 to date: Secretary and Treasurer, Missouri Cobalt Co.

Frederic R. Speed, Catonsville, Md.

Proposed by Howard Eckfeldt, Benj. L. Miller, Jos. W. Richards.

Born 1890, Baltimore, Md. To 1908, Private School education at Boys Latin School, Baltimore, Md. 1908-12, Technical education at Lehigh Univ., So. Bethlehem, Pa.; Geological-Mining course, B. S. 1912-13, Colonial Copper Co., Bondholders Syndicate, Asst. Geol. 1913-14, Cons. Coal Co., Fairmont, W. Va., Asst. Div. Engineer.

1914, E. E. White Coal Co., Glen White, W. Va., Asst. Chief Engr. 1914-16, The Henderson Co., Baltimore, Md., Asst. General Engr.

Present position: Investigating Engr. Organization Specialist for the Cram Engineering Co., Baltimore, Md.; also private practise mining.

William A. Stedman, Bluefields, Nic., C. A.

Proposed by E. E. Carpenter, R. Hawxhurst, Jr., C. Sullivan.

Born 1879, Helena, Mont. 1897, High School, Helena, Mont. 1899-1902, Operator, Helena Light & Power Co., Mont. 1902-04, Operator, Missouri River Power Co., Mont. 1904-06, Mgr. and Chief Elect., Greater Electric Co., Dillon, Mont. 1906-08, Mining own account. 1908-13, Millman and Elect., Tonopah Belmont Co., Tonopah and Millers, Nev. 1913-15, Master Mech. and Chief Elect., Nevada Wonder Mining Co., Wonder, Nev. 1915 to present, In charge of Construction, Eden Mining Co., Pis-Pis River district, Nic., C. A.

Present position: Supt. of Construction, Eden Min. Co.

John Leo Towne, Whitepine, Colo.

Proposed by Donald Dyrenforth, C. R. Hill, P. M. McHugh.

Born 1885, Lynn, Mass. 1901-02, Perley High School, Georgetown, Mass. 1902-03, Haverhill High School, Haverhill, Mass. 1904-06, Dummer Academy, South Byfield, Mass., received diploma 1906. 1906-07, Worcester Polytechnic Institute, left the middle of the year. 1903-04 and summer of 1905, piece work contract box factory, Haverhill, Mass. 1907, Firing Lennox & Briggs power plant, Haverhill, Mass. 1908, Put in year in Haverhill box factory. 1909, Experimental work and in charge of wetting and soling machine, Pepperell, Mass. 1910-12, Construction and operating foreman, placer seasons, Buffalo Mining Co., Dillon, Colo. 1912-14, Foreman, Silver Spar Min. Co., Whitepine, Colo. 1914-15, Diamond drill foreman, Akron Mine, Whitepine, Colo. 1915-16, Leasing, Whitepine, Colo.

Present position—1916 to date: Supt., Akron Mine.

Hugh R. Van Wagenen, New York, N. Y.

Proposed by A. H. Rogers, J. V. N. Dörr, S. H. Ball.

Born 1883, Denver, Colo. 1906, E. M. from Colo. School of Mines. 1906-07, Supt., Castle Dome Mining Co., Dome, Ariz. 1908-09, Supt., San Robert Mining Co., Zacatecas, Mexico. 1910-11-12, Asst. to A. H. Rogers, E. M. 1913-14-15-16, Supt., Cons. Nevada-Utah Corp., Pioche, Nev.

Present position: Examining Mining Engr.

Dorian Guido Vernay, Bethlehem, Pa.

Proposed by Joseph W. Richards, R. J. Wysor, Chas. L. T. Edwards.

Born 1887, Palermo, Italy. 1898-1904, Mandralisca Liceum, Cefalu, Italy. 1905-09, Royal Univ. of Palermo, Italy, L. L. B. 1909-12, Royal Military Arsenal of Turin and special course in Mechanical Science, Politecnico of Turin, Italy, M. E. During summers 1904-06, Panzera Bros. Iron & Steel Foundry. 1909-10, Royal Military Arsenal of Turin. 1911-12, Royal Military Arsenal of Turin, Machine Shop Foreman and Chief Shop Inspector. 1912, Lieutenant of Artillery of a special mission at the Experimental Military Balipedio of Nettuns, Roma, Italy. 1913, Went to France as representative and broker for the Societa Anonima Ferro & Metalli. 1914-15, Worthington Pump & Machinery Corp., Efficiency Engineering.

Present position—1916 to date: Bethlehem Steel Co.

Albion James Wadhams, Bayonne, N. J.

Proposed by David H. Browne, R. A. Nowlin, John F. Thompson.

Born 1875, Annapolis, Md. Phillips Academy, Andover, Mass.; U. S. Naval Academy, class 1895; U. S. Navy, Commissioned Officer until 1901. Upon resignation from the U. S. Navy, employed by the (Orford Copper Company) International Nickel Co. 1904-12, Supt. Camden Works.

Present position—1912 to date: Asst. General Superintendent, International Nickel Co.

Frank A. Wilder, North Halston, Va.

Proposed by Ensor R. Dunsford, Lawrence I. Neale, H. J. Brown.

Born 1870, Akron, Ohio. 1892, A. B., Oberlin College. 1893, Yale Univ. 1893-99, Science teacher, High Schools, Ft. Dodge and Des Moines. 1900, Asst. State Geologist, Iowa. 1902-13, State Geologist, North Dakota. 1903-16, State Geologist, Iowa. 1906-17, Organizing and managing Southern Gypsum Co.

Present position: President Southern Gypsum Co., Inc.

Malcolm Earl Wilson, Rolla, Mo.

Proposed by H. A. Buehler, C. R. Forbes, W. E. McCourt.

Born 1886, Sheldon, Iowa. 1907, B. S., University of Iowa. 1909, M. S., University of Iowa. 1909-14, Instructor in Geology and Mineralogy, Washington Univ., St. Louis, Mo. Part of summer session at University of Iowa. Field Work during summers of 1910-12 with Missouri Bureau of Geology and Mines, Rolla, Mo. 1914, Geologist with Missouri Bureau of Geology and Mines.

Present position: Geologist Mo. Bureau of Geology and Mines.

Fukunosuke Yamada, Shimotsuke, Japan.

Proposed by R. Kondo, Masayuki Otagawa, R. W. Raymond.

Born 1880, Tokyo, Japan. 1899-1902, Mining & Metallurgical Eng. course, Tokyo Imperial Univ. 1902-07, Engr. Mining Dept. Ashio Mines and Works. 1907-08, Spent about six months visiting the Western mining districts of U. S. A. 1908-11, Supt., Honzan Mine, Ashio Mine and Works. 1911-12, Engineer to Furukawa & Co. 1912-14, Technical College of Aix-la-Chapelle and visiting famous mines of Europe and U. S. A. 1914-16, Visited the mining districts of Australia for about three months. 1916-17, Engineer to Furukawa & Co.

Present position: Manager, Mining & Concentration Department, Ashio Mines and Works.

*Associate Members***John Clyde Lavery, Hurley, N. Mex.**

Proposed by Obar Wiser, Fidel C. Martinez, H. G. S. Anderson.

Born 1891, Liberal, Mo. 1897-1905, Grammar School, Liberal, Mo. 1905-09, High School, Liberal, Mo. 1910-11, Lavery Coal Co., Liberal, Mo. 1911-12, Prospecting for coal with J. R. Smith, Farmington, N. M. 1912, Sampling drill holes, Chino Copper Co. 1912-13, Classifier Operator, Chino Copper Co. 1913-14, Classifier Foreman, Chino Copper Co.

Present position—1914 to date: Concentrate Foreman, Chino Copper Co.

James S. Ryan, Hurley, N. M.

Proposed by H. G. P. Anderson, F. R. Wicks, O. Wiser.

Born 1893, Chicago, Ill. 1906, Grammar School. 1910, High School. 1912, Business College (2 years). 1912, Chicago Telephone Co., installation, phone repairs and upkeep of material. 1914, Studying and clerking in Law Office in Silver City, N. M. 1915, Chino Copper Co. as stenographer to Asst. Supt. and Metallurgical Clerk.

Present position: Metallurgical Clerk.

LeRoy Thatcher, Union Mines, Nev.

Proposed by J. G. Scrugham, F. C. Lincoln, H. M. Rives.

Born 1880, Ohio. 1899-1900, Colorado School of Mines. 1894-98, Miner. 1900, Miner, Croesus Mine, Idaho. 1901-02, Supt. Mutual Mine, Idaho. 1903-06, Mgr., Mountain Belle, Idaho. 1907, Promoter of Nev. Mines and Prospects. 1913, Sold property, promoted the Union Mines Co. and have been engaged with this company since that time.

Present position: General Mgr., Union Mines Co.

Clarence A. Tracy, Los Angeles, Cal.

Proposed by L. O. Howard, R. H. Norton, O. J. Tuschka.

Born 1860, Parkman, Ohio. 1874, General Education. 1917, for 16 years, Mining Representative for The B. F. Goodrich Co., San Francisco, California.

Present position: Mining Representative for The B. F. Goodrich Company.

Erne A. Wright, Schenectady, N. Y.

Proposed by E. E. Carpenter, C. Sullivan, R. Hawxhurst, Jr.

Born 1887, Schenectady, New York. 1903, Public Schools. 1908, Schenectady High School. 1907, I. C. S. Electrical Engineering (Graduated). 1905-09, Genl. Electric Co. Jan. to Aug., 1910, Draftsman, Ore. R. & N. Co., Portland, Ore. 1910-12, S. H. & A. S. R. R., El Paso, Tex., Electrician. In charge development and installation of Plant, Electra. C. M. & M. Co., Denver, Colo. 1914-16, Supt. Elec. & Mech. Depts. Lluvia de Oro M. Co., Chihuahua, Mexico. Jan., 1917, In charge of Mill & Elec. & Mech. Depts., Portrero Mining Co., Mexico.

Present position: Chief Electrician, Eden Mining Co., Bluefields, Nic., C. A.

Junior Members

Louis Durand Asmus, Phila., Pa.

Proposed by Joseph E. Hilles, Frank C. Roberts, Harrison Souder.

Born 1894, Phila., Pa. Graduate of the Mechanic Arts Class of 1915, Drexel Inst.

Present position: Member of Junior class in Engineering School at the Drexel Institute.

Cecil Fasson Blogg, Seattle, Wash.

Proposed by Milnor Roberts, Joseph Daniels, Henry Landes.

Born 1894, New Haven, Conn. 1901-09, Grammar School of New Haven, Conn. and Seattle, Wash. 1909-13, Broadway High School, Seattle, Wash. 1913 to date, Student in College of Mines, Univ. of Wash. Summer of 1916, Ore sampling and crushing and asst. fire assayer in Ore Testing Laboratory of Falkenburg & Laucks, Seattle, Wash. Sept., 1917, In charge of stock room, College of Mines, Univ. of Wash.

Present position: Student, Senior Class, College of Mines, University of Washington.

Albert K. Chan, Golden, Colo.

Proposed by W. G. Haldane, H. J. Wolf, J. C. Roberts.

Born 1894, Sacramento, Cal.

Present position: Student, Colorado School of Mines.

Thomas E. Cassilly, St. Paul, Minn.

Proposed by W. H. Emmons, E. H. Comstock, P. Christianson.

Born 1893.

Present position: Senior in College of Mines, Minnesota School of Mines.

Howell Smith Clark, Rolla, Mo.

Proposed by H. T. Mann, A. L. McRae, C. Y. Clayton.

Born 1892, Fremont County, Iowa. 1907-08, Oklahoma State Prep. School, Tonkawa, Okla. 1911-13, Central High School, Kansas City, Mo. 1914-17, Missouri School of Mines, Rolla, Mo. 1909-10, Chemical Laboratory. 1913-14, Armour & Co., Kansas City, Mo. 1915-16, Special Analyst for Professor Gottschalk, School of Mines. 1916-17, Chemist, Experiment Station School of Mines, Rolla, Mo.

Present position: Chemist, Experiment Station, Missouri School of Mines.

Norman R. Copeland, Golden, Colo.

Proposed by H. C. Parmelee, J. C. Roberts, Harry Wolf.

Born 1895, Aspen, Colo. 1910-11, Aspen High School. 1911-14, North Denver High School. 1914-17, Colo. School of Mines. 3 months in 1913, 1914, 1915, 1916, Golden Cycle Mine, Victor, Colo.

Present position: Student, Colo. School of Mines.

Wilbur Crispelle, Golden, Colo.

Proposed by J. C. Roberts, Harry J. Wolf, Geo. J. Young.

Born 1895, Leadville, Colo. 1909-13, Leadville, Colo., High School. 1914-17, Colorado School of Mines. 1913-14, Mechanical Dept. American Smelting & Refining Co. at plant in Leadville, Colo. Summer vacations, same position.

Present position: Student, Colorado School of Mines.

Earl J. Dickinson, Golden, Colo.

Proposed by J. C. Roberts, Harry Wolf, W. G. Haldane.

Born 1895, Denver, Colo. 1909-13, West Denver High School. 1914, Colo. School of Mines.

Present position: Student, Colo. School of Mines.

Thomas Fraser, Urbana, Ill.

Proposed by H. H. Stoek, E. A. Holbrook, C. M. Young.

Born 1893, White Hall, Ill. 1913, University of Ill., Mining Department.

Present position: Student, University of Illinois.

Robert W. Gibson, Golden, Colo.

Proposed by J. C. Roberts, Harry J. Wolf, W. G. Haldane.

Born 1887, Chattanooga, Tenn. 1894-1900, Chattanooga Public Schools. 1900-03, Baylor Prep. School, Chattanooga, Tenn. 1914, Colo. School of Mines. 1904-10, Genl. Engineering & Construction. 1910-12, Engineering-Hydro-electric & Irrigation Development, North Platte Valley Irrigation Co., Douglas, Wyo. 1912-14, Engineering & Construction, with J. E. Hayes, Chief Engr., Denver, Colo. Summer, 1916, Geological Survey, State of Wyo.

Present position: Student, Colo. School of Mines.

Charles Scott Hyatt, South Bethlehem, Pa.

Proposed by Joseph W. Richards, Benj. L. Miller, G. A. Roush.

Born 1894, Cambridge, Ohio. 1907-10, H. S., Cambridge, O. 1910-11, H. S., Marietta, Ohio. 1913-14, Muskingum College. 1915, Student of Metallurgy at Lehigh Univ. 1912-13, Guernsey Works of American Sheet & Tin Plate Co.

Present position: Student, Lehigh University.

Chester M. Knepper, Golden, Colo.

Proposed by Harry Wolf, Geo. J. Young, H. C. Parmelee.

Born 1861, Somerset, Pa. 1884, Graduate U. S. Naval Academy.

Present position: Student.

William Korotkin, Houghton, Mich.

Proposed by A. J. Houle, F. W. McNair, W. E. Hopper.

Born 1888, Orsha, Russia. 1895-1906, Russian Public Schools. 1912-15, Michigan Agricultural College. 1915 to date, Student, Michigan College of Mines. 1906-09, Private Tutor. 1909-10, Bookkeeper. 1910-12, Factory hand. Machinist during summer vacations of 1913-14-15-16.

Present position: Student, Michigan College of Mines.

Stanislaw W. Lesniak, Rolla, Mo.

Proposed by R. G. Bowman, C. R. Forbes, H. T. Mann.

Born 1889, Oswiecim, Poland. 1900-09, Gymnasium in Austria. 1909-11, University of Cracow and Commercial Academy in Austria. 1913-14, Civil Engineering, Missouri School of Mines and Metallurgy, Rolla, Mo. Sept. to Dec., 1915, Butte & Superior Copper Co., Butte, Mont., working in mine and mill. June to Aug., 1916, Diamond drill prospecting, also some experience in surveying, Doe Run Lead Co., Rivermines, Mo. Feb. to March, 1917, Testing Dept., International Smelting & Ref. Co., Tooele, Utah.

Present position: Student, Missouri School of Mines.

Arthur Clayton Mallams, Weir, Kans.

Proposed by B. L. Wolfe, Charles D. Demond, Alfred L. O'Brien.

Born 1895, Weir, Kans. 1901-09, Public Schools, Frontenac, Kan. 1909-11, High School, Frontenac, Kans. 1914-17, Junior in Kansas State School of Mines, Weir, Kan. 1912-13, Rodman, Central Coal & Coke Co., Pittsburg, Kans. 1913, Asst. Engr., J. R. Crowe Coal & Mining Co., Weir, Kans. 1914, Asst. Mining Engr., Central Coal & Coke Co., Bevier, Mo. 1914-16, Student, Kans. School of Mines, Weir, Kans. Summer, 1915, Engr. Mallas & Halstead Coal Co., Weir, Kans. 1916, Engr. Union Ice & C. S. Co., Weir, Kans.

Present position: Student, Kansas State School of Mines.

Nelson Marvin Morris, Champaign, Ill.

Proposed by H. H. Stoek, E. A. Holbrook, C. M. Young.

Born 1894, Lebanon, Ill. 1908-12, Harrisburg High School, Harrisburg, Ill. 1913-17, University of Illinois, Urbana, Ill., B. S. in Mining Engineering. The summers of 1909, '10, and '11 were spent in and around the mines of the Saline County Coal Co., at Harrisburg, Ill. 1912-13, Saline County Coal Co. Big Four R. R. Schott Construction Co. During the summers of 1914 and 1915, I worked on the surface and during the summer of 1916, I worked underground for the Saline County Coal Co.

Present position: Student.

William Joseph Murphy, Golden, Colo.

Proposed by H. J. Wolf, G. J. Young, J. C. Roberts.

Born 1893, Springfield, Mass. 1908-12, Technical High, Springfield, Mass. 1912-14, Penn State College. 1914-17, Colorado School of Mines.

Present position: Student, Colorado School of Mines.

Clarke Osborne Noble, New York, N. Y.

Proposed by R. M. Raymond, Charles P. Berkey, J. F. Kemp.

Born 1890, Albion, N. Y. 1912-14, Dalhousie Univ., Halifax, N. S. 1914-16, Nova Scotia Technical College, Halifax, N. S. 1916, B. S. 1914, summer, Nova Scotia Steel & Coal Co., Bell Island, Newfoundland, Survey Office. 1915, summer, on geological survey of mining districts for the Nova Scotia Govt. 1916, summer, Dome Mines Co., Ltd., So. Porcupine, Ont., Survey Office.

Present position: Student at Columbia University.

Walter T. O'Reilly, Golden, Colo.

Proposed by J. C. Roberts, H. J. Wolf, W. G. Haldane.

Born 1895, Trinidad, Colo. Summer 1916, At the Ohio & Colo. Smelter, Salida, Colo.

Present position: Student, Junior Class, Colorado School of Mines.

James Willard Pugh, Rolla, Mo.

Proposed by H. T. Mann, A. L. McRae, C. Y. Clayton.

Born 1893, Indianapolis, Ind. 1913, Kansas City Central High School, Kansas City, Mo. Three Years, Missouri School of Mines. 1915, Sampler, Detroit Copper Mining Co., Morenci, Ariz. 1916, Flotation Operator, American Zinc, Lead & Smelting Co., Cartersville, Mo.

Present position: Student, Missouri School of Mines.

Thomas Graham Ralph, South Bethlehem, Pa.

Proposed by Benj. L. Miller, Jos. W. Richards, Howard Eckfeldt.

Born 1895, Crafton, Pa. 1909-12, Crafton High School. 1912-13, Mercersburg Academy. 1916 (summer), The Bristol Brass Co., Bristol, Conn.

Present position: Student, Lehigh University.

Harry A. Robinson, Golden, Colo.

Proposed by J. C. Roberts, Harry Wolf, W. G. Haldane.

Born 1894, New York City. 1909-13, Lawrence High School, Lawrence, Mass. 1913-17, Colorado School of Mines. Summer 1916, Butte & Superior Mill, Butte, Mont.

Present position: Student, Colorado School of Mines.

Fitch Robertson, Golden, Colo.

Proposed by J. C. Roberts, Harry Wolf, W. G. Haldane.

Born 1896, Pueblo, Colo. 1914, Centennial High School, Pueblo. 1914 to present time, Colorado School of Mines.

Present position: Student, Colorado School of Mines.

Carroll Cushing Taylor, Golden, Colo.

Proposed by W. G. Haldane, Harry Wolf, J. C. Roberts.

Born 1895, Townsend, Mass. 1908-12, N. Plainfield High School, N. Plainfield, N. J. 1913, '14, '15, '17, Colo. School of Mines. 1915, Chapin Mine, Oliver Mining Co., Iron Mt., Mich. 1916-17, Asst. Metallurgist, International Motor Co., Plainfield, N. J.

Present position: Junior, Colo. School of Mines.

Mark Loren Terry, Rolla, Mo.

Proposed by H. T. Mann, A. L. McRae, C. Y. Clayton.

Born 1896, Jamesport, Mo. 1914, Jamesport High School, Jamesport, Mo. Three years, Missouri School of Mines. 1916, Mill work, American Zinc, Lead & Smelting Co., Cartersville, Mo.

Present position: Student, Missouri School of Mines.

Herbert M. Tichborne, Mount Vernon, N. Y.

Proposed by R. M. Raymond, Wm. Campbell, Robert Peele.

Born Mount Vernon, 1894.

Present position: Student, Columbia School of Mines.

Stratton Vance, So. Bethlehem, Pa.

Proposed by Howard Eckfeldt, Benj. L. Miller, Joseph W. Richards.

Born 1896, Binghamton, N. Y. 1914, Graduated Port Washington High School. Present position: Student, Lehigh University, Class of 1918. Taking Mining Engineering.

Raymond S. Weimer, Weir, Kans.

Proposed by B. L. Wolfe, Charles D. Demond, Alfred L. O'Brien.

Born 1894, Cable, Ill. 1900-08, Viola Grammar School. 1908-11, Viola High School, Viola, Ill. 1914-17, Kansas State School of Mines, Weir, Kans. 1911-12, Coal Miner, Coal Valley Mining Co., Mathersville, Ill. June, July, Aug., 1915, Miner, Empire Mining Co., Galena, Kans. June, July, 1916, Prospector for B. L. Wolfe in Northern Arkansas. Aug.-Sept., 1916, Miner, American Zinc, Lead and Smelting Co., Cartersville, Mo.

Present position: Student in Kansas State School of Mines.

Leonard Hilliard Whitney, Downers Grove, Ill.

Proposed by H. H. Stoek, E. A. Holbrook, C. M. Young.

Born 1894, Downers Grove, Ill. 1908-13, Downers Grove High School, Downers Grove, Ill. 1913-17, University of Ill., Urbana, Ill. Dept. of Mining Engineering (ore mining option), will graduate in June, 1917. June-Sept., 1915, Miner, Camp Bird, Ltd., Ouray, Colo. June-Aug., 1916, Surface work, Butte & Superior Mining Co., Butte, Mont. Aug.-Sept., 1916, Mines (Drilling and Timbering), Diamond and Pilot Butte Mines, Anaconda Copper Mining Co., Butte, Mont.

Present position: Student, University of Illinois.

P. R. Yewell, Stanford Univ., Cal.

Proposed by H. W. Young, D. M. Folsom, W. F. Dietrich.

Born 1892, Albuquerque, N. M. Student, Leland Stanford Jr. University, Stanford Univ., Cal. 1911-12, One and one-half years in the engineering dept. of the Atchison, Topeka & Santa Fe Railroad (San Joaquin Valley, Arizona & Los Angeles Divisions). 1913, Three months with the Commary-Peterson Co., San Francisco, Cal. (State Highway Construction.)

Present position: Student, Leland Stanford Jr. Univ.

Lawrence J. Zoller, Rolla, Mo.

Proposed by C. R. Forbes, A. L. McRae, C. Y. Clayton.

Born 1895, Pine Bluff, Ark. 1901-09, St. Mary's School, Guthrie, Okla. 1909-10, Logan County High School, Guthrie, Okla. 1910-13, Sapulpa High School, Sapulpa, Okla. 1914, Missouri School of Mines, Rolla, Mo. 1913-14, Producers Oil Co., Tulsa, Okla. Barnsdall Oil Co., Tulsa, Okla. Summers 1915 and 1916, Media Mining Co., Webb City, Mo.

Present position: Student, Missouri School of Mines.

CHANGE OF STATUS

From Junior Member to Member

John Orth Liebig, Newark, N. J.

Proposed by J. W. Richards, H. S. Eckfeldt, H. S. Drinker.

Born 1890, Sparrows Point, Md. 1914, Graduate in Metallurgy Lehigh University. 1914-15, Blast Furnace Dept., Maryland Steel Co., Sparrows Point, Md. 1915-16, Silver Shop, Balbach Smelt. & Ref. Co., Newark, N. J. 1916, Shift Foreman, Ore & Bullion Dept., Balbach Smelt. & Ref. Co. 1916, Foreman in Heat Treating Dept., Hyatt Roller Bearing Co., Harrison, N. J.

Present position: With Met. J. G. Ayers, Hyatt Roller Bearing Co., Harrison, N. J.

Eugene George Snedaker, San Francisco, Cal.

Proposed by W. G. Haldane, J. W. Hutchinson, E. A. Julian.

Born 1890, Aspen, Colo. 1908-09, Lawrenceville, N. J. 1909-14, Colorado School of Mines, Golden, Colo. Degree of E. M. 1912, Summer with Cornucopia Mines Co., Cornucopia, Oregon. Genl. Mill work in Cyanide Plant. 1914, August to October, Mill work, Carmi Mine, Carmi, British Columbia. 1915-16, Goldfield Cons. Mines Co., Goldfield, Nev.; General cyanide practice and flotation, also mine staff engineering dept.

Present position: Field Engr., Exploration Dept., Goldfield Cons. Mines Co.

Carl G. Stifel, St. Louis, Mo.

Proposed by C. R. Forbes, P. N. Moore, H. A. Buehler, W. E. McCourt.

Born 1894, St. Louis, Mo. 1907-11, Yeatman High School, St. Louis, Mo. 1911-13, Washington University. 1913-15, Mo. School of Mines, Rolla, Mo. 1915, General Prospecting for J. T. Boyle, Stockton Hill, Ariz. 1916, Three months experimental work in flotation dept., American Lead, Zinc & Smelting Co., Cartersville, Mo.

Present position: Vice Pres., Otto F. Stifel Union Brg. Co.

CHANGE OF ADDRESS

The following corrections for the Year Book have been received at the Secretary's office during the period Mar. 10 to Apr. 10, which taken together with the lists of new members and changes of address published

in the Bulletins Nos. 121 to 124, therefore, supplement the annual Year Book and bring it up to date Apr. 10, 1917.

- ALDRICH, HAROLD W., Genl. Supt., Ladysmith Smelt. Works, P. O. Box 228, Ladysmith, B. C., Can.
- BAIN, H. FOSTER..... P. O. Box 994, American Post Office, Shanghai, China.
- BAKER, JOHN M..... Falk Bldg., Boise, Idaho.
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NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period Mar. 10 to Apr. 10, 1917.

Date of Election.	Name.	Date of Death.
900	Browne, David H.	Mar. 30, 1917.
890	Kerr, Mark B.	— 1917.
889	Leonard, William A.	Mar. 21, 1917.
872	Peters, E. D.	Feb. 17, 1917.
916	Rich, Charles H.	Mar. 24, 1917.

BIOGRAPHICAL NOTICE

HERMANN A. KELLER

Hermann A. Keller was born in Philadelphia, Pa., Mar. 23, 1860. He received a preliminary education at the gymnasium of Darmstadt, in Germany, and subsequently entered the University of Pennsylvania, graduating in 1881. After serving for a few months as an assistant instructor in mining and metallurgy at the University, he went to Leadville, Colo., where he became, at first, assayer and chemist, and in due time (1886) superintendent of the Arkansas Valley smelting works. In 1888, he was appointed metallurgist of the Globe works at Denver; in 1889, superintendent of the Philadelphia works at Pueblo; and in 1891, superintendent of the Parrot works at Butte, Mont., where he remained until 1896, in which year he was employed to reconstruct the smelting plant of the Mountain mines at Keswick, Cal. In 1897, he opened an office as consulting engineer in San Francisco; and from this time on enjoyed a wide practice both as metallurgical and as mining engineer.

In the former capacity, Mr. Keller had already distinguished himself by the invention of the Keller mechanical roasting furnace employed at the Parrot works in Butte and the Germania works near Salt Lake. He was a frequent contributor to the technical journals, and he collaborated in the preparation of the chapter on "Bessemerizing" in Dr. Peters's work, "Modern American Copper-Smelting." It is a curious coincidence that these two friends and collaborators died at almost the same time. Keller in Arizona and Peters in Massachusetts on the 16th and 17th of February respectively.

The latter years of Mr. Keller's life were largely given to the examination and development of mines in all parts of the world—Alaska, Chile, Germany, etc. He finally established his professional headquarters in New York City. His death was caused by an underground locomotive accident at Clifton, Arizona, on Feb. 16, 1917.

Mr. Keller became a member of this Institute in 1881, immediately upon his graduation at the University of Pennsylvania. His contributions to the *Transactions* comprise papers on "The Desilverization of Lead-Slags" (vol. XXI, p. 71); "Improved Slag-Pots" (vol. XXII, p. 574); and "The Development of the Basic-Lined Converter for Copper Mattes" (vol. XLVI, p. 474).

By his fellow-members he was not only highly esteemed, but universally beloved. The Board of Directors of the Institute adopted the following minute concerning his death:

"In behalf of the professional colleagues and personal friends, as well as the surviving relatives of Mr. Hermann A. Keller, this Board receives with sincere regret the tidings of his premature and sudden death on Feb. 16, 1917, an event which removed from earth a metallurgical and mining engineer of genius, industry, great experience and wide repute, enhanced by an upright and generous personal character."

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The Tredinnick-Pattinson Process

BY WILLIAM E. NEWNAM,* B. S., COLLINSVILLE, ILL.

(St. Louis Meeting, October, 1917)

WHEN Hugh Lee Pattinson discovered, in 1829, that the crystals formed during the slow cooling of molten lead were poorer, and the remaining liquid richer in silver, than the original lead, an important step was made in the metallurgy of this metal. Being the first process applicable to the desilverization of low-grade lead bullion, it soon found its way into all European countries having such a product to treat, thereby effecting the saving of many ounces of silver annually which previously had been thrown away. The old hand process was expensive, as viewed from the standpoint of more modern methods; the labor was great and the tonnage was small, but on the other hand the results were fairly satisfactory.

This process flourished for some time after the introduction of the cheaper Parkes process in 1852 and it is still in use in a few European smelteries.

Only one notable improvement was made on the original process, that of Luce and Rozan at Marseilles, France. It consisted in the application of steam for stirring and the tapping of one-third of the liquid lead, through perforated plates covering the tap holes flush with the bottom of the kettle, thus leaving two-thirds in the form of crystals in the original kettle. This modification considerably lowered the operating costs as it permitted of much larger kettles, an increased tonnage and eliminated much of the slow, hard labor of the hand process. The Luce and Rozan process has persisted in its original form to the present day.

Stephen Tredinnick, English born, spent most of his years around the Luce and Rozan plants at Marseilles and elsewhere in Europe, coming later to Eureka, Nev., in 1878, to operate the Luce and Rozan plant at that point.

Mr. Tredinnick was a competent Luce and Rozan operator and being devoted to the process he firmly believed that with modifications the costs could be so lowered as eventually to supplant the Parkes process.

Out of many schemes he decided that this could be accomplished by placing the Luce and Rozan kettles upon hydraulic rams so that each

* Superintendent, St. Louis Smelting & Refining Co.

kettle could be raised or lowered at will. He accordingly took out patents to cover this idea.

As Mr. Tredinnick's age and infirmities incapacitated him for active service, the writer was commissioned by a large lead refinery to design, erect and place in operation a plant of the Tredinnick-Pattinson type.

Working on refined desilverized lead from the Parkes process, carrying variable percentages of bismuth, two objects inspired the installation: First, to produce a large tonnage of refined lead low enough in bismuth (0.05 of 1 per cent.) for corroding purposes; and, second, a small tonnage in which the major portion of the bismuth would be concentrated so that it might be recovered by further treatment in the Betts electrolytic process, all of which constituted a step in metallurgy hitherto untried.

After considerable experimental work, it became evident that in order to produce a large tonnage of corroding lead at one end of the plant and at the other end a small tonnage of anode lead carrying 1.0 per cent. of bismuth from a refined lead supply carrying 0.33 per cent. bismuth, an 11-kettle plant would be required with the charge-lead going into the eighth kettle from the corroding end.

As the operation of such a plant would resemble a train of gear wheels, inasmuch as the breaking down of one unit would stop the entire plant, it was necessary to make decided changes in the Luce and Rozan equipment so that all parts from the kettle down could be quickly replaced.

In the Luce and Rozan operations the crystallizing of the kettle required about 1 hr., and the melting of the 30 tons of crystals resulting from the operation required about 2 hr., thus 3 hr. were required to an operation.

It was evident that such speed would be fatal to our requirements and that it would be necessary to crystallize and tap in about 30 min. and melt 42 tons of crystals in about 45 min. This was eventually accomplished in the first instance by increasing the steam pressure from 45 to 110 lb., thus permitting the more copious use of water for cooling down, and in the second instance by an efficient application of fuel-oil heating. The increase in steam pressure necessitated a different and much stronger type of steam valve and a secure anchorage for the baffle plate at the bottom of the kettle.

As each kettle would be raised and lowered 7 ft. at frequent intervals, the air, water and oil connections would, of necessity, be flexible rubber and metallic hose and the steam, fume, and smoke connections of a telescoping type.

Thus it will be seen that this plant necessitated a considerable departure from the Luce and Rozan as regards equipment and methods of operation, as the liquid lead would be tapped directly to the next kettle to be operated, thus avoiding the use of auxiliary melting pans.

In this paper I will omit mention of the mechanical difficulties en-

countered, although they were numerous, describing only the strong and simple final construction.

Equipment

The completed 11-kettle plant contained the following:

Eleven stands of Pattinsonizing kettles having a working capacity of 63 tons each and placed in a line at 12 ft. centers;

One 180-ton storage furnace receiving refined lead direct from the Parkes process refining furnace;

One 150-ton molding furnace for the corroding lead coming from No. 1 kettle in frequent taps of 21 tons each;

One 42-ton anode molding kettle for the high-bismuth anode lead coming from No. 11 kettle in occasional taps of 21 tons each;

One 150-ton dross-reducing furnace;

One oblong spout kettle, filled with hot lead, for heating the tapping spouts, the spouts being kept therein until needed;

One 18-ton ladle for charging molten lead from the holding furnace into the charge kettle, which is usually No. 8.

All of the above equipment was covered by a 25-ton traveling crane which was used primarily for charging molten lead to the process, and secondarily for the replacement of broken kettles and other heavy equipment. (A defective kettle can be removed and a new one installed in 20 min.)

Each stand consists of a 63-ton kettle and combustion chamber inclosed in a brick-lined steel casing having an I-beam base, all superimposed upon a hydraulic ram 26 in. in diameter with a 7-ft. stroke. These rams are connected with a pump and accumulator and operate under a pressure of 500 lb. per square inch. Thus any kettle can be raised and lowered at will, by means of a Critchlow valve, and its contents tapped to either adjacent kettle (Fig. 1).

Each kettle is supplied with a special steam valve which enters the side close to the bottom and terminates under the center of the heavy baffle plate which is 45 in. in diameter, being perforated so as to secure as equal distribution of the steam as possible throughout the lead mass (Fig. 2). This steam valve is so constructed that it can be replaced in a few minutes. The steam is carried to the valve through a telescope pipe connection at a pressure of 110 lb.

Each kettle is provided with a truncated cone cover having four working doors. Four inches above the top of the cover is placed a circular $\frac{3}{4}$ -in. water pipe, 24 in. in diameter, having eight equally spaced $\frac{3}{32}$ -in. holes on the under side for introducing water into the kettle by means of funnel cups which pass through the cover. The water connection to the stand is a $\frac{3}{4}$ -in. hose 9 ft. long. The water and steam are both controlled by the operator on the second, or kettle floor.

Each kettle cover has a 13-in. round opening and collar in the center connected by a telescope pipe to a sheet-metal flue. This flue is provided with an 8-ft. exhaust fan running at 300 r.p.m. whereby a strong draft is created to remove the waste steam and lead oxide dust from the top of the kettle.

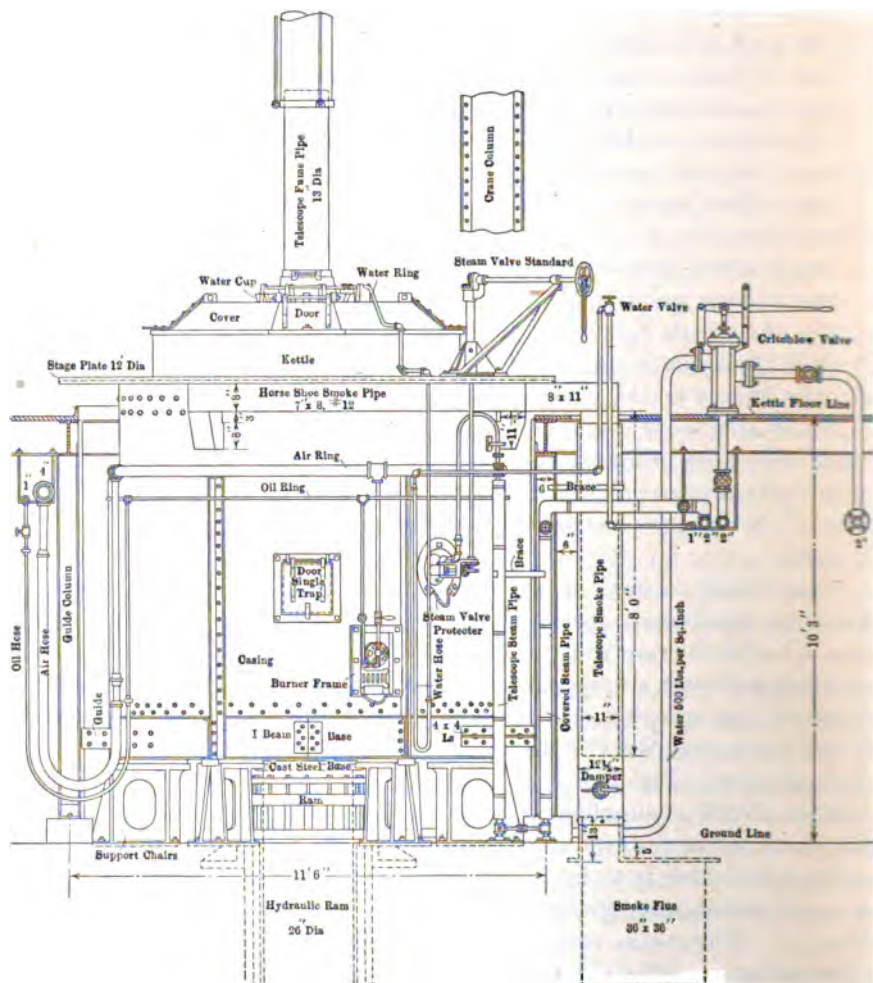


FIG. 1.—KETTLE STAND AND CONNECTIONS.

Heat is applied by three fuel-oil atomizers equally spaced around the casing and operated on an oil pressure of 40 lb. per square inch and a blast pressure of 24 oz. The oil is connected to the stand by a $\frac{1}{2}$ -in. metallic hose 9 ft. long and the air by a flexible 2-in. suction hose.

The waste gases from the oil combustion are drawn off at three equally spaced openings by a horseshoe pipe around the top of the casing and

provided with a telescope pipe connection to an underground smoke flue. This smoke flue is provided with a 48-in. fan running at 440 r.p.m. and serves to draw the flame well up around the sides of the kettle, thus providing equal heat to its entire surface.

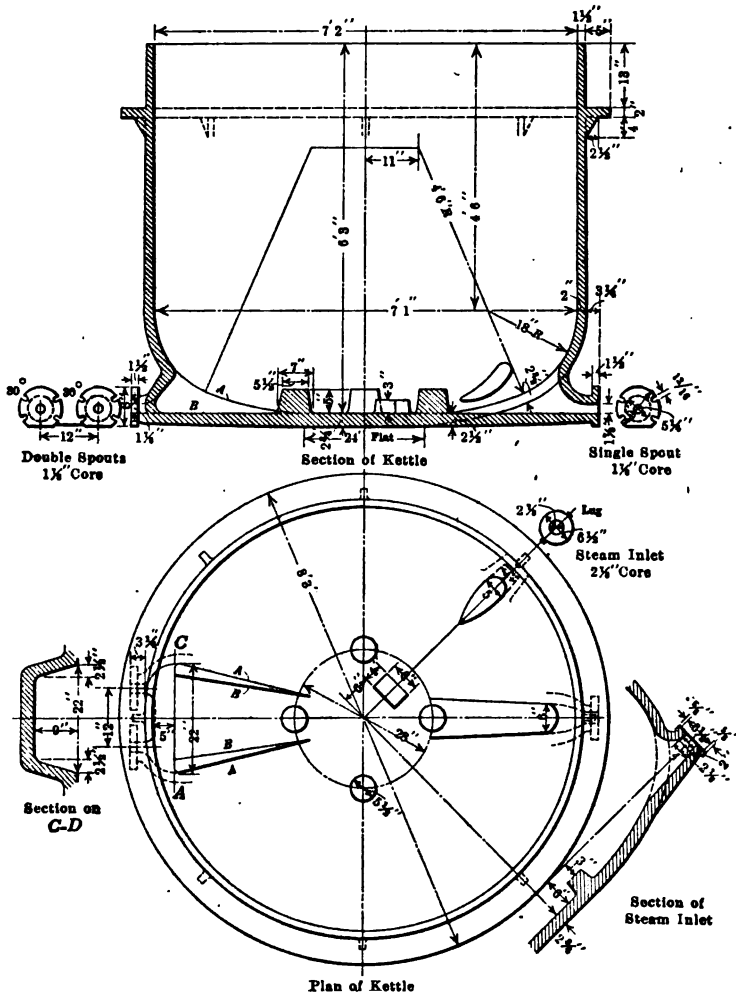


FIG. 2.—KETTLE FOR PATTINSON PLANT.

Method of Operating

In the hand Pattinsonizing, it was possible, with very low-grade bullion, to dip out as much as seven-eighths of the contents of the kettle as crystals, leaving one-eighth remaining as liquid, although the more common practice was to remove two-thirds as crystals, leaving one-third liquid behind. In the Luce and Rozan process, as the crystals remain

in the kettle and the liquid is tapped out through screens which hold the crystals back, it is not practicable to thicken to more than two-thirds crystals and effect a good separation. Therefore, there was no choice but to follow the custom of thirds and make 42 tons of crystals and 21 tons of liquid at each operation.

Order of Operations

Assuming that the supply of lead is fairly constant as to its bismuth contents, the periodical charge of lead to the plant will always be made



FIG. 3.—CHARGING NO. 8 KETTLE.

in the same kettle. In the present instance this would be the eighth kettle from the corroding end of the plant (Fig. 3).

In order to place the plant in operation, each kettle must contain a certain tonnage of lead with a fixed percentage of bismuth depending upon the position of the kettle in the string. That the process may be continuous, the kettles must be crystallized according to some definite system, as otherwise the plant would soon be in a serious muddle which would require considerable time to straighten out. Two such systems

are possible, each permitting of numerous variations, and have been designated the Tredinnick system and the Newnam system (see Figs. 4 and 5).

If the plant is correctly charged to begin operations, according to either system, it is evident that the first operation will change the distribution of the lead in the plant, as will each additional operation, and a

Condition of kettles at beginning of cycle.

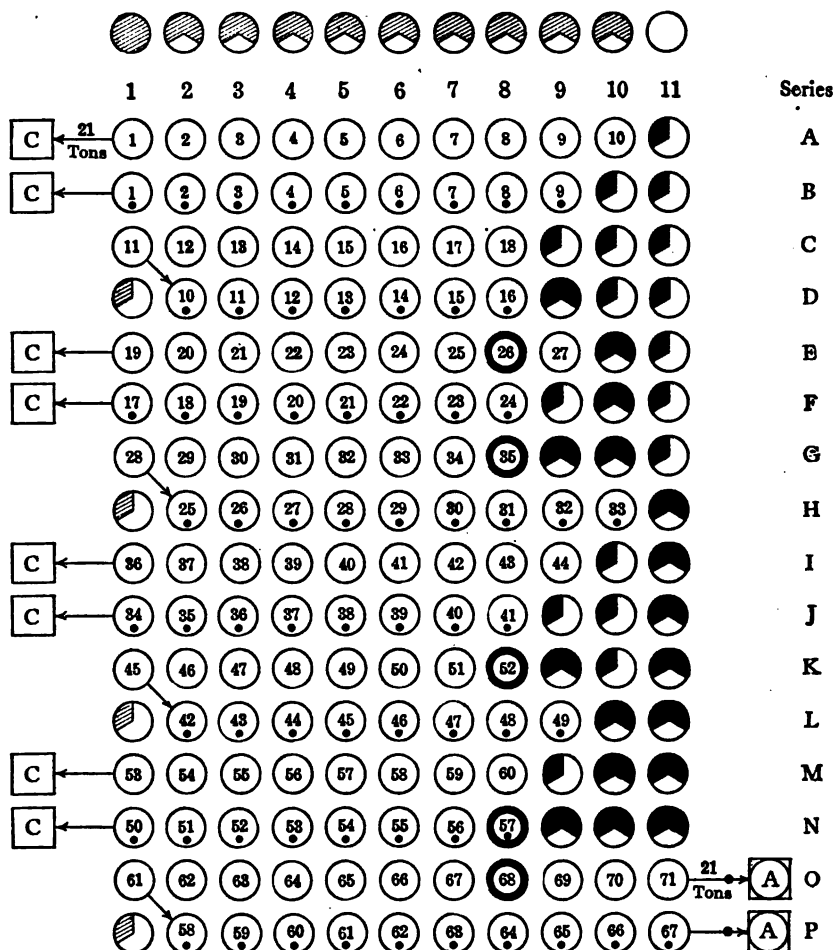


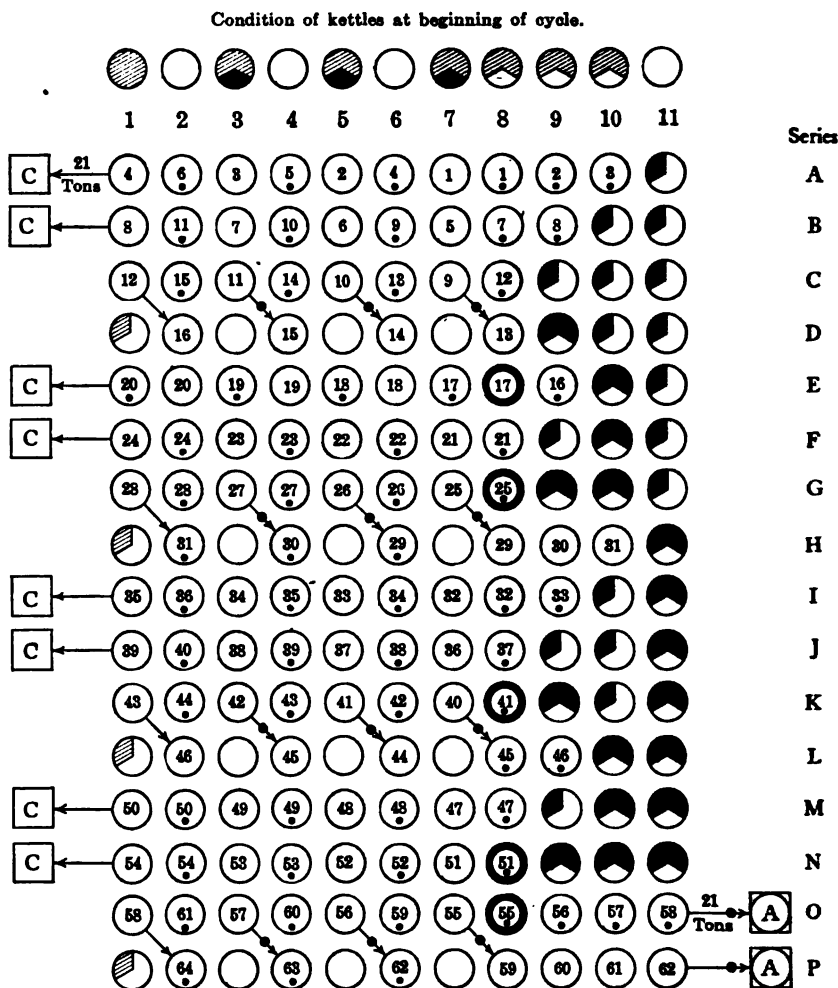
FIG. 4.—THE NEWMAM SYSTEM.

fixed number of operations must be performed before the plant will work back to its original state.

This period of operations is termed a cycle, and, charging in No. 8 kettle, two types of cycles are possible. The first, or single cycle, is that in which one tap of anode lead is made during the period, and the second, or double cycle, that in which two taps of anode lead are made before the plant returns to its original condition.

When the plant is in such a condition that the 9th, 10th and 11th kettles are not operated, the work on Nos. 1 to 8 is called the short string, whereas when these upper kettles come into play the long string is operated.

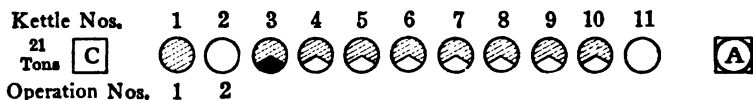
With the short string, 40 to 50 operations are possible in 24 hr., while, with the long string 70 to 80 operations can be made in that time.



As very few persons on this continent are acquainted with Pattinsonizing operations, a detailed description thereof will be necessary to make the accompanying cycle diagrams intelligible. I will first describe the Newnam system (Fig. 4) as that is the easiest to follow, using the following conventional figures to illustrate the process.

No. 2 kettle being full, it is now crystallized and 21 tons of liquid tapped into No. 3 kettle, filling it. As soon as the 42 tons of crystals in No. 2 are melted the kettle is again elevated and contents tapped to No. 1 kettle, filling the latter and completing operation No. 2.

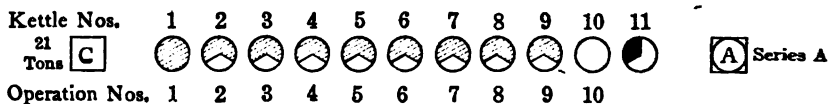
The plant stands as follows:



Kettles Nos. 1 and 3 now being full, it is evident that these two kettles may be operated at the same time, requiring two crews. As the first crew progresses up the string, the second crew would follow behind and as close up as melting of the crystals would permit. In actual practice, usually two or three kettles intervene between the two crews. In order to avoid confusion in the cycle diagram, the work of the two crews is shown in alternating lines; thus, the first crew works out the string marked Series A after which the second crew works out the Series B although, as stated, in actual practice they follow as close behind one another as plant conditions will permit.

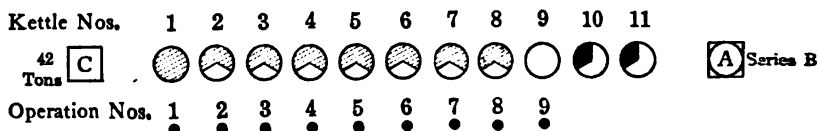
Returning to the last diagram, the first crew will successively crystallize the kettles 3 to 10 inclusive and the operation on No. 10 will place 21 tons of liquid into the empty kettle No. 11. In each case the melted crystals are tapped toward the corroding end.

The plant will stand as follows:



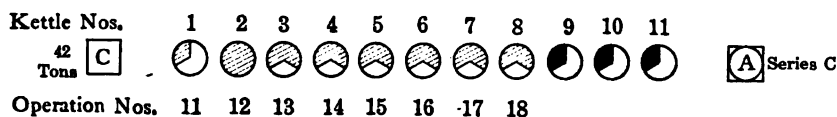
The first crew has thus made 10 operations on the long string, Series A. The second crew now begins on No. 1 kettle and, in like manner, successively operates the kettles Nos. 1 to 9 inclusive and the operation on No. 9 will place 21 tons of liquid into the empty kettle No. 10.

The plant will stand as follows:



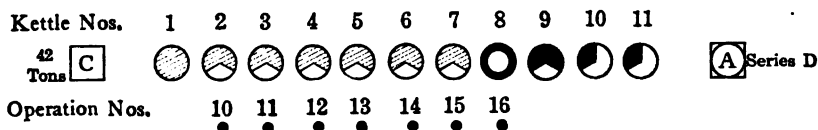
The second crew has thus made nine operations on the long string, Series B, and the first crew now returns to No. 1 kettle to work out its second string, or Series C. Owing to the necessity of correcting the grade of the kettles, as explained later on, a variation of the procedure now takes place in Series C. During this series none of the melted crystals

are tapped toward the corroding end but are left in their original kettle, with the exception of the crystals in No. 1, 21 tons of which instead of being tapped to the corroding furnace are tapped into No. 2 kettle, and the plant stands as follows:

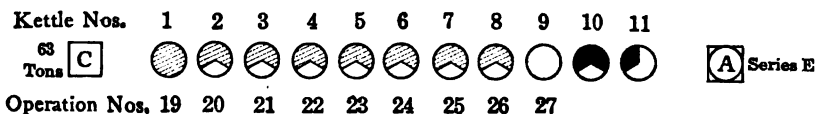


The second crew returning to the corroding end to work out its short string, Series D, is obliged to pass over kettle No. 1 and begin operations on the full kettle No. 2, thus proceeding straight up the line with the operation on No. 8 kettle placing a second tap of 21 tons of liquid in No. 9 kettle. As all crystals on this series are tapped back as usual, and as this will leave the charge kettle No. 8 empty, it is in order to give it a 42-ton charge.

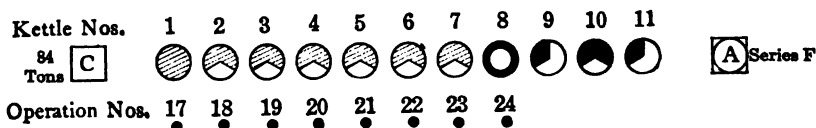
The plant stands as follows:



No. 8 having been charged, the Series E will now be run out by the first crew, and it will be observed that the operation of No. 8 will fill kettle No. 9 which must then be worked, placing an additional 21 tons of liquid in No. 10, or 42 tons in all. Therefore, at the close of Series E by the first crew the plant will stand as follows:



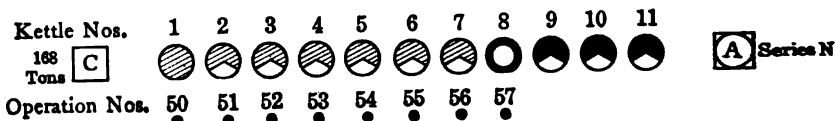
The second crew now begins its 17th consecutive operation with kettle No. 1 of the Series F, at the conclusion of which the plant will stand as follows:



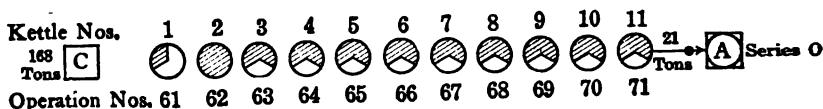
As No. 9 was not operated in Series F, there were no return crystals to No. 8; therefore, the second charge of 42 tons was introduced into that kettle.

On Series G the correction for grade is again made by omitting the corroding tap and running 21 tons of No. 1 crystals into No. 2 kettle.

The plant is steadily worked according to the above plan until the end of Series N, when the plant stands as follows:



As the 9th, 10th and 11th kettles now contain 42 tons each, it is evident that the operations on Series O will include all kettles from 1 to 11 inclusive and that 21 tons of anode lead will be tapped from No. 11 kettle into the anode molding kettle. As the correction of grade occurs at this time, none of the crystals produced on Series O will be tapped back, and at its termination the plant stands as follows:



The first crew will now run out the last string of the cycle, Series P, operating all the kettles from No. 2 to 11 inclusive and placing a second tap of 21 tons of anode lead into the anode kettle. All the crystals having been tapped down the line, at the termination of this series the plant returns to its original condition, as first shown, and a new cycle begins.

During this cycle the following operations were made:

First crew.....	71
Second crew.....	67
Total.....	138

As four corrections for grade were made during this period, $\frac{138}{4}$ or $34\frac{1}{2}$ represents the number of operations made for each correction in grade. On the cycle shown corroding lead was made in sets of two consecutive taps between corrections; now should these taps be made in groups of one, three or four, an entirely different cycle would result, necessitating a separate diagram for each instance.

During the period, the ingoing and outgoing lead was as follows:

	Tons
Corroding lead, eight taps of 21 tons each.....	168
Anode lead, two taps of 21 tons each.....	42
Charge lead, five charges of 42 tons each.....	210

Dividing these tonnages by the total number of operations, 138, the tons of lead charged and produced may be expressed as follows:

Tons per Operation

Charge	Corroding	Anode
1.522	1.218	0.304

In order to handle the same tonnage and obtain the same results by the Tredinnick system, an entirely different procedure is necessary. Whereas in the Newnam system each series is worked out by one crew, in the Tredinnick system each series is split up between the two crews and in order to understand its workings the series must be shown in two parts. The condition of the kettles at the beginning of a double cycle is entirely different, and the plant, ready to operate, stands as follows:

Kettle Nos.	1	2	3	4	5	6	7	8	9	10	11		
C												A	
Percent Bi	0.050	0.068	0.081	0.104	0.130	0.164	0.206	0.280	0.330	0.440	0.580	0.770	1.000

Observe that there are four full kettles, Nos. 1, 3, 5 and 7, and that Nos. 8, 9 and 10 contain 42 tons each while Nos. 2, 4, 6 and 11 are empty.

The first crew operates in the following order: No. 7, No. 5, No. 3, No. 1. All resulting crystals are tapped down the line, 21 tons of No. 1 crystals being tapped to the corroding furnace. At the end of the first half of Series A, the plant stands as follows:

Kettle Nos.	1	2	3	4	5	6	7	8	9	10	11
21 Tons											
Operation Nos.	4		3		2		1				

The second crew will now work successively Nos. 8, 9 and 10, placing 21 tons of liquid in No. 11. They will then operate in order Nos. 6, 4 and 2 and the plant stands as follows:

Kettle Nos.	1	2	3	4	5	6	7	8	9	10	11
21 Tons											
Operation Nos.	4	6	3	5	2	4	1	1	2	3	

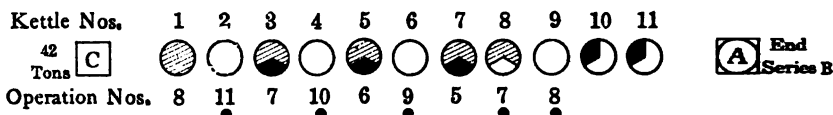
The first crew now returns to No. 7 and works successively Nos. 7, 5, 3 and 1, 21 tons of crystals being tapped to the corroding furnace from No. 1 kettle. At the end of the first half of Series B the plant stands as follows:

Kettle Nos.	1	2	3	4	5	6	7	8	9	10	11
42 Tons											
Operation Nos.	8		7		6		5				

The second crew will complete Series B by operating kettles 8 and 9,

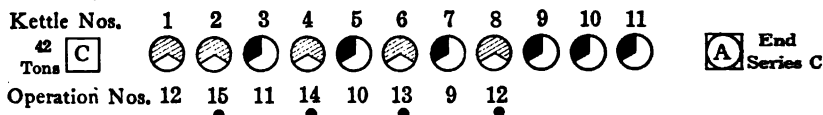
placing 21 tons of liquid in the empty kettle No. 10 and then operating in order Nos. 6, 4 and 2.

The plant stands as follows:

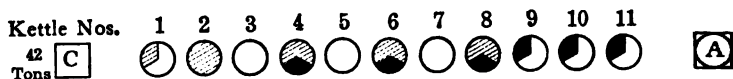


Series C is now run out by the first crew taking the kettles 7, 5, 3 and 1 and then the second crew the kettles 8, 6, 4 and 2. As a result of this series, 21 tons of liquid are placed in the empty No. 9. Since preparation for correction of grade begins at this point, the tap to the corroding furnace is omitted, and the crystals in 8, 6, 4 and 2 are not tapped back.

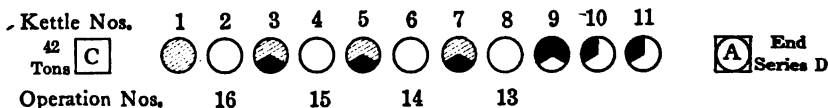
The plant stands as follows:



Now the 21 tons of liquid in Nos. 7, 5 and 3 are tapped respectively to Nos. 8, 6 and 4 and 21 tons of crystals in No. 1 are tapped back into No. 2, and the plant stands as follows:



The first crew now operates Nos. 8, 6, 4 and 2 and the resulting crystals being tapped back as usual, Series D is terminated and the plant stands as follows:



As No. 8 kettle is now empty it must receive a charge of 42 tons.

With slight variations the foregoing procedure is repeated until the plant returns to its original condition, which will take place at the 64th operation of the second crew.

The above double cycle produces the following data:

Operations

First crew.....	62
Second crew.....	64
Total.....	126

Tons per Operation

Charge	Corroding	Anode
1.666	1.333	0.333

A comparison of the two systems may be summed up as follows from the two cycles shown:

	Tredinnick	Newnam
Maximum lead in process.....	399	483
Minimum lead in process.....	336	399
Number of operations per cycle.....	126	138
Tons charged per operation.....	1.666	1.522
Tons corroding lead produced per operation.	1.333	1.218
Tons anode lead produced per operation....	0.333	0.304
Total tons charged.....	210	210
Total tons produced.....	210	210

From an inspection of the above figures, it would seem that the Tredinnick system has decided advantages in every respect. But with the material on which we were experimenting we found it necessary periodically to make a correction for grade. We also found that the necessity of skimming kettles one-third full of lead was a laborious task, and if not properly done the bottom of the kettles soon took on a heavy coating of litharge most prejudicial to the heating and to the life of the kettles.

In overcoming these disadvantages, we worked out the Newnam-system, and found that with our particular class of work its relative simplicity more than offset the advantage of the Tredinnick system. Other plants might have a different experience and might find the Tredinnick system superior.

Grade of Kettles

Each kettle in the string has its own particular grade, as determined by experience, and it must be maintained at that grade to insure the production of corroding lead from No. 1 kettle crystals.

The grade of a kettle is governed by the bismuth contents of the mixture of one-third liquid and two-thirds crystals going to fill that particular kettle. It is manifest that this should be the same at all times, but this is far from being the case, and thereby serious complications arise.

As the liquids all travel toward the anode end of the plant and all the crystals travel toward the corroding end, it is evident that if one kettle becomes overgrade it will communicate that effect to its neighbors and a correction of some kind will be necessary.

Owing to the natural percentage of enrichment of bismuth in the liquid lead by crystallizing, there is a decided tendency for the grades of all kettles between the corroding end and the charge kettle to increase steadily. This insidious tendency has been termed the "crawl."

The following semi-mathematical exposition of this "crawl" will clearly show its cause and effect.

The average percentage of bismuth in the liquid from any operation is about 1.4 times the percentage of bismuth in the kettle charge before crystallizing. Although not strictly true, it has been the custom to call this 40 per cent. enrichment.

The average percentage of bismuth in the crystals is about 0.8 times the percentage of bismuth in the kettle charge before crystallizing. In like manner, this is spoken of as 20 per cent. impoverishment.

For the sake of simplicity, consider the effect of this 40 per cent. enrichment on one kettle only and it will be evident that this applies to any kettle between No. 1 and No. 11, except that the kettle considered, No. 8, has a partial correction made to it every time it receives a charge of new lead, hence the crawl is not so constantly accumulative in No. 8 as it is in any other kettle.

Assume that No. 8 is charged with lead containing 0.300 per cent. Bi. Crystallize the kettle of 63 tons which will furnish:

No. 7 with 42 tons of crystals carrying 0.240 per cent. Bi.

No. 9 with 21 tons of liquid carrying 0.420 per cent. Bi.

It is legitimate to assume, at this point, in the next series of operations up the line that No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.240 per cent. Bi.

The kettles stand thus:

		No. 7	No. 8	No. 9
I.	Tons.....	63	Empty	21
	Per cent. Bi.....	0.240		0.420

Now crystallize No. 7, which will furnish:

No. 8 with 21 tons liquid carrying 0.336 per cent. Bi.

No. 6 with 42 tons crystals carrying 0.192 per cent. Bi.

Charge No. 8 with 42 tons of lead at 0.300 per cent. Bi; the average contents of No. 8 will then be 63 tons carrying 0.312 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.437 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.250 per cent. Bi.

Observe that the liquid from No. 7 to No. 8 is 0.336 per cent. Bi, or 12 per cent. higher than it should be. This before it has been subjected to the influence of any of the higher kettles. It is therefore correct to say that the liquid from No. 6 to No. 7 will increase 12 per cent. every time the operations proceed up the line. Hence, as No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.268 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
II.	Tons.....	63	Empty	42
	Per cent. Bi.....	0.256		0.428

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.358 per cent. Bi.

Charging No. 8 kettle with 42 tons lead at 0.300 per cent. Bi; it will contain 63 tons of lead carrying 0.319 per cent. Bi.

Crystallize No. 8, which will furnish:

No. 9 with 21 tons liquid carrying 0.446 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.225 per cent. Bi.

No. 9 now contains 63 tons of lead at 0.434 per cent. Bi, which crystallized will furnish:

No. 8 with 42 tons crystals carrying 0.347 per cent. Bi.

As No. 7 is approached the next time, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.300 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
III.	Tons.....	63	42	Empty
	Per cent. Bi.....	0.270	0.347	

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid at 0.378 per cent. Bi. It will now contain 63 tons carrying 0.357 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.500 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.286 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.336 per cent. Bi.

The kettles will stand thus:

		No. 7	No. 8	No. 9
IV.	Tons.....	63	Empty	21
	Per cent. Bi.....	0.302		0.500

Crystallize No. 7 which will furnish:

No. 8 with 21 tons liquid carrying 0.422 per cent. Bi.

Charge No. 8 with 42 tons of lead at 0.300 per cent. Bi, and it will contain 63 tons carrying 0.341 per cent. Bi.

Crystallize No. 8 which will furnish:

No. 9 with 21 tons liquid carrying 0.477 per cent. Bi.

No. 7 with 42 tons crystals carrying 0.273 per cent. Bi.

As No. 7 is again approached, No. 6 will furnish:

No. 7 with 21 tons liquid carrying 0.376 per cent. Bi.

The kettles will stand:

		No. 7	No. 8	No. 9
V.	Tons.....	63	Empty	42
	Per cent. Bi.....	0.341		0.488

To Recapitulate

	No. 7	No. 8	No. 9
I.....	0.240	0.300	0.420
II.....	0.256	0.312	0.437
III.....	0.270	0.319	0.446
IV.....	0.302	0.357	0.500
V.....	0.341	0.341	0.477

It can readily be seen that this is an accumulative "crawl" and while it is aggravated each time No. 9 is brought into play and in like manner more so when No. 10 and No. 11 are drawn upon, yet it would exist were there no kettles above No. 8. Furthermore, the percentage rate of "crawl" of No. 2 or any other kettle will be the same as that of No. 7.

It is evident that the percentage of enrichment does not fit the method of crystallizing in "thirds" as it does in the case of the enrichment of silver. Compare the kettle grades as they are maintained in the case of bismuth and silver.

Kettle No.	1	2	3	4	5	6	7	8	9	10	11		
Bi per cent.....	0.050	0.063	0.081	0.104	0.130	0.164	0.206	0.260	0.330	0.440	0.580	0.770	1.000
Ag os.	$\frac{1}{4}$	$\frac{1}{2}$	1	2	4	7	12	20	35	60	100	180	300

It has been shown that with "40 per cent." enrichment and consequent "20 per cent." impoverishment, if the impoverished lead from any other kettle be crystallized it will yield a liquid 12 per cent. higher than the grade of the original kettle from which it came. This does not represent the rate of "crawl" proportional to it on account of the charges introduced into No. 8 of proper grade. For the sake of comparison, however, the following table will show what the relative per cent. of "crawl" is for the different percentages of enrichment.

Enrichment, Per Cent.	Faulty Increase, Per Cent.
10	4
20	8
30	10
40	12
50	12.3
60	12
70	10
80	8
90	4
100	0

Therefore, the only percentage of enrichment with which there would be no "crawl" would be 100 per cent., which is nearly the case in that of silver, thus in the old Pattinson operations for silver the "crawl" was insignificant.

Correction of Grade

In order to offset the "crawl," it is necessary to resort to some mechanical method of correction (at least up to our present knowledge), such as tapping a kettle overgrade to the kettle of next higher grade or to increase the number of crystallizations upon a given tonnage of lead.

This is accomplished by a so-called "jump"¹ whereby a combination of these two schemes is effected but which results in greatly lowering the tonnage of the plant with a corresponding increase in the cost per ton charged.

The "jump" is made by periodically tapping 21 tons of melted crystals from No. 1 kettle onto the 42 tons of crystals in No. 2 kettle and then going straight up the line, without any of the crystals in advance being tapped back. Thus the crystals receive an extra operation and one-third liquid is pushed by successive stages up the line.

The method of "jumping" is shown by the cycle diagrams under "Order of Operations."

Table 1 gives the relative tonnages according to the frequency of the "jump" and the kettle receiving the charge.

TABLE 1

Charge No.	Jump	Cycle Operations	Tons per Operation Charged	Tons per Operation Corroding	Tons per Operation Anode	Anode Fraction of Tons Charged
6	1-28- $\frac{5}{8}$	256D*	1.634	1.471	0.163	$\frac{1}{10}$
7	0	244S†	2.754	2.668	0.086	$\frac{1}{32}$
7	1-40	120S	1.750	1.575	0.175	$\frac{1}{10}$
7	1-31- $\frac{3}{4}$	190D	1.547	1.326	0.221	$\frac{1}{4}$
8	0	131S	2.565	2.405	0.160	$\frac{1}{16}$
8	1-44	176D	1.670	1.432	0.238	$\frac{1}{4}$
8	1-34- $\frac{1}{2}$	138D	1.520	1.210	0.310	$\frac{1}{6}$
9	1-36	76D	1.658	1.105	0.553	$\frac{1}{2}$

* D = double cycle, two taps of anode metal.

† S = single cycle, one tap of anode metal.

Experience has shown the necessity of jumping:

	No.
Once in 28 $\frac{5}{8}$ operations if charging in.....	6
Once in 31 $\frac{3}{4}$ operations if charging in.....	7
Once in 34 $\frac{1}{2}$ operations if charging in.....	8
Once in 36 operations if charging in.....	9

In each of the above four cases, two consecutive taps of corroding lead are made between corrections for grade, the operations in each case being similar to those shown in the cycle diagrams.

The following analytical results shown in Table 2 were made on a

¹ So called on account of passing over No. 1 kettle during the period of correction.

10-kettle plant, each determination being made on a sample taken from a full kettle just before crystallizing.

TABLE 2
No. 1.—Showing the Steady Increase in Grade Without "Jumping"

Corrod- ing "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
0.061	0.075	0.110	0.170	0.208	0.251	0.315	0.400	0.468	0.675		
0.061	0.073	0.131	0.161	0.211	0.285	0.348	0.401	0.529			
0.067	0.079	0.129	0.169	0.229	0.295	0.358	0.412	0.462			
0.071	0.093	0.138	0.148	0.233	0.276	0.346	0.454	0.498	0.793	0.862	0.878
0.081	0.093	0.138	0.181	0.237	0.287	0.358	0.407	0.562	0.641		
0.076	0.093	0.154	0.197	0.254	0.299	0.331	0.399	0.529			
0.076	0.095	0.148	0.197	0.249	0.299	0.362	0.445	0.482			
0.078	0.102	0.142	0.199	0.257	0.328	0.388	0.486	0.476	0.691		
0.074	0.092	0.146	0.204	0.261	0.345	0.366	0.413	0.548			
0.086	0.104	0.158	0.202	0.262	0.316	0.370	0.463	0.495			
0.088	0.117	0.164	0.219	0.274	0.342	0.405	0.445	0.478			
0.095	0.116	0.168	0.225	0.280	0.353	0.408	0.423	0.571	0.796		

No. 2.—Showing Maintenance of Grade by "Jumping"

Corrod- ing "C"	1	2	3	4	5	6	7	8	9	10	Anode "A"
0.050	0.063	0.079	0.103	0.136	0.187	0.236	0.316	C 0.390			
0.048	0.060	0.086	0.112	0.157	0.202	0.265	0.378	0.442	0.521		
0.048	0.055	0.096	0.127	0.180	0.219	0.296	0.367	0.425			
0.045	0.058	0.099	0.136	0.177	0.227	0.285	0.357	0.428			
0.056	0.069	0.108	0.141	0.192	0.248	0.307	0.373	0.422	0.581	0.775	1.045
	0.073	0.110	0.153	0.203	0.254	0.314	0.371	0.475	0.636		
	J	0.081	0.119	0.161	0.212	0.282	0.325	0.409			
0.048	0.058	0.097	0.132	0.174	0.236	0.291	0.359	0.436			
0.049	0.064	0.102	0.137	0.179	0.237	0.294	0.364	0.415			
0.058	0.074	0.109	0.147	0.197	0.245	0.296	0.369	0.417	0.588	0.829	1.103
	0.083	0.111	0.150	0.190	0.253	0.297	0.358	0.436	0.631		
	J	0.080	0.114	0.158	0.218	0.274	0.318	0.373	0.500		
0.053	0.070	0.103	0.136	0.178	0.234	0.282	0.345	0.423			
0.054	0.081	0.105	0.134	0.183	0.229	0.270	0.330	0.353			
	0.066	0.099	0.148	0.197	0.250	0.308	0.329	0.338	0.560	0.830	1.177
	J	0.079	0.124	0.163	0.204	0.275	0.294	0.325	0.430	0.595	0.853
0.046	0.062	0.094									
0.055	0.071										
0.056	0.076										

Description of an Operation

A full kettle (63 tons) skimmed clean and ready to crystallize, should have a temperature not much above the melting point of lead, but the brickwork surrounding the kettle should be hot to prevent a crust freezing

on the inside of the kettle. Correct temperature is an important factor, as lead too hot greatly prolongs the time of crystallizing and produces an excessive amount of dross. In practice, the lead is tested by thrusting a broom handle into the molten metal; if the lead freezes to the handle the temperature is low enough to proceed with the operation. If the lead does not freeze to the handle the kettle must be cooled down by freezing crusts with water and pushing these crusts under the surface with a pole until the proper temperature is secured. As this also consumes time, it is necessary to keep a sharp watch of the temperature.

The kettle being in the proper condition, the operator opens the steam valve slowly until the surface of the lead is violently agitated. (In order to prevent slop the kettles are filled to within 15 in. of the top only.) Water is now cautiously introduced through the eight water cups on the cover, by a valve close to the steam gear wheel. The operator regulates the admission of steam and water so that a maximum amount of water is introduced without causing explosions, the formation of chunks or the slopping of lead through the cover doors.

The water cups occasionally become clogged with lead and have to be freed by a special punching rod in the hands of the barman.

Lead soon freezes in a crust on the upper ring of the kettle and to the cover. Periodically the water is turned off, the cover doors thrown back and the crusts barred down with a 6-ft. steel bar, 1 in. in diameter, having a chisel point. Considerable judgment must be exercised not to allow the crusts to become too thick, as in this case they are difficult to break up with the steam and tend to form chunks. Also, too frequent barring down consumes time, as the water is turned off during that period.

To facilitate barring down, the inside of the cover and the upper ring of the kettle should be as smooth as it is possible to make them.

Soon crystals of lead, from $\frac{1}{16}$ to $\frac{1}{8}$ in. in diameter, begin to appear in the bath, and from this point on they multiply with ever increasing rapidity, the violent agitation by the steam keeping the crystals from adhering to one another.

When the consistency of two-thirds crystals is reached, the surface of the lead appears as an exceedingly thick mass of boiling crystals. At this stage the water is shut off, the kettle barred down for the last time and the crusts broken up by steam. Steam is now turned off, the kettle elevated and two hot spouts, just out of the spout kettle, are placed on the double lead cocks. These cocks (which are kept hot with charcoal) are opened slowly and the one-third liquid tapped to the adjoining kettle through a screen in the bottom of the kettle covering the double taps. This screen has 96 holes $\frac{3}{16}$ in. diameter, 2 in. centers.

As soon as the last crust is barred down, the burners are fired, and by the time the liquid has run out the temperature is rapidly rising.

It is surprising how accurately a good crystallizer can judge the pro-

portion of liquid and crystals in the finished kettle. A good man will seldom be in error over 2 tons and this inequality may be eliminated on a subsequent operation by slightly over- or under-crystallizing as the case may require.

Under the proper conditions, a kettle can be crystallized in 15 min. An average operation, crystallizing and tapping, requires about 30 min.

It will be noted from the cycle diagrams that any delay on one kettle affects the whole plant. Quick crystallizing and quick melting are therefore necessary to speed and the more rapidly these are performed the fewer kettles need intervene between the kettles being operated, thus permitting of more kettles in operation at the same time.

On the short string, two crews are operating simultaneously, whereas on the long string as many as four crews may be operating.

As soon as the crystals are melted the lead is skimmed and tapped to the opposite adjacent kettle through the single lead cock without a screen. The average melting period is about 45 min. This completes an "operation."

The kettles near the corroding end are much more difficult to operate than those near the anode end, also the finished kettle at the anode end appears much thicker. In both instances this is due to the crystals near the anode end being larger than those at the corroding end of the plant.

Dross

Dross has been one of the most objectionable features of a Pattinson plant, both hand and steam. In the early stages of the Tredinnick plant, 21 per cent. of the lead charged was skimmed out as dross. This was not only expensive to handle, reduce and recharge, but it also left the lower kettles short of lead and thereby reduced the tonnage materially.

It was found that by throwing small quantities of fuel oil into the kettle on top of the crystals during the melting period that the melting was greatly hastened and a large percentage of the dross formed during the crystallizing was reduced in the kettle.

Handled in this manner, the reducing furnace only operates 4 or 5 days a month, and the lead removed as dross is from 2 to 3 per cent. of the lead charged.

During the crystallizing, about 50 lb. of litharge, fine as flour, is formed, which is drawn off by the telescope fume pipe and caught in the flue. A very strong draft is necessary to keep this fume from blowing out through the cover doors and it is absolutely essential to the health of the men that none of it be allowed to get into the working atmosphere.

Fuel

The fuel requirements of the plant depend upon the number of operations. An average of 50 gal. of fuel oil is required per operation. Of

this, 40 gal. pass through the burners and 10 gal. are thrown in the top of the kettle over the melting crystals.

Labor

The accompanying labor table is based on a crew sufficient to keep the operations up to the melting and capable of making from 50 to 70 operations each 24 hr., depending on the number of kettles in the operating line or "string." Thus, whereas an 8-kettle plant would mean 50 operations per day, an 11-kettle plant would make 70 operations, the number of kettles in the string depending upon the condition of the kettles above the charge kettle. An inspection of the flow sheet will make this point clear.

As this plant was placed in operation at a time when the 10- and 12-hr.

TABLE 3.—*Labor Table*

Day Shift, No. Men	Rate	Duties	Per Month
1	\$2.65	General foreman.....	\$80.00
1	2.40	Burnerman.....	72.00
1	2.20	Burnerman assistant.....	66.00
2	2.25	Crystallizers.....	135.00
2	1.85	Barman.....	111.00
1	1.75	Barman in training.....	52.50
1	2.00	Tapping and cleaning kettles.....	60.00
1	2.25	Craneman.....	67.50
2	1.75	Clean-up and general utility.....	105.00
12			
Lap Shift			
1	\$2.40	In charge at lap and noon time.....	\$72.00
1	2.00	Crystallizer in training.....	60.00
1	1.85	Barman.....	55.50
2	1.75	Barman in training.....	105.00
5			\$1,041.50
Night shift.....		Same as above.....	\$1,041.50
Reducing Furnace			
1	\$2.00	Furnaceman, 5 days per month.....	\$10.00
1	1.65	Furnaceman helper, 5 days per month.....	8.25
3	1.65	Drossmen, 5 days per month.....	24.75
1		Day superintendent.....	125.00
1		Night superintendent.....	87.00
		Total per month.....	\$2,338.00

day was in vogue, and as it was necessary to keep the plant working 24 hr. per day and not disturb the labor conditions in other parts of the works, a lap shift was introduced, four men coming on at 9 o'clock and working until 7 on each shift, thus keeping the plant moving between shifts and during the luncheon hour.

This arrangement was necessary, as temperature control is of the highest importance to secure tonnage. When the plant is moving at its maximum rate the temperatures take care of themselves to a great extent, as kettles ready to operate are not allowed to stand, but if for any reason the plant is held up for an hour or more it is very difficult to keep both the brickwork and the lead at the proper point.

Experiments

The addition of a third substance or metal was tried in many experiments, in a vain endeavor to find a cheap addition agent that would increase the natural enrichment of bismuth in the liquid.

Out of the many substances tried, tellurium was the only one that had a beneficial influence, but on account of its cost and the difficulty of adding it to the lead it was not available.

Arsenic, tin, cadmium and zinc actually decreased the enrichment the actions of arsenic being especially harmful.

The small quantity of gold and silver in the charge lead is practically all recovered in the anode metal.

The following shows the distribution of copper in the kettle sample taken after the plant had been in operation for a considerable period on refined lead.

Kettle	Cu, Per Cent.
1	0.000075
2	0.000050
3	0.000050
4	0.000250
5	0.000250
6	0.000250
7	0.000250
8	0.000400
9	0.001500
10	0.002800
11	0.003500
Anode	0.009800

The Tredinnick-Pattinson Process as a Desilverizer

Under the heading of "Grades of Kettles," it was shown that the enrichment in the case of silver is nearly 100 per cent. and that a correction of grade would only occasionally be necessary. It was this ideal natural enrichment that made the Pattinson process a success in its day, and although the Tredinnick-Pattinson process has never been used since

desilverizer I believe that I am justified, with the figures at hand, in indulging in some speculation as to its possibilities.

As a basis of calculation, I assume that one Tredinnick-Pattinson operation costs \$7.50, this including all overhead expense such as insurance, taxes, amortization, royalty, general expense, etc. The average Parkes cost per ton of bullion charged in a moderate-sized plant treating lead of 100 oz. silver, with zinc at a normal figure of 5 c. per pound, is about \$4 per ton.

The Tredinnick-Pattinson plant would turn out a high percentage of the lead treated as common lead containing $\frac{1}{4}$ oz. of silver. This common lead would be suited to the ordinary purposes to which common lead is put but it could not be used as corroding lead in the manufacture of white lead as its copper contents would be prohibitive, as the analytical results given in Table 4 will show.

TABLE 4

Kettle	Copper, Per Cent.
Bullion charged.....	0.084
Enriched lead.....	0.044
12	0.052
11	0.080
10	0.084
9	0.056
8	0.056
7	0.064
6	0.056
5	0.052
4	0.048
3	0.044
2	0.048
1	0.040
Impoverished lead.....	0.038

The small percentage of rich lead varying from 100 to 500 oz. of silver would be best treated by the Parkes process, therefore a combination of the two processes would be in order.

Disregarding the periodical correction of grade and avoiding fractions, Table 5 will show the approximate distribution of tonnage for varying amounts of silver, the kettle into which the charge is introduced and the number of kettles in the plant.

From Table 5 and the costs given above, it will be seen that 4 oz. bullion charged in No. 4 kettle in a 9-kettle plant will require 4.35 tons of charge for each operation in that plant, and the cost per ton would be $\frac{\$7.50}{4.35} = \1.72 . Again, 0.07 ton of rich lead would go to the Parkes process for treatment at a cost of \$0.28 ($0.07 \times \4), which would bring the total cost of the combination treatment to \$2 per ton of bullion charged.

TABLE 5.—*The Tredinnick-Pattinson Process*

Silver, Ounces per Ton	Charge in Kettle, No.	Kettles in String	Tons per Operation			Ounces Silver Rich Lead	Cycle No. of Operations
			Charged	Desilverized	Rich Lead		
4	4	9	4.35	4.28	0.07	100	309
4	4	10	4.28	4.25	0.03	180	628
7	5	9	3.71	3.60	0.11	100	181
7	5	10	3.61	3.56	0.05	180	372
12	6	9	3.33	3.12	0.21	100	101
12	6	10	3.17	3.07	0.10	180	212
20	7	10	2.90	2.72	0.17	180	116
20	7	11	2.77	2.68	0.09	300	243
35	8	11	2.56	2.40	0.16	300	131
35	8	12	2.45	2.37	0.08	500	274
60	9	12	2.30	2.16	0.14	500	146
100	10	12	2.27	1.99	0.28	500	74

Table 6 shows the probable outcome by applying the foregoing cost figures.

TABLE 6.—*Comparative Costs of Processes*

Silver, Ounces per Ton Charged	Charge in Kettle, No.	Kettles in String	Tredinnick- Pattinson Cost per Ton Charged	Parkes Cost on Rich Lead per Ton Charged	Total Cost Combination Process per Ton Charged	By Parkes Process Alone	In Favor of the Combina- tion Process	Value of Silver Re- covered, 60 c. per Ounce
4	4	9	\$1.72	\$0.28	\$2.00	\$4.00	\$2.00	\$2.25*
4	4	10	1.75	0.12	1.87	4.00	2.13	2.25*
7	5	9	2.02	0.44	2.46	4.00	1.54	4.05
7	5	10	2.08	0.20	2.28	4.00	1.72	4.05
12	6	9	2.25	0.84	3.09	4.00	0.91	7.05
12	6	10	2.37	0.40	2.77	4.00	1.23	7.05
20	7	10	2.58	0.72	3.30	4.00	0.70	11.85
20	7	11	2.71	0.36	3.07	4.00	0.93	11.85
35	8	11	2.93	0.64	3.57	4.00	0.43	20.85
35	8	12	3.06	0.32	3.38	4.00	0.62	20.85
60	9	12	3.26	0.56	3.82	4.00	0.18	35.85
100	10	12	3.30	1.12	4.42	4.00	0.42	59.85†

* Loss by Parkes.

† Loss by combination.

The figures in Table 6 indicate the possibilities for the combination process with bullions assaying between 4 and 60 oz. of silver per ton, but the price paid for fuel oil or gas and the labor rate would have a very vital bearing on the results.

A plant such as described should treat from 150 tons to 200 tons of bullion per 24 hr., and the construction cost of a completely equipped 12-kettle plant, with material at its normal figure, would be approximately \$85,000.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Coal Wastage*

BY FRANCIS S. PEABODY,† PH. B., CHICAGO, ILL.

(St. Louis Meeting, October, 1917)

THIS paper will not be a technical paper, because, although I have been in the business of mining and selling coal for 30 odd years, I am neither a mining engineer nor a practical miner. If I digress from time to time from my thesis, I must be forgiven by the expert engineers and highly technical gentlemen for whose proceedings this article is written.

Waste of the wonderful store of power that the Creator placed at our disposal appears to have been the theory and the basis of our methods of extracting this power—waste from the time the coal is mined to the time it is consumed. Let us consider the prime causes of this wastage which has been the heritage of the coal mining industry.

In the early days—about 150 years after “Stone Cole” was discovered in Illinois by Joliet and Marquette, and Father Hennepin noted a “cole” mine on his map—the mode of mining prevalent among farmer land-owners, on whose property coal was discovered near the surface or outcropping on the hillsides, was to enter the seam by a shallow shaft or drift into the hillside and remove the coal by wedging it down until a large room was left. The roof without support would fall and our primitive coal operator would then sink another shaft or drive another drift. In these early operations only the large lumps were taken and all small pieces and screenings were left to be covered by the fall of the roof—in fact this condition prevailed in Illinois up to about 1885.

Gradually the method of mining changed from the open chamber, to a single entry with rooms turned to the right and left. Later, as the demand for coal increased and the necessity for larger mines became evident, the “room and pillar” system grew to be regular practice.

The room and pillar system of mining consisted of starting away from the shaft bottom or drift mouth, with a pair of entries usually 12 to 15 ft. wide and about 30 ft. center to center, one entry being used for the air intake and the other for the return air and haulage; from each entry rooms or chambers were turned, narrow at the mouth and widening

* Originally presented at a meeting of the Chicago Section, held May 22, 1916.

† President, Peabody Coal Co.

to between 20 and 30 ft., these rooms being driven to a depth of about 200 ft. from the neck or mouth.

Practically all present-day methods and systems of mining bituminous coal in this country, except "longwall," are based on this system, and it is surprising to note how little improvement has been made over the old method even in what is considered the most approved system of today, the "panel" system.

Using the panel system the operator starts away from the shaft or drift mouth with a pair, or sometimes three entries; but instead of turning rooms off the main entries, "stub" or "panel" entries are turned. These panel entries are driven between 1,000 and 1,200 ft., and from them are turned rooms as the entries *advance*.

It is not my intention to dwell on the merits of our present methods, but rather to show their demerits.

As previously explained, the rooms in both the room and pillar and the panel systems are driven while the entries are also advancing, the first room reaching its limit of 200 or 250 ft., while the second room will be a little shorter, and so on successively up to the face of the entry where the last room will be just started. In Illinois mines of this type, the "room pillars," coal between the rooms, and the "chain pillars," coal between the entries, are left in the mine, generally through inability to remove them on account of the "gob" or refuse which has been placed alongside them while cleaning the coal taken from the room, and because of the fall of slate and roof resulting from insufficient and temporary timbering.

Nor is the coal left in the ground our only loss. The pillars prevent the overlying strata from subsiding evenly, in most cases breaking through to the surface, thereby spoiling the surface drainage, frequently leaving a previously level surface rolling and full of "sink" holes.

We know by visible evidence the conditions on the surface resulting from this uneven settlement; then what greater damage must be done to the thinner seams of coal and other minerals overlying the seam mined, and at present not considered valuable. Thus we are not only wasting our heritage but we are placing almost insurmountable difficulties for our successors to overcome to work the thinner seams.

As we study the old laborious methods of our fathers and compare them with present-day practice, we realize we have not improved coal mining methods at a pace equivalent with other industries. True, we have machines for mining and modern high-speed hoists at our mines producing 5,000 or 6,000 tons of coal per day, but our percentage of recovery is little, if any, higher than in early practice.

Some mining reforms have been forced into the industry by the lessors of coal lands. About the year 1890, automatic stokers began to come into general use, thereby creating a market for the cheaper grades of coal screenings, etc., which up to this time had been thrown away and left

in the mines. It is estimated that probably 57,000,000 tons of screenings had been wasted in Illinois.

With all credit to present-day coal operators, let me remark that most, if not all, mines are started with the view of recovering the chain pillar coal, at least, on the second mining, or as the mine retreats after having reached its boundaries and just prior to abandonment. I am afraid, however, it is mostly a case of "the spirit is willing but the flesh is weak," for I know of only one mine about to be abandoned which worked for nearly 2 years before abandonment recovering pillar coal and recovered approximately 350,000 tons out of possibly 4,000,000 tons left in the ground.

Another, and the most deplorable condition created by present haphazard methods, is the danger to life and limb. Our miners produce a great many more tons of coal per year per man employed than they do abroad, because we have fewer supervisors, and therefore naturally fewer men are employed per ton produced.

In Germany, there is a supervisor for every 15 or 20 men. In this country if we have one pit boss to 150 men we consider it enough; we must have closer supervision to get better extraction results and reduced loss of life and limb.

Foreign reports of 1912—the later reports being deficient on account of the war—show the loss of life in coal mines in Belgium to be 1 man for every 1,000 employed; in Germany, 1.54; in Great Britain, 1.17; in the United States, 3.35. Our wastage of life, considering these figures, cannot but be apparent. Our losses in life are more than three times as much as in the coal mines of Belgium where thin seams are all that is left and these are operated at 2,500 to 3,000 ft. below the surface and with a recovery of about 97 per cent. of the entire seam.

I have endeavored to show that the coal industry is beset by all manner of waste, waste of natural resources, waste of the human element and waste of capital, and we do not seem to realize how dearly future generations must pay for it.

It would be far better if a situation could be created in the near future with strong governmental control, preferably through the medium of the Federal Trade Commission, so that the bituminous coal industry could be thoroughly regulated with respect to the operation of present properties, so that all may operate on a reasonable basis, returning a fair percentage of recovery, with regulations that will insure the best conditions for the safety of life and limb, and so founded that the operator will be assured of a reasonable return on his capital invested.

Such regulation of the coal industry when it does come must begin at the bottom; the industry must be regulated from every standpoint. The governing commission must be assured that the prospective operator owns, or controls, sufficient coal land to permit a mine large enough

to produce a sufficient tonnage to return the investment in the surface plant and non-movable machinery, etc.—in other words, to wipe out the capital accounts.

The sinking of the shafts must be regulated. Our present law in Illinois regarding the sinking of shafts is adequate in its provision for the protection of life, and I believe is fair to the operator.

A system of mining whereby all, or at least 97 per cent., of the coal in the ground must be recovered, would have to be provided and rigidly enforced; a system providing for the protection of the now valueless thinner seams of coal and also for the protection of the other minerals, sandstones, shales and other rocks, which are now of no conceivable value, but may in future years be discovered to be a very important factor in some industry yet undeveloped.

All this regulation would necessarily involve much time and study and would gradually be revised as new conditions were met. Objections would be raised; no doubt many attempts would be made to prove the early acts unconstitutional; no doubt claims would be made, that only those controlling large amounts of capital could enter the business, and such would undoubtedly be a fact.

Let me say that the coal business is not a business for small capital; that is one of its greatest difficulties today. If we want a charter for a coal company now, we can get several dummy directors and pay a lawyer \$50 or \$100 and secure a charter for a full-fledged coal company, with the rights to sink shafts and produce coal, and incidentally waste 45 to 50 per cent. of the natural resources in the ground.

In Germany, where wages are much lower than in this country where machinery is less expensive, the investment in the coal business per ton of the annual production, not including any sums spent for mineral rights, for the government owns it all, is \$2.50. It is a law, in this war-besieged country, that all the coal must be mined. In the States of Illinois and Indiana the investment per ton of the annual production is about \$1.46 and we are mining only about half of the coal in the ground. When we consider the greater money value in Germany, it is apparent that coal mining in Germany is limited to those who are able to do business in a large way and with business-like methods.

I do not know of, nor would I attempt to specify, any particular system whereby all the coal can be recovered, but I do know that if the "retreating" system or something similar to it were adopted and so regulated that all coal operators must meet this standard, it would be a very desirable advance over our present methods.

In the retreating system, as it is talked of among coal men, main entries are driven to the boundaries of the coal property after which stub or panel entries are driven. The main body of the coal is extracted as the mine retreats to the shaft. To do this it is necessary that the mine

be sunk and worked for a number of years, probably averaging between 10 and 15, at a loss.

The surface plant would be fully equipped at a cost of approximately \$300,000, taking the average mine of today as a standard. The coal rights, either owned or leased, would necessitate a carrying charge of say \$25,000 to \$30,000 per year. Until the boundary is reached by the main entries the output would be small, probably no more than 200 tons per day in the first year and about 1,500 tons per day the year the boundary is reached. From then on the tonnage would rapidly increase and the mine be on a paying basis immediately.

Every ton of coal taken from the entries would undoubtedly cost the operator not less than \$2.50 to \$3 per ton to produce and would probably sell, judging from present market conditions, for \$1.50 per ton, leaving a deficit of \$1 to \$1.50 per ton produced, to be charged to the capital account. I do not wish to infer that our present entry coal costs this much to produce, but with the system outlined it would also be a part of the proposed scheme of things that all supports, such as we now call "timbering," would be of a permanent and possibly a recoverable nature. Our mine tracks would be laid of far heavier steel than used at present and all items of operation would be of a far more permanent nature than is now required.

Assuming we opened a mine according to this ideal standard, carried it through for a number of years until the entries reached the limit, all work done being of a permanent nature, with the losses on production capitalized up to the time when production reached a stage sufficient to put the mine on a paying basis, our investment, according to my general figures, would amount to \$500,000. Our mines naturally being fewer in number than at present, our operations would be steady and our ideal mine would produce possibly 2,000,000 tons of coal per year. Then taking into consideration our sinking fund for all causes, we would have to be guaranteed not less than 20 c. per ton net profit to realize an equitable return on our investment.

In Germany, where regulations similar to those I speak of are in effect, persons entering the coal business must show unquestionable evidence that they are able financially to stand the enormous investment called for.

It is not unreasonable to assume that the investing public and the capital interests would not bear the same ill feelings toward the coal industry they now do, if they could be assured the business was so regulated that the possibilities of loss were minimized. It would not be any more difficult to call upon them for large sums to be invested in operating property than it is at present to secure capital for investment in coal lands.

I venture to say that many operators in the business today, if com-

pelled to keep their costs in a standard commercial manner, charging just and fair depreciation on their coal lands and their plants, would find that they are not making any profit; however, I maintain that it would not be against the interests of the public to exact a profit, for we would be saving the natural resources for the years to come, which, under our present method of extravagance, will be so minimized in the future that five and six times the profit I mentioned will be regarded as just.

By government control or regulation of our methods of mining and the natural following of the greater permanency of the work we do, our fatalities among coal mining labor would naturally decrease.

During the first 3 months of 1916, 259 men were killed in the United States through falls of slate and rock in our coal mines; or approximately two men lost their lives from this one avoidable cause, for every million tons of coal produced. This would be particularly safeguarded by our better and more permanent roof supports. With our roof supports more firmly installed, consequently fewer falls, fewer electric wires would be damaged and we could save a proportion of the 18 men killed in the first 3 months of 1916 by electric shocks. The pockets in the roof of the entries from which the loose rock has fallen would not be present to be a gathering place for gas, which caused the death of 82 men in the United States in the first 3 months of 1916.

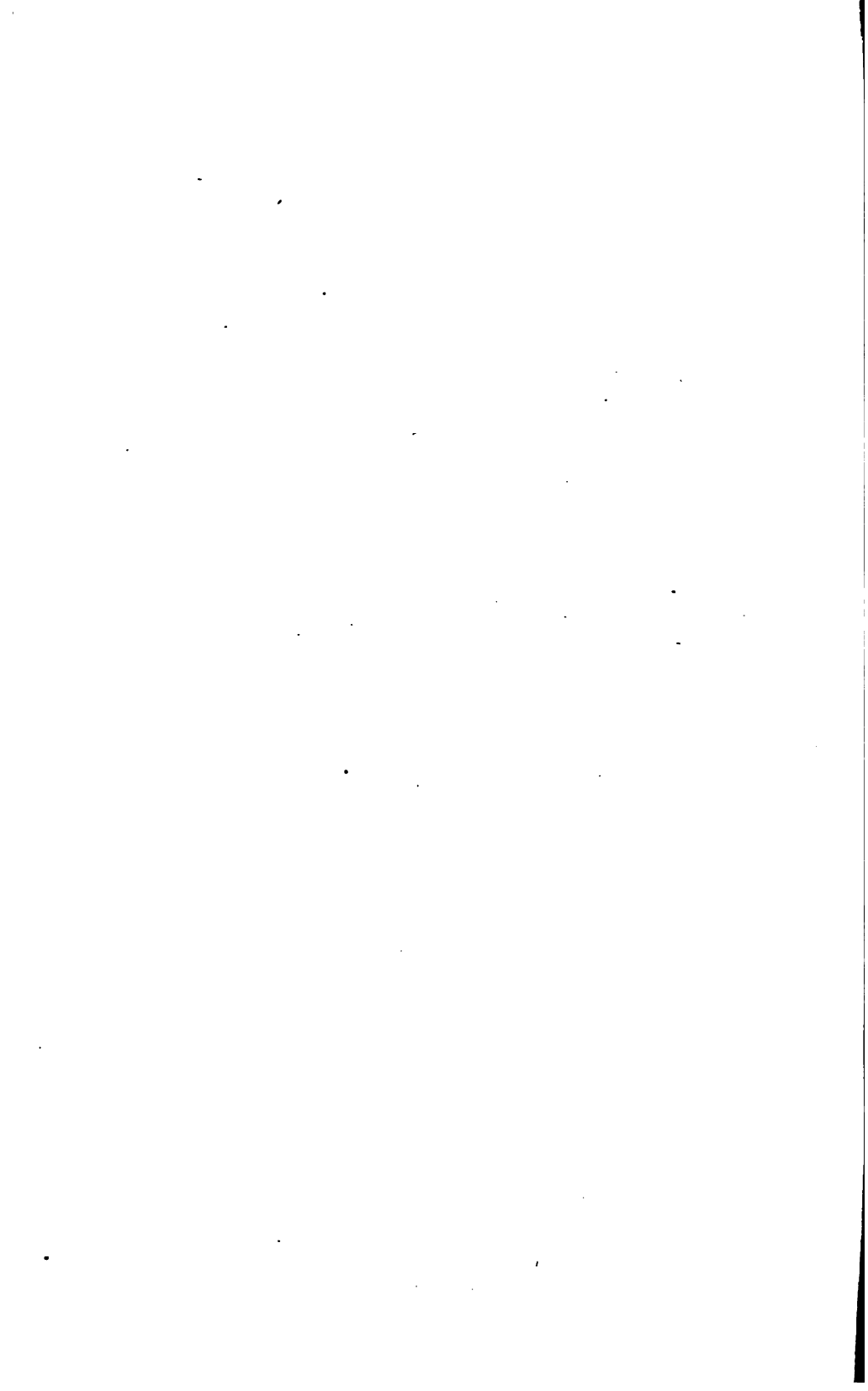
We have many laws on our statute books which I freely say are good laws. Many have been passed to guard the safety of our workmen; very few to protect our natural inherited wealth. We have the Sherman Anti-Trust Law, which most of us have an idea means that competitors cannot combine for any purpose whatever, if the result of that combination is to restrain fair competition or if such combining results in the creation of a monopoly. In England this is common law; in Australia they have a similar law called the Australian Industries Preservation Law, which I believe is a far more pleasant term than Anti-Trust. Our own Supreme Court has interpreted the Sherman Law, "that people cannot get together and combine to create a monopoly or restraint of trade, when it is against the interest of the public to do so." The Australian Law says: "when it is against the interests of the public, you cannot combine."

Unfortunately, as yet there has been no case before our Supreme Court in which they could say the restraint would be unreasonable, so we do not know what facts would constitute a reasonable restraint, or a reasonable monopoly under our laws. In Australia they have had a case affecting coal companies, in which it was shown that there was ruinous competition between two coal fields, and the operators, to remedy conditions, got together and entered into contracts which would raise the price of their product. It was shown that the prosperity of their towns which were dependent upon the coal industry was seriously threatened and in the

operation of the two agreements the selling price of coal at the mines was increased 40 per cent. The House of Lords or the Privy Council in rendering their decision said "That it was impossible to disregard the interests of those who are engaged in production and distribution, and further that it can never be of interest to the consumers if any article of consumption should receive fair remuneration for the capital employed and the labor expended."

There can be no doubt in our minds that these agreements were entered into to raise the price of coal and to preclude competition. However, it was proved that the prevailing prices previous to negotiations of the agreement were disastrously low, owing to the cut-throat competition. Where these conditions prevail the less remunerative collieries will be closed down; there will be a great loss of capital, miners will be thrown out of employment, less coal will be produced and prices will consequently rise until it becomes possible to reopen the closed collieries.

The consumers will lose in the long run if the mine operators do not make a fair profit or the miners do not receive a fair wage; therefore, in the opinion of the English Privy Council, the mere intention of an agreement to raise prices does not always prove the intention to injure the public. To prove an intention to injure the public by raising the prices, the intention to charge excessive or unreasonable prices must be apparent. I, therefore, believe it is to the best interest of all, not only those in the coal industry but also those who have dealings with the coal industry, to advocate strongly the enactment of a law providing for combinations and agreements of the kind which will permit producers of natural resources to produce and market their products under a uniform cost-accounting system, and a uniformly regulated manner of production, safeguarding the natural resources of the earth from wanton waste, and returning to the men in the industry an amount commensurate to the value of their services to society. When I say the men in the industry, I mean the men who work with their hands, the men who are charged with the executive management, and the men who furnish the necessary capital and credit.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

A New Silicate of Lead and Zinc

BY P. A. VAN DER MEULEN,* PH. D., ITHACA, N. Y.

(St. Louis Meeting, October, 1917)

SOME time ago, the writer received from W. O. Borchardt, Manager of the works of the Bertha Mineral Co. at Austinville, Va., several specimens of a dense yellowish slag-like material, containing cavities showing clear needle-like crystals, and representing a product formed by the fumes of zinc oxide and basic lead sulphate from the oxide furnace, attacking the firebrick lining of the flues.

The matrix, which is of a greenish-yellow color, is made up chiefly of an amorphous glass, carrying numerous crystals, the latter being well-defined, clear, and transparent, but colored a very light yellow by small amounts of ferric oxide.

A number of the crystals were carefully separated from the matrix and tested with the blowpipe. They have a fusibility of about 2, yield a globule of lead and a coating of lead oxide before the blowpipe on charcoal, and contain some zinc. Their hardness is 5 to 6.

Under the microscope the crystals appear to be made up of a prism, with narrow faces, and a fairly large pinacoid, but unfortunately show no terminal faces (Fig. 1). Between crossed nicols they gave parallel extinction in the three planes containing the crystallographic axes, and are therefore orthorhombic. The index of refraction is high.

A number of the best crystals were measured on the reflection goniometer, the angles given below representing the average of measurements made on different individuals.

$$mm = 46^{\circ} 52'$$

$$mb = 66^{\circ} 34'$$

$$a:b:c = 0.4334:1:?$$

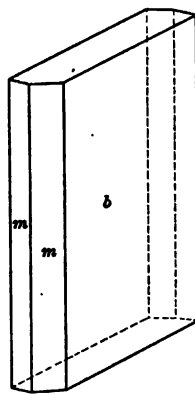


FIG. 1.

Inasmuch as the results indicate that the crystals do not correspond with any silicate of lead and zinc, either natural or artificial, heretofore

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described, a complete chemical analysis of them was undertaken. An analysis of the matrix was also made.

The method employed in each case was briefly as follows: A weighed amount of the finely powdered substance was heated with dilute hydrochloric acid on the water bath, this treatment resulting in the complete decomposition of the crystals. The matrix left a slight undecomposed residue which was separated from the solution by filtration, and fused with a small amount of sodium carbonate, and then dissolved in dilute hydrochloric acid. The solution was evaporated to dryness and taken up in about 200 c.c. of warm water containing a few drops of hydrochloric acid, filtered while warm, and the filtrate again evaporated, diluted, and filtered to remove the last traces of silica. To the filtrate, dilute sulphuric acid was added and the solution evaporated until sulphur trioxide fumes were freely given off. It was then diluted with distilled water, and alcohol added to the amount of 50 per cent., after which the lead sulphate was filtered on a weighed Caldwell-Gooch crucible. From the filtrate, zinc was precipitated with hydrogen sulphide by the modified Waring method,¹ and determined as pyrophosphate in the usual manner. Iron and aluminum were removed by double precipitation with ammonia, and magnesium determined in the filtrate.

The results are given below.

	Crystals			Matrix, Per Cent
	Per Cent.	Ratio		
SiO ₂	13.38	0.2218	1 2	7.11
PbO.....	60.37	0.2706	} Combined	54.25
ZnO.....	22.70	0.2789		2.477 5
Al ₂ O ₃	1.43			3.55
Fe ₂ O ₃	1.53			2.33
CaO.....	none			tr.
MgO.....	0.87			0.13
Total.....	100.28			99.88

Leaving out of consideration the minor amounts of magnesia, alumina, and ferric oxide,² the crystals are seen to be a basic silicate of lead and zinc of the type R''₂Si₂O₆, in which R'' is replaced partly by Pb and partly by Zn, in nearly equivalent amounts. The density of the powdered crystals was determined by the pycnometer method and found to be 6.153 at 20° C.

¹ *Journal of the American Chemical Society* (1907), 29, 284.

² If the iron oxide and alumina are calculated with the silica as acid-forming groups and the magnesia with the bases, the ratio of base to acid becomes 4.690:2 instead of 4.954:2, but it appears to the writer that these are admixed impurities and not part of the regular crystal compound.

The matrix is seen to be still more basic than the crystals, and probably contains some of the free oxide of lead or zinc or both, as a thin section of the matrix shows numerous needle-like crystals of the lead zinc silicate embedded in an amorphous groundmass.

In order to obtain crystals that might be measured and described more completely, a portion of the matrix was placed in a fire-clay crucible, melted in a gas crucible furnace and allowed to cool slowly. Crystals were readily obtained which could be identified under the microscope as being of the same kind as those described, but they were so small and poorly formed that it was not possible to measure them.

From a careful search of the literature, it does not appear that any basic silicate of lead and zinc, or of lead or zinc alone, of the type $R''_2Si_2O_6$, has been described heretofore, either as a mineral or as an artificial product. The system $PbO-SiO_2$ has been investigated thermally by Cooper, Krauss, and Klein.³ They established the existence of the compounds $PbO.SiO_2$; $2PbO.SiO_2$; and $3PbO.2SiO_2$, and considered the existence of the compound $3PbO.SiO_2$ as probable, although no evidence of it was obtained in the melting-point curve. It seems not improbable, in view of the results of the analyses of the crystals described, that the compound $5PbO.2SiO_2$ should also exist and would probably be formed from a melt which is more basic than the crystals themselves.

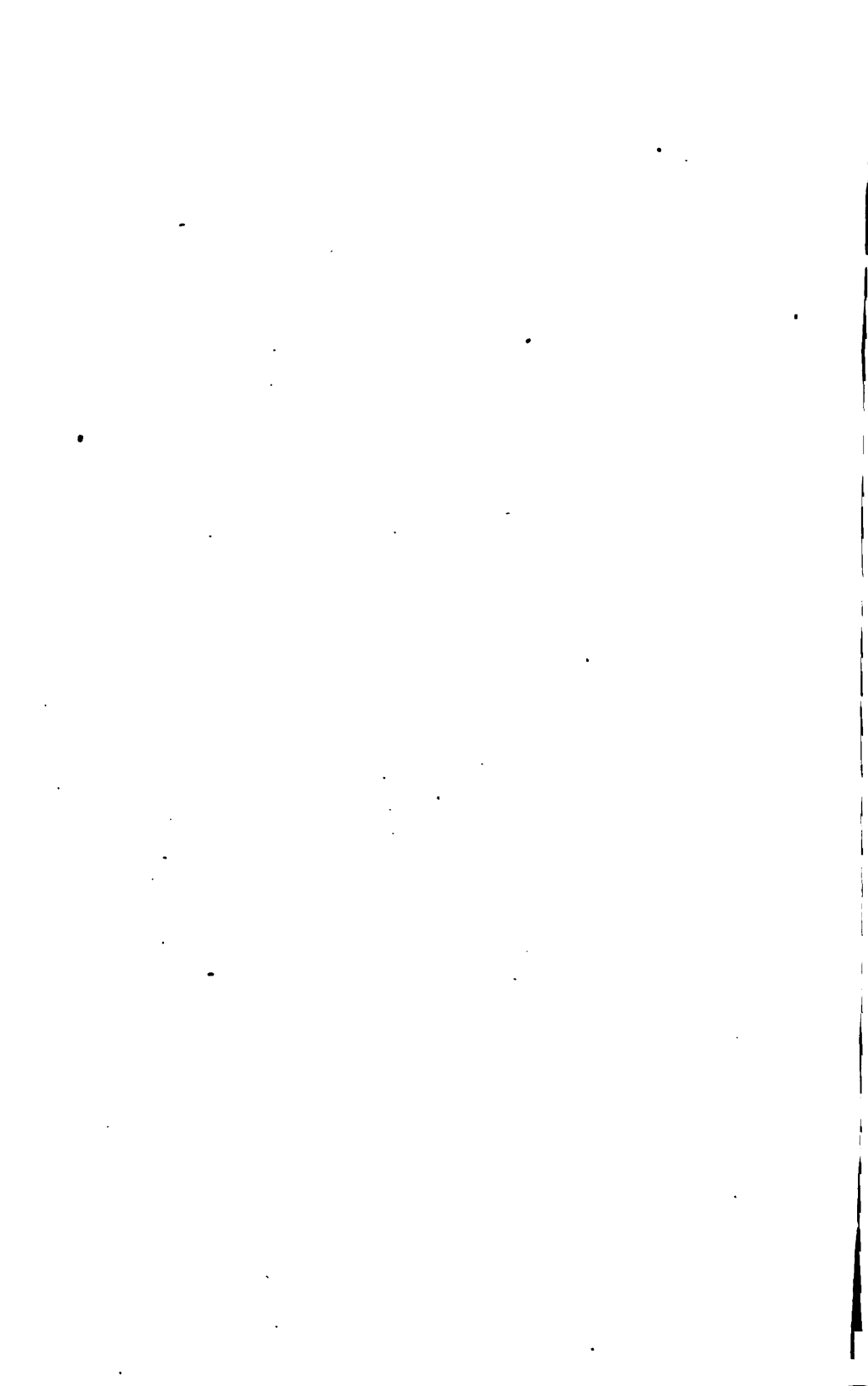
Summary

Crystals of a new silicate of lead and zinc obtained from a lead-zinc furnace slag are described. They are orthorhombic and made up of the prism and the brachypinacoid, and have an axial ratio 0.4334:1:?. . .

Chemical analysis shows the composition of the crystals to be represented by the formula $R''_2Si_2O_6$, in which R'' represents lead and zinc in nearly equivalent amounts. They contain small amounts of magnesia, iron oxide, and alumina.

These crystals can be produced by the slow cooling of a matrix which is more basic than the crystals themselves.

³ *American Chemical Journal* (1910), 47, 273-85.



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The Media Mill, Webb City, Mo.

BY H. B. PULSIFER,* CH. E., CHICAGO, ILL.

(St. Louis Meeting, October, 1917)

THE unprecedented high price of zinc ore prevailing through the early months of 1915 caused great activity in the Joplin district of Missouri. The Media mill is conspicuous as one of the first of the new "big mills" of the district. The mill was built over a mine previously worked but abandoned in 1912 because of the leanness of the ore and the large quantity of water encountered. The ore prices prevailing—\$100 a ton in June, 1915, as against \$50 in 1912—favored the new undertaking. The mill was designed for good recovery so that it might operate in times of depression. In design and equipment, the mill is exclusively local in conception and construction.

Former Exploitation

The tract of land is a part of the large Guinn holdings; 70 acres are comprised in this particular lot, which is situated just a mile north of the center of Webb City on the Alba line of the Southwest Missouri Electric Railway. The ore lies 242 ft. below the surface at the top of the knoll which is surrounded by the well-known properties: Black Cat, Ground Floor, Florence, Hurry-Up, Electrical and Mercantile. Since the abandonment, in 1913, of the Yellow Dog a bit farther to the north, the Black Cat, Ground Floor and Florence workings had been flooded. However, as there was no cut through from this side into the area it was proposed to exploit, it was thought that moderate pumping facilities would take care of the water.

Three shafts had already been sunk to the ore horizon during the previous operations; the ground was connected by wide drifts and possibly 10 acres mined out, principally about the two southerly shafts which at surface are 8 and 13 ft. lower, respectively, than the collar of the shaft at which the new mill was built. The former mill on the lease was known as the Bohemian Girl. This mill, built in 1910, was removed after some 3 years' work. A head frame and hopper still existed at the middle shaft, sometimes known as the lower Hold Out shaft, but at the most northerly

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of the three shafts there was nothing but a boulder pile when the building of the Media commenced on June 5, 1915.

The previous work at mining this particular ground indicated that the ore from a 10-ft. face would mill to recover slightly over 2 per cent. of concentrate. The concentrates were expected to mill to the usual 60

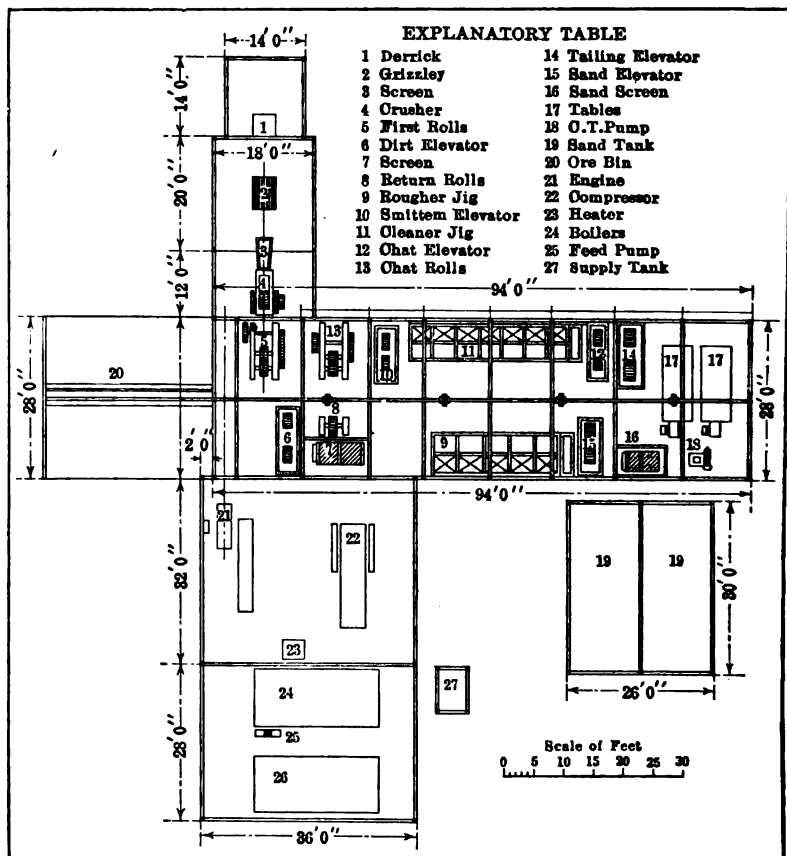


FIG. 1.—FLOOR PLAN OF BOHEMIAN GIRL MILL.

per cent. zinc content, to contain not over 2 per cent. iron, and to be accompanied by some lead.

Figs. 1 and 6 are of interest because they show jig rooms of almost identical size (28 by 100 ft.), but, with widely different capacities, that of the Bohemian Girl mill per 10-hr. shift being only 250 tons as compared with 750 tons for the Media mill. The Bohemian Girl mill is a typical example of Joplin district practice. It has comfortable space only for the usual ore fall from the elevated hopper through the crusher-elevator screen-jig and tailing elevator system with elevation of rougher heads to the cleaner jig. The two tables occupy about an eighth

of the floor space. The boiler plant with steam-engine drive has always been the main source of power in the district and for the Media mill no system was presented which could compete in low cost of operation and reliability. A year's operation confirms the latter statement. These prime features, which are classical in the district, were merely elaborated in the Media. The Media has the hopper raised enough to get a direct ore flow through the crushers and screens onto the



FIG. 2.—HOPPER OF MEDIA MILL ON AUGUST 31.

rougher jigs; this subordinates the first elevators to lifting reground material only. It also avoids free "jack" and fines passing the rolls. Other variations from the Bohemian Girl detail consist in treating ground "chats" and coarse sand on a sand jig after elevation and screening. This, however, is again a practice not foreign to the district; dewatering the rougher tailings has long been partially used, although the screen method has come into vogue only during the last decade.

Building the Media Mill

On Saturday, June 5, 1915, decision was reached to start building the mill and prosecute mining on the lease.

William Kenefick and Paul Easby, as proprietors of the Electrical mine, built the Media out of its profits and acted as overseers of the new operations. Paul Easby, in particular, was often on the ground and procured much of the machinery. James J. McLellan acted as manager, did most of the buying and had personal charge of the pumping and shaft enlargement. The author had nominal supervision of the plans and construction of the mill, although, like the others, he devoted fully half the time to other properties. Claude Raymond was appointed engineer of machinery and pumps, also on part time, and William Turner was employed as carpenter foreman.



FIG. 3.—CLOSE VIEW OF MEDIA MILL FROM TAILING PILE.

We had some discussions about general plans. The author proposed a double wing, duplicate section mill with four 42-in. rolls and maximum choke in the 6-cell, 48 by 42-in. roughers. Mr. Easby held for the most compact setting possible, the least possible number of units, and ore elevators opposite the crushers, as in the Bohemian Girl plan. Mr. McLellan strongly favored ore elevators beside the rolls, these to be used only as elevators for reground material, by getting one continuous fall from hopper to tailing elevator on the main ore course. A combination of the Easby and McLellan ideas prevailed, thus fully establishing the Missouri type. It was decided to get the mill equipment into a room 28 by 100 ft. and have the return elevators at the side of the three 42 by 16-in. rolls which should be below the crusher and allow of an ore fall from hopper to crusher, to screen, to rougher with a drop of only the over-

size to the rolls to be elevated back into the stream between the crusher and trommel (see Fig. 5).

Construction was practically always ahead of what drawing was done for the mill, proper. By June 27 the mill was covered in. During July no spectacular building took place, although the work advanced rapidly on the interior, the office building, the sand plant, the change house, the engine and boilers, the sand tanks and the pond. The 10 by 10-in. hopper timbers having at last arrived on Aug. 2, the hopper was quickly put in shape (see Fig. 2).

The entire surface plant was about 98 per cent. finished when the author's work was completed and he left for Chicago. The plant could



FIG. 4.—TAILING PILE IN MAY, 1916.

possibly have been in shape for actual milling on that day if the underground water had not made unexpected trouble, which held up operations for 7 weeks before the drifts could be entered. The pace on the mill had been slackened, although it had already been turned over to the operating crew which was adjusting jigs, lining spouts, and putting up pulleys and belts. The mill was first run on Nov. 7, 1915.

The Mill Equipment

The mill equipment included:

Two 12 by 18-in. Webb City crushers; 400 r.p.m.

Two 8 by 4-ft. Webb City trommels, extra heavy fittings; 20 r.p.m.; jackets with 0.5-in. holes.

Three 42 by 16-in. Webb City rolls.

Two return elevators, 22-in. cups, 18 in. apart on belts over 30-in. pulleys at 40 r.p.m.

Two 6-cell rougher jigs, Cooley type; $\frac{1}{8}$ -in. grates on 48 by 42-in.

cells; 95 r.p.m., 3-in. drop between cells; plunger stroke graded from 2.5 to 1.0 in.

Two 48 by 48-in. Ford-type dewatering screens, belt suspension; 5 r.p.m.

Two tailing elevators, 20-in. cups on 20-in. belt over 30-in. pulleys.

One smittem elevator, 20-in. cups on 20-in. belt.

One chat elevator, 20-in. cups on 20-in. belt.

One sand elevator, 20-in. cups on 20-in. belt.

One chat roll, 30 by 14 in.; Webb City make.

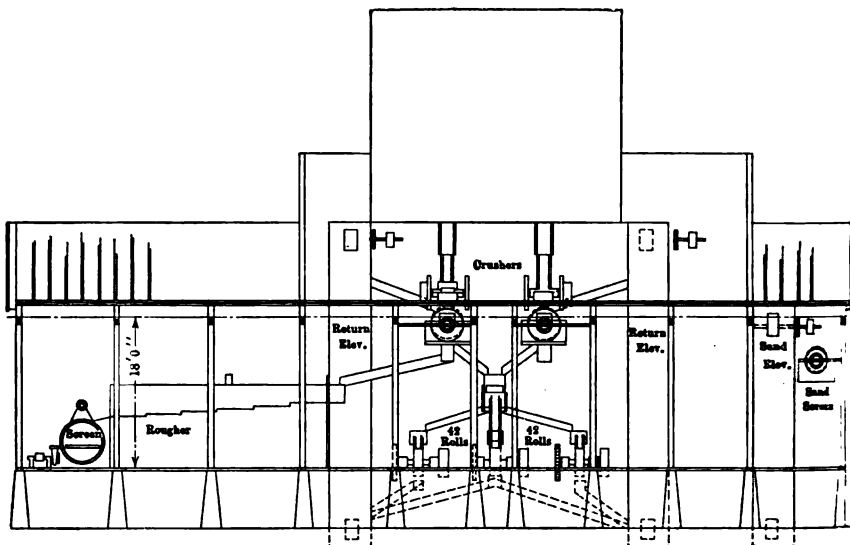


FIG. 5.—LONGITUDINAL ELEVATION OF MEDIA MILL.

One 7-cell cleaner jig, Cooley type; $\frac{1}{8}$ -in. and $\frac{1}{12}$ -in. grate slots on 36 by 48-in. cells; plunger stroke, 140 per minute; 1.5-in. drop between cells; plunger stroke from 1.25 to $\frac{3}{8}$ in.

One 36 by 72-in. sand trommel, 2-mm. holes.

One 3-cell sand jig, Cooley type; $\frac{1}{12}$ -in. grates on 36 by 36-in. cells; 175 strokes per min.; 1.5-in. fall between cells, stroke from 1.25 to $\frac{3}{8}$ in. in length.

One 6-in. Swaby centrifugal pump, 600 r.p.m.

One 2.5-in. American Well Works centrifugal pump to supply water to crushers.

Hoist

Cross compound, 15 by 30-in. cylinders and 48-in. drum steam hoist; 2-ton skips, in balance, capable of round trip each minute; sheaves 110 ft. above ground. Grizzly bars of 60-lb. rail spaced 5 in. apart on incline, 5.5 in. for flat rails.

Sand Plant

Three 20 by 30-ft. concrete tanks, diagonally inclined bottom to outlet 10 ft. deep.

Sand plant building 60 ft. square with flat trussed roof, elevators extending up through roof.

Originally equipped with feed, return and tailing elevators. The return elevator was removed after trial showed the man in charge that the 11 tables did not give enough middlings to bed even one table.

Tables as follows:

Three wooden decks, Lycans and Stilwell.

Two linoleum decks, medium, Lycans and Stilwell.

Three linoleum decks, fine, Lycans and Stilwell.

One linoleum deck, slime, Lycans and Stilwell.

Two Ford slimers, inclined vibration.

Elevators are 4 ft. width with 20 by 30-in. pulleys, direct geared, and belted to make 45 r.p.m.; 16-in. cups on 16-in. belts.

Trommel after feed elevator, 36 by 72 in.; 1.5-mm. holes.

Hindered settling classifiers 1 to 3 ft. deep along overhead trough for first eight tables, wooden construction.

Three 6-ft. thickeners on floor at head of each slime table.

4-in. centrifugal pump.

Tables driven by 4-in. belts over 14-in. pulleys to approximate 245 strokes on coarser tables and 270 strokes per minute on slimers.

Costs

Some of the more general items may be summarized:

Labor, mill construction.....	\$8,100
Lumber.....	8,000
Equipment, general, mostly in mill.....	12,000
Supervision and office.....	1,500
Cement.....	1,200
Pond and ditch to mill.....	1,860
Generator for lighting.....	600
Compressor.....	7,500
Labor, power and water installations.....	1,350
Boilers with fittings (first 4).....	5,800
Brick and clay.....	950
Office supplies.....	70
Tools.....	500
Insurance during construction.....	1,500
Concentrating tables.....	2,400
Painting, material and labor.....	350
Mill engine (first engine).....	650
Hoist engine.....	925
Condenser and pump.....	960
Belting.....	2,500

Approximate cost of plant..... \$58,715

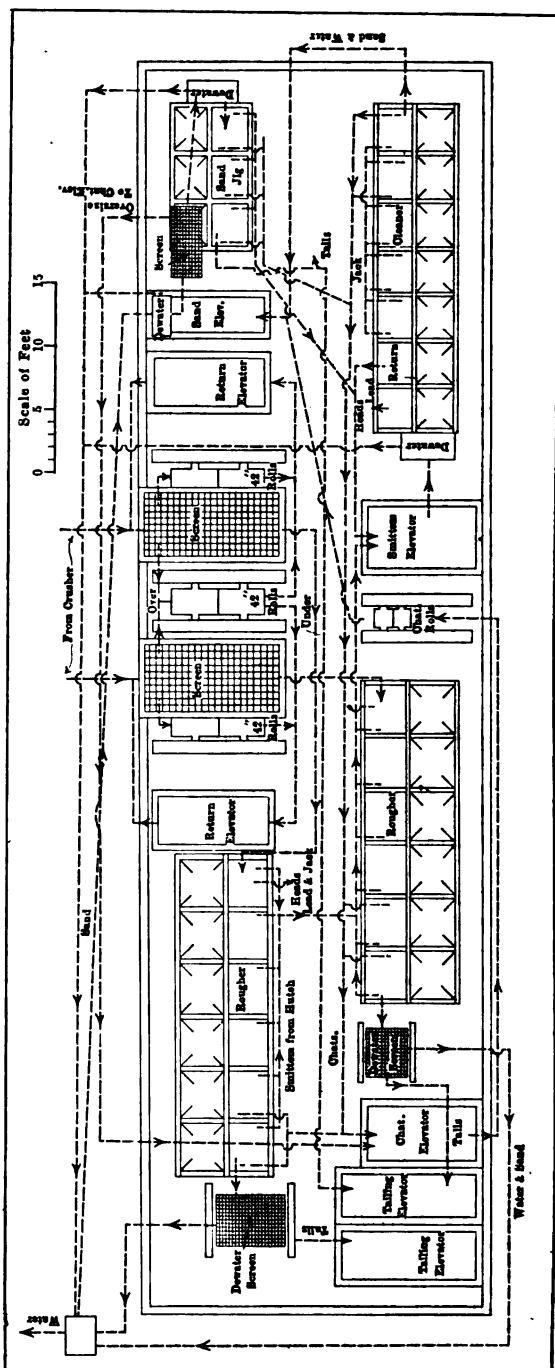


FIG. 6.—FLOW SHEET OF MEDIA MILL.

The above costs comprise the surface equipment. Nearly as much again was expended in draining the mine, enlarging the shaft and excavating the underground hopper for the skips, and for the mine equipment.

Figuring a daily capacity of 1,500 tons and 300 working days a year, we find the plant cost amounts to about \$40 per ton of daily capacity, or about 14 c. per ton of annual capacity.

It is interesting to note that after the mine began to produce a respectable tonnage, the first month in 1916, only about 4 months' operation sufficed to yield a net profit equal to the entire outlay for plant and mine.

On Jan. 26, 1916, the Hurry Up mine operations opened a water course which wholly drained the Media workings. To get water for the mill it was then necessary for the Media to put in pumps and build a flume so as to lift water at the Hurry Up and run it across to the Media pond. This was a startling transition from a huge pumping cost to *remove* water, one day, to an almost equally great cost, the *very next day*, to *get* water!

Summary

The Media mill was rushed to completion during the summer of 1915 on war-time prices of zinc ore, and although the mine operation was delayed over 2 months after the practical completion of the surface plant the whole enterprise proved highly successful from every point of view.

The most conspicuous large mill previously existing in the southwest Missouri section was the Yellow Dog which had run its course only $\frac{1}{8}$ mile from the Media shaft; the Yellow Dog, however, was really two independent mills with a combined capacity of 1,000 tons, per 10-hr. shift.

The Media has continuously ground 1,200 tons in 20 hr. in its single jig room. On occasion, 1,800 tons have been treated without difficulty in 24 hr. The capacity has, up to this writing, been limited more by conditions inside the mine than in the mill.

The Media hopper is by far the largest ever erected in this district. The hopper contains, without leveling, 1,200 tons; it rests on thirty-six 24-ft. posts which in turn rest on a massive foundation joined to the concrete lining of the upper shaft. In my opinion, the derrick is far too light for the load and acceleration commonly used, for the vibration at the sheaves during hoisting is considerable. The upper tier of 10-in. posts was never cross-braced and those on the east side of the hopper, which are the longer of the upper tier, gradually yielded to throw the floor of the hopper toward the hoist which is on the north side. The uprights on the sides of the bin remained vertical, but this edge of the floor had shifted approximately 6 in. before it was noticed during the spring of 1916; the movement then apparently ceased, for the measures

taken could not possibly have restrained further progress in this twisting of the floor.

A large pond was constructed at considerable expense; the enormous output of tailings and sand was allowed to encroach until within a few months the effective pond capacity had been reduced to meager proportions. Such management is characteristic of the district.

The three thickeners in the sand plant were overflowing on one side only, at the time of the author's visit in May, 1916. The settling tanks were designed with the intake at the shallow end so as to permit a continuous flow from the deep corner to the tables—a sort of combination of intermittent tank filling and continuous thickening; the local operator, however, quickly changed to the customary way, filling the deep end with sand and making it a wholly intermittent operation. It has already been mentioned that the return elevator was discarded, the middlings being put in the main stream again. These three instances indicate the strong retrograde impulse which is so highly characteristic of the district.

The mine rock has run fairly uniformly a little over 2 per cent. in mineral recovery. The mineral content has averaged about 4 tons of blende to 1 ton of galena. As the tonnage can probably be kept high and the mill has several strong features for unusual recovery, the venture promises well even during periods of low ore prices.

The best features of the ore dressing may be enumerated:

1. Main tonnage fall direct from hopper to tailing elevator; only oversize is re-elevated. Bad turns in main stream nearly absent.

2. Splendid balance of unit capacity; no one unit crowded or choked while others are running light. If any capacity weakness develops it will probably be in the cleaner, chat-regrinding or sand units rather than in main tonnage units.

3. The cleaner is of unusual size, only a very few of this size ever having been built in the district.

4. Ample facilities for dewatering sands and a sand jig in the main mill to treat cleaner sand and ground chats (settled rougher sand can also come back to this jig) relieves the roughers.

5. The sand plant is entirely separate from the jig room, as is now fairly customary in the district. It has an equipment carefully selected and capable of excellent work.

6. Mill water is elevated no higher than necessary; a second small pump supplies the high line to the crushers.

7. The main tonnage flow is through twin units; this arrangement is happily combined with large single units on the enriched segregate. Both streams being in the closest possible proximity, elevator fronts being open and nearly every unit being in full view, minimum attendance is required and instant response to mishap is possible.

The greatest disadvantage which has come to the attention of the author is the undue crowding in the jig room. The elaborate tangle of shafting, belts, spouts and pipes is very unsatisfactory; it is difficult to get units apart and new parts into place. The main aisle is not wide enough, there is no place for spare parts and tools, or even lockers for clothes.

Full use is made of the principle of equipping only with units carried in stock by local supply houses, repairs are thus most fully assured for instant delivery.

The author fully believes this principle of local inheritance was decidedly overdone. A glaring instance is in the case of the tailing disposal—within a very few weeks the great tonnage had built the pile right up to the spout and the customary “dummy” elevator became necessary. The condition existing in May, 1916, is plainly seen in Fig. 4. Tailing disposal as abundantly demonstrated in various other localities could much better have been installed.

The inevitable inquiry as to the application of flotation is answered by saying that no adequate demonstration of its successful application has yet been made in the Joplin district. At another plant the author made experiments on the recovery of blende from the fine waste sludge; preliminary tests indicated that it would not be difficult to obtain a froth product containing over 40 per cent. zinc. However, the present practice is so firmly established that it will be difficult to convince operators of the desirability of adopting flotation on a large scale.

I wish to acknowledge my indebtedness to the management of the Media company for permission to publish this account; to Mr. Fred Hill for assistance in preparing some of the drawings; and to Mr. J. J. McLellan for especial service in confirming the details of this manuscript.



DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

Characteristics of Zinc Deposits in North America

Their Bearing on Origin and Tenure of Economic Life

BY FRANK L. NASON, M. A., WEST HAVEN, CONN.

(St. Louis Meeting, October, 1917)

THE complete statistics of zinc-ore production in the United States for 1916 are not yet available. The following figures are, therefore, only approximate. The total production of concentrates for this year, including New Jersey and Tennessee, amounts to 1,280,000 tons. Of this tonnage, five mines produced approximately 536,000 tons, or about 42 per cent. The remaining concentrates came mainly from the "Valley States," Missouri, Wisconsin, Kansas, Oklahoma, and Arkansas. The 1,280,000 tons of concentrates represent about 25,000,000 tons of crude or run-of-mine ore. Of the five mines producing the 42 per cent., the dimensions of three are approximately 25 by 1,500 by 2,000 ft. (7.6 by 457 by 610 m.) or more. Four of the five mines are undoubtedly veins in the common acceptance of the term. They are located in great vein systems. The mines of "Valley States" have not, so far, been classed as veins; simply as "secondary deposits." The origin of these secondary deposits has been a warmly debated subject. Many geologists contend that they are the results of leaching from overlying and from adjacent rocks carrying but traces of zinc, these traces leached out and deposited in favorable loci. If this theory is correct, deposition is strictly limited to shallow depths. No volume of ore comparable to that of "true fissure veins" is to be expected. Just what this means to the future of zinc mining in these States is here indicated. The total zinc areas of these States amounts to about 9,124 sq. miles (23,631 sq. km.). Assuming that 10 per cent. of this total will be more or less productive, there are 912 sq. miles (2,362 sq. km.) of "mineral" ground. The average depth of the operated mines is something like 100 ft. or less. In *Bulletin 606, U. S. Geological Survey* (1915), p. 202, Siebenthal makes the following statement: "The writer is informed by William Waugh, one of the drillers of the Stone City well, that cuttings from other wells which he drilled in the vicinity of Pittsburgh, Kan., have shown galena and sphalerite taken from a depth of 591 ft. (180.14 m.) and that the showing of ore at some places would be considered good enough to justify the sinking of a shaft had it been found in the Joplin district and at shallower depths." Leaving out of the question the prohibitive cost of such a shaft, there is

means the total past production plus orebodies that may yet be found at the present average depth. In the body of this paper I have cited examples of drill holes having cut zinc ores at a depth of over 1,000 ft. (304.8 m.) below base level of drainage. Ores occurring at this depth below base-level drainage could hardly be accounted for by downward-flowing waters; this, according to Mr. Siebenthal, is not necessary. In the *Bulletin* already referred to, p. 42, he says: "The writer believes that, directly or indirectly, the Cambrian and Ordovician limestones and dolomites have furnished the major part of the ores now found in the Joplin district; that these ores were precipitated from ascending alkaline-saline sulphureted waters."

Mr. Siebenthal's theory is not new; he was anticipated by 28 years. Walter P. Jenney¹ comes to the following conclusion: "No good reason appears for placing these ore deposits (Valley States) in a separate class from fissure veins because of distinctions based upon the form and position of the orebodies relative to fissures. On the contrary, these runs, considered in connection with the fissures through which they were formed, belong to the great class of ore deposits in which the origin of the ore is in mineral-bearing solutions, ascending through the fissures from some source of unknown depth in the crust of the earth, of which class fissure veins are the simplest typical form." Both Jenney and Siebenthal base their conclusions on long and close study of the mines in the fields of the Valley States. The recorded conclusions of two such hard and conscientious workers seem to make this paper superfluous. I can add little, if anything, new to their evidence. The most that I can expect to do is to place my years of experience in the zinc fields of the United States, Canada, Mexico, Guatemala and San Salvador in a different manner. I record the results of my observations under the following heads: Formative Influence of Eruptives; Contact Phenomena; Fissuring and Faulting; Minerals Accompanying Fissure Deposits without Eruptives; Minerals Accompanying Fissure Deposits with Eruptives; Zinc-Bearing Fissures in Limestones and Other Rocks Accompanied by Eruptives; Contact Zinc Deposits; Secondary Enrichment of Zinc Deposits; Depths at Which Zinc and Lead Deposits Have Been Found.

The conclusions which I have personally drawn from the above phenomena seem to strengthen the position taken by Jenney and Siebenthal that fissuring and faulting are the prime prerequisites for the formation of zinc-ore deposits; Jenney seems to go further than Siebenthal in that he concludes that the zinc-ore deposits of the Valley and Appalachian States are, in effect, fissure veins with all of the essential characteristics of this class of deposits; that in no essential do they differ from the recog-

¹ *Trans.* (1893), **22**, 217.

nized veins of the Rocky Mountain region. My observations compel me to follow Jenney.

Formative Influence of Eruptives

The greater number of zinc deposits in the Rocky Mountain region in British Columbia, the United States, Mexico, Guatemala and San Salvador are characterized by the presence of eruptives. This is also true of the zinc-lead deposits accompanying the fluorspar deposits of the Kentucky-Illinois field. In Butte, Mont., the ores of the great zinc mines of the Butte and Superior, the Elm Orlu, Lexington, Alice, Moulton, Emma and Nettie and other minor deposits, occur in great fissures in the granite. The eruptives, aplite and rhyolite, seem to have occurred later than the mineralization of the zinc veins. Certainly many of the copper veins have been broken and displaced by the intrusion of later dikes. The later eruptives in this district thus seem to have played no part, even indirectly, in the zinc mineralization.

In the Baker-Neihart district the silver-lead-zinc veins are in fissures in probably pre-Cambrian gneisses and in Paleozoic limestones accompanied by eruptives in the form of dikes. In this district evidence as to formative influence of eruptives is neutral. In the Troy district in northwest Montana, the lead-zinc deposits occur in fissures in the pre-Cambrian slates. The same is true of the Highland-Surprise and the Douglas on Pine Creek near Kellogg, the Interstate-Callahan and the Success near Wallace. In these mines the country rock is faulted heavily, and in all of them occur dikes and bosses of quartz monzonite and dikes of other eruptives.

The present working adit of the Interstate-Callahan is 1 mile (1.60 km.) long. It followed in on a thin leader of zinc-lead. At a distance of about 1 mile it cut through a trap dike and picked up the great vein on the far side. One wall of the shoot was slate, the other quartz monzonite. Heavy faulting as evidenced by the juxtaposition of two distinct sedimentary horizons, is present.

In Madison, N. H., a wide vein of silver-lead-zinc occurs in a fissure in the granite. No later eruptives are noted in this locality. In the zinc fields of Missouri, Kansas, Oklahoma, Illinois and Wisconsin, and of eastern Tennessee, Virginia and Pennsylvania, the ore deposits are characterized by heavy faulting and are also remarkable for the entire absence of eruptives of any kind.

The presence of fluorite, supposedly an evidence of the presence of eruptives, is found sparingly at Austinville and Cedar Springs. In one Virginia locality an undeveloped deposit of zinc is characterized by the fact that at least 50 per cent. of the gangue is fluorspar. The color of the latter is the characteristic deep purple. It may be that white fluorite occurs and has escaped detection in the calcite and bitter-spar gangue,

but if it occurred in any great quantity its presence would have been noted by the smelters. Fluorite is the only hint of the possible presence of eruptives in the vicinity of the zinc mines.

The Hanover, N. M., zinc deposits are, in places, accompanied by rhyolite and basic dikes in addition to the great body of porphyry directly east and west of the railroad.

In Pembroke, Maine, there are a few remarkably rich pockets of blende in persistent fissures in diorite. On Deer Isle, at Cape Roxier, are numerous lenses of rich zinc-lead carrying silver. These probably occur in pre-Cambrian sheared eruptives. These deposits are accompanied by diorite dikes.

In the Charcas district, San Luis Potosi, Mex., which almost identically reproduces the Hanover district, numerous acid and basic dikes are present in addition to the basal porphyry. These cut the orebodies in several places, as a rule, displacing them. At Charcas, in the highly metamorphic hill known locally as "La Bufa," numerous acid dikes cut the orebodies, seemingly later than the orebodies themselves. Feldspar porphyry is an abundant rock. It appears flanking the limestones, and in dike-like masses in the limestones. Whether the porphyry is really an eruptive or only has the appearance of it, depends wholly on whether it is basal or not, taking the appearance of an eruptive by means of faulting. The latter explanation appears to me to be the only tenable one. This much is certain in either case—no important mineralization is found either at the contact with bedding planes or at the contact of apparent dikes. This condition is eminently true at Hanover.

Near Wallace, Idaho, one prospect, carefully examined, showed a zone about 7 ft. (2.13 m.) wide in which three stringers of blende could be seen. These occurred in pre-Cambrian slates. No eruptive was nearer than 100 ft. (30.48 m.). Crushed rock evidenced faulting. On the slope of the hill was a small outcrop of quartz monzonite.

Typical illustrations of the lack of effect of eruptives on mineral deposition and metamorphism are shown in Fig. 6. West of the Nason tunnel, dikes and sills exist with neither effect. A dike occurs cutting through both the orebody and the garnet at the east end of the above tunnel. Evidently the dike could have had nothing to do in this place with the mineralization or with the metamorphic garnet. Fig. 5 seems to present even stronger positive evidence. At the western end of the main tunnel a block of ore and limestone is cut off entirely from the main body by two dikes; farther in, the ore zone is faulted in contact with dikes, probably because of them. Here again the eruptives are younger and have no formative effect. These, as was stated, are types selected from a large number of similar examples.

To sum up briefly, in some of the cases cited, eruptives are to be noted; in every case, faulting and fracturing occur. The only conclusion to be

drawn seems to be this: Whatever the formative influence of eruptives on mineral deposits may be, they are not absolutely essential. On the other hand, faulting and fracturing are essential. Certainly this is true: In very many instances eruptives have followed mineral deposition; not preceded it.

Contact Phenomena

It seems to be the general impression that when an eruptive cuts across bedding planes of sediments, cuts through an older eruptive, or interpolates between sedimentary planes as blankets or "mantos," the plane of contact is a favorable locus for mineralization. This mineralization may take the form of economic minerals or metamorphic minerals not useful.

Indirect negative evidence is more than suggested in the paragraphs immediately preceding. Eruptives are not universal accompaniments of ore deposits. Direct evidence of the lack of metamorphic influence of eruptives alone is as follows:

In the San Carlos mountains, about 90 km. (55.9 miles) southeast of Linares, Mex., are copper mines which have been worked extensively. The mountains are comparable to a great volcanic crater; the village of San Carlos lies in the cup of the crater. The crater rim on the north, west and east, as well as large areas in the cup itself, are massive limestones. Within the cup, and in places on the rim, are immense masses of solid garnet. In this garnet the copper veins are found. In more or less intimate relation with this garnet are dikes and, possibly, eruptive bosses. The eruptives have been supposed to be laccoliths, mushroom-shaped eruptives, where the volcanic neck is the stem and the eruptive, spreading from this neck, the umbel. It has been assumed that the garnet and the cupriferous veins were the results of contact.

A large part of the cup is soil-covered; the rim, bare rock. Careful study of several miles of the rim disclosed the following facts: Starting on the unchanged blue limestone, and following along the exposed strata, white spots began to appear and along the bedding planes a slight bleaching was noted. The bleaching extended through the entire mass; the limestone changed to a white, crystalline marble; then clots of amorphous garnet; finally, great, vertical, dike-like masses of pure garnet. Still following along the line of strike, the changes were reversed until another garnet dike was approached. The above phenomena were observed on the three limestone rims. For miles, between the garnet dikes, absolutely unchanged limestone lay above and below the intrusive sheet and in absolute contact with it.

At Charcas, San Luis Potosi, Mex., are two mines, the "Tiro General" and "La Bufo." These mines lie in a U-shaped valley whose axis runs nearly due east and west. The mines are about 2 miles (3.22 km.) apart.

equally divided between a coarsely crystalline feldspar porphyry and a blue limestone.

The Tiro General is located on a well-defined east and west fissure. La Bufa is not certainly located on a fissure. The orebody, zinc and lead, is, so far, confined to bedding planes that dip to the north at an angle of about 45° . The hanging wall of the orebody has not been definitely located; the foot wall appears about 1,500 ft. above the outcrop and on the slope of the mountain. It has also been located immediately under the orebody between the fourth and fifth levels in the Malacate shaft. (See Fig. 8, "Probable Foot Wall.") Within the great orebody are masses of garnet and epidote with equal masses of wollastonite. These rocks, especially the garnet, carry large masses of pure hematite. Along the east and west strike and up the slope of the mountain is what was formerly a limestone, now entirely changed to massive wollastonite. Brecciated porphyry carrying interstitial blende is found a little above the fifth level (Fig. 8) and below. The limestone is unchanged.

At the extreme east end of the first level, a supposed porphyry mass was driven through, seemingly indicating a dike rather than a massive body. The rocks, however, are so rotted that on the east ends of levels 1 and 2 positive identification is impossible. A porphyry contact indicated as possible is thus unproved. The garnet, hematite and wollastonite noted in the section cannot thus be certainly ascribed to contact reactions. Fissuring, only, is absolutely certain. For a long distance around the mine, the chert of the limestone is changed to the same mineral. The same mineral with crystalline quartz occurs between bedding and joint planes. For 2 miles or more west of La Bufa, the above conditions obtain except that zinc and lead ores are interrupted along the line. The western end of the valley is terminated by a connecting north and south ridge that unites the two ranges. Where the axial line of the Tiro mine crosses the ridge, there is an immense dike-like mass of garnet and wollastonite rock. North and south of this dike the limestones are unchanged, though there are many eruptives in the form of dikes. Numerous draws on either flank of the valley expose sills of eruptives in immediate contact with both unchanged limestone and slates. There is a perfect exposure of both upper and lower contacts, but at no point is there any sign of metamorphism or mineralization. In Animas Forks, Colo., is a great fault fissure 100 ft. (30.48 m.) thick. At one end is the Sunnyside mine; at the other, the Gold Prince. Both of these mines have free gold and this is accompanied by sulphides of zinc, lead and copper. There is also much iron pyrites with rhodocrosite in a gangue of crystalline quartz. The mines are about 3 miles (4.83 km.) apart. Between, the vein is exposed as crystalline quartz with much silica in the colloidal form. In

small bunches, galena and sphalerite occur with rhodochrosite. These are of uneconomic size. I saw no eruptives in the locality. The throw on the fault vein is, as I recall, not less than 3,000 ft. (914.4 km.). The vein filling was, in the less metamorphic portions, composed of partially digested fragments of the original rock breccia. At Breckenridge, Colo., limestones and shales are exposed in the various gulches with sills of eruptives in immediate contact but with no metamorphic changes. The mines here are sometimes located in faults or fissures in the eruptives, often at seeming contacts, but invariably accompanied by faulting. Judging by cited examples—and localities have by no means been exhausted—simple contact between eruptives and sedimentaries, between older and newer eruptives, both metamorphism and mineralization depend either wholly or in part upon other phenomena than simple contact. In the case of the great vein near Animas Forks, it appears to be evident that the metamorphism here exhibited depended upon causes wholly independent of eruptives. We can go a step further and say that there are many zinc and lead deposits that are wholly independent of eruptives, apparently depending upon fissuring alone. The citations from the San Carlos mountains and the Charcas district in Mexico, at the most, can only be put into the neutral class since, in the instances cited, the extreme metamorphism is invariably accompanied by fissuring or faulting and nowhere else along the line of contact is any metamorphic change to be noted. The same conditions prevail to a striking extent at the zinc mines of Alotepec in Guatemala and near Metapan in San Salvador. In the last cases cited, veins in the same eruptives that form dikes in the limestones and occur in the zinc-lead veins, are filled with sulphides of lead, zinc and copper. There can be no question in this case but that the minerals in these veins are later than the eruptives; by natural inference, if not by direct evidence, these same eruptives had nothing to do with the mineral veins in which they occur. In the Malacate shaft at La Bufa mine, at a depth of about 700 ft. (213.36 m.) the shaft cuts through the foot-wall limestones that appear under the zinciferous blankets. At about this point in the shaft brecciated porphyry appears. The interstices are filled with blende and galena. The dike is nearly vertical and until the dike is passed through the lead and zinc minerals are found. Below the dike (in the shaft) lead and zinc appear in brecciated limestone.

In the "southwest opening" of the New Jersey Zinc Co.'s mine at Mine Hill, N. J., a large trap dike cuts the willemite-zincite-franklinite orebody in an east and west direction. Between the dike and the regular orebody there is a more or less persistent vein consisting mainly of cleiothane, a white sulphide of zinc. It is accompanied by a considerable amount of fluorite. This occurrence is to be contrasted with an occurrence of honey-yellow blende in a fault plane at the Stirling Hill mine. No eruptive accompanies this last occurrence.

In eastern Tennessee and southwest Virginia are many striking examples bearing on the above subject. The most noted example, owing to its successful commercial development, is the Mascot mine about 14 miles (22.53 km.) east-northeast of Knoxville. This mine is located on a brecciated fault line that can be almost continuously traced from Eves Mills in Loudon County to Talbot in Hamblen County, a distance of about 75 miles (120.7 km.). Zinc and lead are found at Eves Mills, Unitia, Friendsville, Phillips farm near Caswell, Loves Creek, McMillan, Roseberry, Mascot, New Market and Mossy Creek on this line. At Talbot, heavy beds of breccia occur but so far as known they are wholly barren. Mascot is the only locality that has developed blende in economic quantities. New Market, Emberville and Mossy Creek have produced and are now producing a considerable amount of oxidized zinc. Fairly extensive drilling in these last three localities has, so far, failed to reveal economic blende. Intense brecciation along this line is self-evident. The evidence of faulting is more or less indirect. Both foot and hanging walls are of Knox dolomite and, except for position, indistinguishable. United States Geological Folios covering this field show overthrust faults, in many localities the fault plane being almost parallel to the dip of the strata. Local identification of a fault at any given point is thus almost impossible. Only by following the strike line of brecciation to points where readily differentiated rocks occur unconformably can the overthrust fault be established. A specific instance on, or near, the line of the Mascot brecciation is west-southwest of Morristown where the Rome formation, normally about 2,000 ft. (609.6 m.) below the Knox, is thrust over this latter formation. The fault plane thus represents a vertical lift of 4,000 ft. (1219.2 m.), more or less, and the break must thus extend far below the Knox formation.

In Hancock County, south of Sneedville, there are deposits of zinc and lead, both sulphides and oxides, for a strike distance of over 7 miles (11.26 km.) along Comby Ridge. The ridge is cut by deep draws at right angles to the strike. In these draws zinc and lead are found along evident slip (not to say fault) planes, along both bedding and jointing planes. Between Comby Ridge and the Clinch River are several distinct strike faults where the Marysville limestone is brought up against the Conasagua shales. Blende is found along these fault planes also, impregnating the limestone for a considerable distance from these planes. The Marysville limestones here (west of Comby Ridge) rest directly on the Briceville shales (Carboniferous) establishing a fault with a vertical lift of not less than 3,500 to 4,500 ft. (1,066.8 to 1,371.6 m.).

In Union and Claiborn Counties, Tennessee, are numerous fissures running nearly due east and west. In places these fissures are parallel

and are separated by about 100 ft. (30.48 m.) (see Fig. 2, cross-section of New Prospect mine). The rock between the fissures is more or less brecciated. In other places, as at Goin, a single smooth-walled fissure from a few inches to more than 3 ft. (0.9 m.) thick is observable. The fissures are occasionally filled with plates of solid zinc carbonate or silicate. In these counties are many examples of these two types, single and double fissures. No one of them lacks more or less mineralization. From personal examination, these fissures extend in an east-northeast line for 25 miles (40.23 km.) and east and west for at least 15 miles (24.14 km.). Not certainly the same fissure, but at least the same line, has been traced for over 75 miles (120.7 km.), from the eastern foot of the Cumberland Mountains through to the foot of the Appalachians. Near New Market in Shenandoah County, Virginia, there are three parallel sets of fissures. One line is picked up about 1 mile (1.6 km.) south of New Market and can be traced for over a mile; the other two are about 5 miles (8.04 km.) to the west. The Shenandoah River seems to flow along the eroded crest of an anticline since the inclosing limestones dip to the southeast and northwest respectively. I personally followed two of these fissures for 4 miles (6.43 km.). I was told that they could be traced continuously for 4 miles additional.

In Virginia the most important fault occurs on the New River in Wythe County. The New River here, as at New Market, flows along the eroded crest of an anticline. The fault line can be followed more or less continuously from Ivanhoe to near Allisonia, a distance of over 20 miles (32.18 km.). Two large zinc mines have been proved on this line—the Austinville and the Bertha. There are several other localities with favorable zinc showings, but these have not been developed. The Austinville mine is located on a fault striking about N. 45° E. It has a vertical throw of not less than 3,000 ft. (914.4 m.) (see Fig. 1). The fault line proper cannot be seen nor can it be exactly located, owing to the mantle of clay. The fact that there is a fault with the given throw is established by the identity of the “ribbon” limestones abutting against the dolomites at the “Stamping Ground” (see Fig. 1) with the ribbon limestones underlying the dolomites at New River at Thorn’s ferry. Without details, I merely call attention to fault lines in Pennsylvania, notably at Friedensville; blende-bearing fissures or faults in the white limestones in Sussex County, New Jersey. The Cruz del Aire and Dulces Nombres mines in Nuevo Leon; the Bajan and Encantada zinc-mining districts in Coahuila, Mexico, are districts in which extensive folding with resultant faulting has taken place. The mountains in which these mines occur are limestone. In the Bajan district the underlying sandstones are exposed and the mines occur a comparatively few feet above the sandstones in the limestones. In the Dulces Nombres district, shales show at the base of the limestone formation. The zinc mines themselves occur at

the very summit of the mountains, 10,000 to 14,000 ft. (3,048 to 4,267.2 m.), geologically, above the shales.

This much should, however, be added. I examined the zinc deposits in the above-mentioned fields but had time to examine the limestones only in the immediate vicinity of the mines. There might have been eruptives back of the outcrops and on the far slopes of the mountains, but extensive excursions through the Sierra Madre Oriental Mountains in Tamaulipas and Nuevo Leon showed this range to be remarkably free from eruptives. One cannot speak positively on this point without careful local investigation. This much is certain. In many respects the zinc deposits in the above sections of Mexico resemble very closely the fissure deposits of eastern Tennessee and southwest Virginia.

In Wisconsin, Missouri, Kansas, Oklahoma and Arkansas, zinc and lead fields are often cited as examples of large areas, not only notable for the lack of eruptive rocks, but of faulting and fissuring as well. On p. 426 of *Mineral Deposits*, Lindgren gives a typical illustration of "flats" and "pitches" as well as of disseminated lead and zinc in Wisconsin. The illustration is credited to T. C. Chamberlin. The generalized illustration by Prof. Chamberlin shows little or no folding that might account for the "pitches;" while the underlying glass rock and oil rock shows no disturbance whatever. Fig. 9 is a sketch which I made on the ground at the Kennedy mine near Hazel Green, Wis. The axis of the mine runs north-northwest; the section is made at right angles to the axis. The rolls A and B are from 10 to 15 ft. (3.05 to 4.57 m.) high and in this particular section, at F-F, are faults with throws of from 18 in. to 2 ft. (0.46 to 0.61 m). One can pass entirely through the Kennedy mine into the workings of the Winnebago on the north and find the same structure almost continuously wherever the workings have gone down to the glass rock.

In every mine that I examined in Wisconsin, similar structures show rolls in the glass rock with minor faults and slips which show more or less in the overlying galena formation. I quote from a formal report which I made on the Wisconsin zinc field in 1909: "In the underground study of exploited mines one will note that the richest ore occurs in zones of broken rock, locally known as pitches; that this ore goes from the pitches into flats, but only where the rocks are more or less warped and disturbed; that in the glass rock acute rolls 15 ft. high, more or less, are faulted; that these fault planes pass into the galena limestones above and into the underlying rocks."

The axes of these rolls conform to the direction of drainage streams. By studying the topographical maps of the Wisconsin Geological Survey one will note that the Platteville limestones, the St. Peters sandstone and the Prairie du Chien limestones occur in a manner proving that the rolls noted in the glass rock in the mines belong to the same general system

of folding. Fig. 10 shows a measured cross-section through the Kennedy mine on the glass-rock horizon. Faulting is very evident in the Arkansas zinc field; it is evident on the surface as well as underground in the Oklahoma field.

In the disseminated lead fields of southeast Missouri, extensive and heavy faulting is readily noted underground and is often plainly to be noted on the surface. The lead deposits are closely associated with these faults.

Minerals Accompanying Fissure Deposits Without Eruptives

At the Mascot mine in Tennessee, while the foot and hanging wall dolomites are, in places, slightly bleached, there is hardly a perceptible change from the general light-colored, crystalline structure of the Knox as a whole. The brecciated fragments seem to be bleached in direct proportion to their size. The larger masses are bleached on their rims. The smaller fragments are sometimes more coarsely crystalline than the rock mass but the fragments are still recognizable as such. The interstitial filling is almost wholly calcite and bitter spar. The blende, which is practically the only other mineral, is sometimes attached to the rim of the fragments; more rarely, imbedded in the interstitial crystalline filling.

At the New Prospect mine, the heavier beds at the base of the deposit are broken and warped but not brecciated. There is considerable interstitial secondary dolomite. The metallic sulphides in the order of their abundance are sphalerite, galena and iron pyrites. The lead and zinc occur in fractures in the dolomite and along jointing and bedding planes; also in the main north and south vertical fissures. In these loci the lead and zinc minerals are very coarsely crystalline. Iron pyrites occur in small crystals. There are dolomite strata from a few inches to 3 ft. (0.9 m.) in thickness where zinc is 50 per cent. (sphalerite, 74 per cent.) of the weight of the mass. Other strata have no more than 1 per cent. or less of zinc. The blende in these strata is very fine-grained, seemingly replacing the dolomite molecule by molecule. The replacement seems to have eaten into the dolomite mass in spherical form, entering through capillary fractures. The center of the sphere runs high in zinc, fading to zero on the circumference. At the New Prospect neither iron pyrites nor galena appears in the replacements.

At Straight Creek the orebody seems also to lie along a fault plane, but there seems to be but a single fissure at this mine. As at New Prospect, the dolomites carry the orebody, which lies at the base of the Knox and is in contact with the "ribbon" limestones. The Conasagua shales are close by but not in visible contact with the ore. The entire orebody seems here to be a replacement and the blende is fine-grained and intimately mixed with fine crystals of iron pyrites and galena.

At the Caldwell mine, about 3 miles (4.82 km.) above the New Prospect, on the surface the ore appears exclusively in fissures from $1\frac{1}{2}$ in. to 20 in. (3.17 to 508 mm.) thick. Here blende, galena and iron pyrites occur intimately mixed in fine grains and masses, the vein filling occasionally running upward of 90 per cent. combined sulphides. By diamond drills these fissures were followed down to a depth of 1,200 ft. (365.76 m.) below the level of the Powell River. As long as the cores showed fissures, the mineralization continued as above. At about 1,100 ft. (335.28 m.), one drill core showed 1 ft. (0.3 m.) of nearly solid blende with no galena or iron pyrites. The core showed the ore as a replacement of horizontal strata of calciferous Conasagua shale. Three drill holes showed similar cores in the same kind of rock but there was a difference of about 100 ft. (30.48 m.) in elevation. These facts seemed to point to faulting and displacement along the fissures but no signs of this were to be noted on the surface. The broken line in Fig. 3 shows the contour of the Conasagua shales, as revealed by diamond drills along a line of 45,600 ft. (13,898.88 m.). The contour may be faulted instead of corrugated by folding, as indicated by the figure.

In the Atustinville mine, generalized mineralization is shown in Fig. 4. The mineralization shown in strata I, II, III and IV has been proved to extend vertically for 435 ft. (132.58 m.) by a shaft (see also *a-b-c*, Fig. 1). From the surface to the 235-ft. (71.63-m.) level several strata of oxidized ores have been worked continuously and drifts from the 335-ft. level and ore at the bottom of the 435-ft. shaft line up with these strata on the surface. Mineralization occurs on distinct bedding planes (see Fig. 4. *b-b*); on jointing and fracture planes (*c-c*) and, though not shown in the figure, whole strata seem to have been replaced in brecciated, highly crystalline bodies. The orebody, aside from the gangue, is made up of sphalerite, galena and iron pyrites. The vein filling aside from the metallics is mainly calcite, dolomite crystals and, very rarely, scattering crystals of purple fluorite. The sphalerite and galena occur in the ratio of about 3 to 1, are fairly coarsely crystalline in the main. The iron pyrites occur both as fine-grained crystals with the blende and galena, frequently in great masses entirely replacing the other metals. In these masses lead and zinc runs from zero through traces up to an economic per cent.

As at New Prospect, when the blende and galena occur along bedding joint and fissure planes, both minerals are coarse-grained. In general, where the limestone is replaced the mineralization is fine-grained. In places in the mine, a large part of the galena occurs in dense masses of highly crystalline, secondary, dolomite in filaments after the manner of mushroom spawn in the "brick seed" of the market. In the Forney and Ivanhoe deposits in Virginia and in the Friedensville mines in Pennsylvania, minerals are about as described in the cited cases. The

main differences between the minerals of the eastern deposits described and those of the "Valley" States seems to lie in this. In the "Valley" States the iron is mainly marcasite and on crystals of both blende and galena there are often implanted crystals of tetrahedral copper.

Barite occurs more frequently in zinc fissure deposits without eruptives than in fissures where eruptives are present. Near Rhodelia in Union County, Tennessee, there is an outcrop where zinc in the form of blende is 15 per cent. In the gangue, more than 50 per cent. is barite. It is a fairly prominent gangue mineral in the White Pine belt that reaches northeast from White Pine to Mosheim. On the Mascot fissure, beginning near Kiser, barite is found with zinc blende as far south as Sweetwater. About here the barite becomes the dominant mineral to the practical exclusion of zinc especially.

Barite is not infrequently a gangue mineral in some of the zinc deposits in the Ozark region, Missouri.

It has been mined in Wythe County, Virginia, associated with galena and sphalerite, but, so far, no commercial deposit of zinc has more than traces of barite.

In the Kelly district, New Mexico, barite is found with zinc and lead in considerable abundance above the Kelly-Graphic fault only.

Minerals Accompanying Fissures in Eruptives

The Butte and Superior and other zinciferous veins in Butte, Mont., seem to carry silver sulphides in addition to the silver in the galena. According to Walter Harvey Weed:² "The minerals of the Butte ores are neither numerous nor rare, nor of great variety." Chalcocite, bornite, enargite are the most common copper minerals. Tetrahedrite, pyrite, native gold and silver, marcasite, sphalerite, galena, pyrargyrite, hubnerite, rhodonite, rhodochrosite, silica, crystalline and colloidal, occur more or less abundantly.

In the Silver Pick mine near Mt. Wilson, Colo., a narrow vein carried gold up to 5 oz. or more per ton, silver up to 18 oz. in associated minerals, galena, chalcopyrite, stibnite, tetrahedrite, iron pyrites and a very black sphalerite. The gangue mineral was mainly quartz. When sphalerite and pyrite began to replace galena and chalcopyrite to any great extent the gold and silver vanished. Zinc in this part of the veins was 20 to 40 per cent. The vein was in a great diorite dike.

In the Alotepec mines in Guatemala, zinciferous fissures in rhyolite dikes carry, in addition to a pure blende, high-grade argentiferous galena and chalcopyrite. No other minerals were observed save the gangue, consisting mainly of calcite.

²Geology and Ore Deposits of the Butte District, Montana, *U. S. Geological Survey, Professional Paper* 74 (1912), 72.

In Madison, N. H., a large vein in granite carries coarse blende, argentiferous galena and some iron pyrites. The gangue is mainly epidote, derived apparently from metamorphosed feldspar. No other minerals were observed.

Zinc-Bearing Fissures in Limestones and Other Rocks, Accompanied by Eruptives

From an economic standpoint, silver is the dominant economic element in the above class of orebodies. Up to 1901 the presence of zinc sulphide in ores of this class prevented the operation of many of these mines. The silver content of the galena concentrates from these ores sometimes ran as high as 1,500 oz. per ton. In many cases sphalerite as well as galena is argentiferous. The presence of silver in these ores is in sharp contrast to the lack of silver in fissures unaccompanied by eruptives. In southeast Missouri the silver content averages about $\frac{3}{4}$ oz. per ton of pig metal. The galena in the zinc fields of the Joplin region and in Wisconsin has about the same silver tenor. The visible difference between the above classes of deposits and fissures without eruptives is equally striking. Metamorphic minerals, such as wollastonite, zoisite, hornblende, garnet, epidote and scheelite, may be present. Hematite is an abundant mineral at Hanover and Kelly (in the Graphic mine only), N. M.; Fierro, Nuevo Leon and Las Charcas, San Luis Potosi and also at the Calera mine in Sonora, Mex. Vanadinite, descloizite, wulfenite and pyromorphite are among the lead minerals; rhodonite and rhodocrosite are two manganiferous minerals; hubnerite is found in Butte, Mont., mines, in mines in the Silverton district and in the vicinity of the Poland mines in Arizona. Tetrahedrite and stibnite are both very common antimony minerals; also many copper and silver sulphides, antimonides and arsenides. Native gold is found certainly in sphalerite from the Tomboy mines near Telluride and from the Gold Prince and Sunnyside near Silverton, Colo.; it has also been found intergrown with galena and arsenopyrite at many other zinciferous mines in Colorado.

Contact Zinc Deposits

I refer under this heading to contacts with eruptives. In general, where mineralization occurs at or near contacts of sedimentaries with eruptives, minerals described under the preceding head *may* be found, but are not necessarily. As pointed out under "Contact Metamorphism," aside from occasional "baking" effects, neither metalliferous mineralization nor metamorphic minerals are the result of simple contact. Profound fissuring or faulting seems to be absolutely necessary to accomplish this.

Secondary Enrichment of Zinc Deposits

This rather overworked term seems to be well covered by "aggregation" or even by "segregation." Leaving it at its face value so far as zinc deposits are concerned, evidence gathered from a wide range of observation seems to point to subtraction rather than to addition, in many if not in all cases, in accounting for rich deposits of oxidized ores. The Bertha mines in Wythe County, Virginia, are excellent examples of the point in question. The surface of the worked deposit was originally covered by a thick mantle of clay. Mining operations disclosed pinnacles or chimneys of limestone as high as 100 ft. (30.48 m.). Some of these pinnacles were coated with oxidized ores and the "valleys" were occasionally floored with massive deposits from 5 to 40 ft. (1.52 to 12.19 m.) thick. In addition, there were layers of soft and hard "buckfat" carrying from 15 to 17 per cent. of zinc.

The chimneys, while showing more or less blende, were nowhere, en masse, of economic value. In no place was there blende sufficient to account for the probable average of 15 per cent. zinc in the valleys between the pinnacles. The conclusion would seem to be warranted that the rich mass of oxidized ore had been formed by aggregation from the blende-bearing limestones. Yet examination shows that the chimneys and floors are dense, compact dolomites with no evidence of channels, even capillary, through which the solutions might have traveled. This would leave the only source of enrichment to overlying blende-bearing dolomites. But on all sides of the old workings the surrounding dolomites are wholly barren of zinc. In short, there are, at present, no visible channels or sources from which or through which enriching solutions could come. Taking all data now observable into consideration, the conclusion seems to be inevitable that the rich oxidized Bertha deposits were formed *in situ* by the subtraction of the soluble dolomites in which the zinc orebody was originally located in the form of sulphides.

This conclusion seems to be fortified almost to complete demonstration at the Austinville mines about 8 miles (12.87 km.) to the southwest. These mines were originally worked for oxidized lead exclusively, later for both lead and zinc sulphides. The outcrop of these mines is about 235 ft. (71.63 m.) above the base level of drainage—the New River (see Fig. 1). Mining operations here disclosed, though in a lesser degree, the same chimney formation as at Bertha. The oxidized ore was of the same character as well. The clays in places were 90 ft. (27.43 m.) thick. Instead, however, of the ores being confined to chimney coatings and floorings, at least six well-defined blankets of oxidized ores were followed, not continuously, for at least 6,000 ft. (1,828.8 m.) on strike and were followed down on the dip to the level of the river. At this level the original sulphides appeared to the exclusion of the oxidized ores. In other

words, in repeated instances, oxidized ores were followed down to the original sulphides and these, in turn, have been followed down to the 435-ft. (132.58-m.) level, 200 ft. (60.96 m.) below the base level of drainage (see Fig. 1). The combined lead and zinc averages about 14 per cent., large masses running as high as 30 per cent. Here there seems to be no possible room for doubt but that the rich oxidized masses of ore, averaging upward of 40 per cent., were formed by subtraction of the dolomite, not by the extraneous addition of zinc and lead.

Another striking example is to be found at the New Prospect mine in Union County, Tennessee. This mine (see Fig. 2) can be represented by a prism about 900 by 50 by 90 ft. (274.32 by 15.24 by 27.43 m.). For 600 ft. (182.88 m.) the western end of the prism consisted wholly of oxidized ores. Six hundred feet to the east the face of the open cut is about 90 ft. high. The bottom 20 ft. of the mine is shattered dolomite running about 20 per cent. combined zinc and lead. For 30 ft. above this shattered zone the combined metals will average about 10 per cent. Here there seems to be no possible room for doubt that the rich oxidized ores were derived wholly from the decomposition of the mass in place, the result being that the soluble dolomite was, in the main, leached out, leaving the oxidized lead and zinc. Again, this is an example of subtraction, not addition. The same conclusion seems to be inevitable at the Straight Creek, Mascot, in Tennessee; the Hanover, Kelly and Cleveland in New Mexico; the San Xavier in Arizona; the Leadville and Gilman veins in Colorado; the Tiro General, La Bufa, Cruz del Aire and other mines in Mexico and the United States. In Guatemala and San Salvador are mines showing the same characteristics. In each of the mines cited, rich oxidized ores have been followed down to correspondingly rich sulphides. The Bertha and Delton mines in Virginia, the Embreeville, New Market and Mossy Creek mines in Tennessee, show no commercial zinc sulphides below the oxidized surface ores even with fairly extensive drilling.

Some mines in Wisconsin and in the Joplin field (in its most comprehensive meaning) seem to be examples where the subtraction has been carried to the point of exhausting the zinc and lead as well. In these instances, where masses of stained clay evidence the work of meteoric waters, secondary enrichment ought to have followed. On the contrary, however, these waters have removed not only the limestones but most of the zinc as well.

There is a peculiar mine near Santa Eulalia, Chihuahua, Mex. A shaft about 1,100 ft. (335.28 m.) deep was sunk through a rotted rock of undetermined origin; then came about 200 ft. (60.96 m.) of oxidized ores, then about 200 ft. of zinc and lead and other sulphides, and finally, oxidized ores again. In the zinc mines of Alotepec, Guatemala, there are great dike-like masses of garnet in which zinc and lead run as high as

16 per cent. The galena and blende often come to the surface. In places, however, the blende is decomposed and the cavities are filled more or less completely with zinc oxide. The mine outcrops for about $2\frac{1}{2}$ miles (4.02 km.) along the Rio de las Minas. The greater part of the ore zone has a hornblende gangue instead of garnet. The hornblende is badly decomposed but carries zinc oxides from 3 to 11 per cent. zinc. When undecomposed hornblende is found, zinc in the form of sulphide is found in the same quantity. In the San Juan de Calera mine in San Salvador, zinc and lead are 39 per cent., mainly as a sulphide. The gangue is garnet and occasionally, as at the Alotepec mines, cavities in the garnet are filled with zinc oxide.

In this same district is El Tejada mine. No blende was observed here but loose sheets of high-grade calamine were seen *in situ*. These sheets were observed in both jointing and bedding planes flanked with rotted, granular masses of white limestone. The rotted planes were from a few inches to 5 ft. (1.52 m.) thick. These were succeeded by blocks of fresh limestone. Observed faces resembled masonry walls, the calamine representing the mortar.

At Hanover, N. M., oxidized ores with rotted hornblende abut in places against solid garnet carrying upward of 5 per cent. zinc as blende. The oxidized ores also occur in jointing and bedding planes.

Without exception, where oxidized ores have been worked underground, the cavities are not entirely filled with clay and ore but it is impossible to calculate the relative proportions of original limestone and zinc. An estimate would lead one to conclude that, were the cavity re-filled with the original limestone with the corresponding zinc as blende, it would run about 15 per cent.

In the Bajan district in Mexico, cavities from which zinc carbonate was being mined were 1,500 to 3,000 ft. (457 to 609.6 m.) down a slope of 20° .

Depth at Which Zinc and Lead Deposits Have Been Found

In *Bulletin 606, U. S. Geological Survey*, p. 202 *et seq.*, Siebenthal says that at Stone City, Kan., zinc ore is reported in a well at a depth of 591 ft. (180.14 m.); at Pittsburg, Kan., at the same depth; in the Joplin district in wells at depths between 300 and 800 ft. (91.44 and 243.84 m.); near Rock Island, Ill., at more than 700 ft. (213.36 m.) and at Peoria at a depth of 699 ft. (213.05 m.). Siebenthal seems to think that artesian circulation might account for these ores, some of which would be economic if at a shallower depth.

While I can only be sceptical regarding it, changing topography might make such an origin possible, but the ores would then have originated many years ago. I call attention to the following localities where no

reconstruction of ancient topography could possibly account for their origin.

At Austinville, Va., the New River is the base level of drainage and this river flows on the eroded crest of an anticline. As seen in this section (see Fig. 1) the outcropping ribbon limestones on the river are more or less mineralized with zinc and lead sulphides. These rocks dip to the south at an angle of about 45° . The drill hole is located only a few feet above the river level and 1,000 ft. (304.8 m.) south at the foot of the bluffs. At a depth of 750 ft. (228.60 m.) pyrite was cut with traces of zinc and lead. The core shows 10 to 80 per cent. pyrite. At a depth of 1,000 ft. another band of pyrite was cut. North and south of the river, ribbon limestone and shales outcrop, dipping respectively north and south. As has already been stated, the block of nearly pure dolomite between these rocks can be accounted for in no other way than by a fault with a throw of something like 3,500 ft. (1,066.8 m.). If the ore was deposited subsequently to the faulting, and this seems certain, artesian circulation could hardly account for the mineral deposit. Even supposing that the folding and faulting occurred simultaneously with the great Appalachian folds, the crest of the restored anticline would have been at least 4,000 ft. (1,219.2 m.) above the present river level and in order to have deposited where they now are, their point of origin must have been far east, at least 50 miles (80.47 km.), and there can be no certainty that at this eastern locus the elevation was above the crest of the New River anticline.

The Mascot mine in Tennessee has the Holston for base level of drainage. The deposit of zinc here goes far below the level of the river. In this mine there can be absolutely no question but that the deposit was formed subsequently to the faulting and brecciation. Here it is quite impossible to reconstruct a topography that would allow artesian origin of the ore.

In Claiborn and Union Counties, Tennessee, the Powell and Clinch Rivers are the base level of drainage. The Powell, at least, flows on the eroded crest of an anticline. West of the river, the Knox formation dips rather steeply to the foot of the Cumberland Mountains; east, the dip is gentle as far as Tazewell where the Conasagua shales outcrop. At the Caldwell mine (see Fig. 3) blende was cut in the shales 1,100 ft. (335.2 m.) below the level of the river. Blende was also cut in adjacent holes at depths of 900 and 1,000 ft. (274.32 and 304.8 m.).

About 15 miles (24.14 km.) below Austinville, Va., is a partially developed property known as the Forney mine. The New River here flows between bluffs about 250 ft. (76.20 m.) high. Some zinc and lead has been mined in these bluffs in the bedding planes. These planes can be traced to the level of the river. In the river bottom, shafts have been sunk to bed rock to the river level. In these shafts zinc and lead occur

in what seem to be rich layers between the bedding planes. Along these planes the limestones seem to be more or less brecciated. These mineralized planes have been traced clear across the river. The rocks dip steeply. The river cuts across the strike of the rocks. A drill hole is said to have been put down in the river bed at a low stage of the water and is reported to have shown ore to the bottom.

At Friedensville, Pa., the zinc ore occurs in steeply dipping rocks that have the appearance of anticlines. The surface of the mine is only about 400 ft. (121.92 m.) above tide.

At Cape Rozier, on Deer Isle, Maine, zinc and lead sulphides are found over 100 ft. (30.48 m.) below sea level. It might be added here that the ore cut in the Caldwell drill holes was 100 ft. below tide.

To sum up: the entire zinc-lead area in eastern Tennessee, southwest Virginia and in Pennsylvania is so broken and faulted that it seemingly precludes the idea of artesian water. Even the reconstructed topography fails to show reasonable grounds for artesian water in the past. Moreover, leaving out of consideration the bearing of previous topography on artesian water, the fact remains that the greater number of known deposits in the localities above mentioned are, in effect and reality, "true fissure veins" with all that this term connotes.

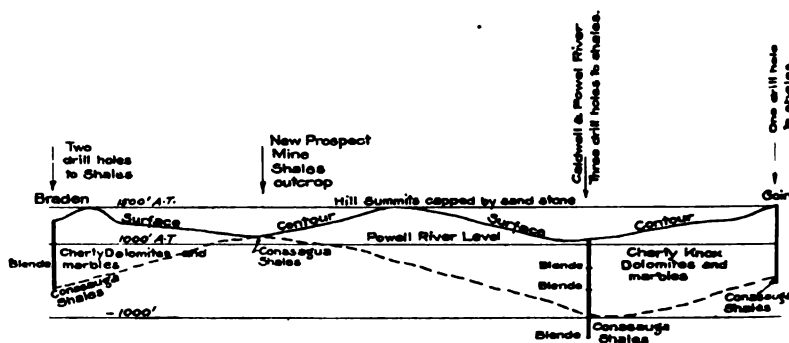
Conclusions

In the body of this report it has been pointed out that in zinc veins accompanied by eruptives the main characteristics are the presence of high silver values in the galena and, in many instances, in the blende, and the presence of comparatively rare minerals—vanadinite, pyromorphite, wulfenite, copper and silver sulphides, antimony and other minerals of this class—and of metamorphic minerals such as garnet, hornblende, epidote, wollastonite, etc. It was further pointed out that at Charcas, Hanover, the Alotepec and San Salvador mines, the simple presence of eruptive dikes produced none of these effects. In fact, in many instances, notably at Hanover (Fig. 6), a dike cuts through both the blende and the metamorphic garnet. In Fig. 7, also at Hanover, the blende occurs between the limestone and garnet with no dikes whatever.

Another coincidence, not pointed out, in the cases of Alotepec, San Juan de Calera, Charcas, Hanover, and notably at the copper mines in the San Carlos Mountains, Tamaulipas, Mex., the limestones occur mainly as great islands floating in a sea of basal porphyry. In these localities hornblende, garnet and epidote and other metamorphic minerals appear in great and persistent dike-like masses. It has forcibly occurred to me that such great masses of porphyry would, for ages, retain high temperatures in their interior and that earth movements, such as would result in dikes, would also open channels through which heated, mag-

matic waters would rise and manifest their presence by the intense metamorphism noted. It is to be noted that dikes do not always produce open channels and lack of metamorphism accompanying these dikes seems only to lend strength to the assumption that fissuring may or may not produce metamorphism, depending wholly on whether the fissures open channels to highly heated masses lying deep. Whether such heated masses exist or not, in a given locality, is, perhaps, not always demonstrable, but these are at least coincidences. In the localities named great masses of porphyry do exist and I do not now recall a locality that would present an exception. However, this suggestion is an inference and cannot, at present, take rank as a demonstrated conclusion. Whatever rank is assigned to it by others, I regard the facts as of academic rather than of economic importance.

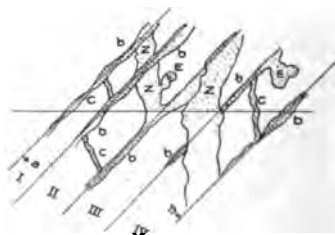
If, however, the other observations which I have recorded find acceptance, they have a most vital economic bearing. These are the universal accompaniment of fissures with zinc-ore deposits whether accompanied by eruptives or not. This fact puts even the zinc deposits of the Valley and the Appalachian States in the class of fissure veins. It follows from this that there is no reason why these fissures should not reach the productive depths of the deepest worked fissures in the United States, at Butte. Here economic fissures have been and are being worked at a depth of over 3,000 ft. (914.4 m.). It promises to put the great zinc fields of the Mississippi valley into great permanent camps instead of an ephemeral field of shallow diggings with a comparatively small individual volume of ore. The fissure character of all of our zinc deposits I regard as a demonstrated conclusion, not as a plausible inference.



Vertical Section from Braden to Eli Goin Prospects Union and Claiborne Counties Tenn. N 65° E. — Showing relation of Conasauga shales to surface as proved by drill holes.

NOTE:—The shales are the one horizon of New Prospect and Straight Creek Mines.

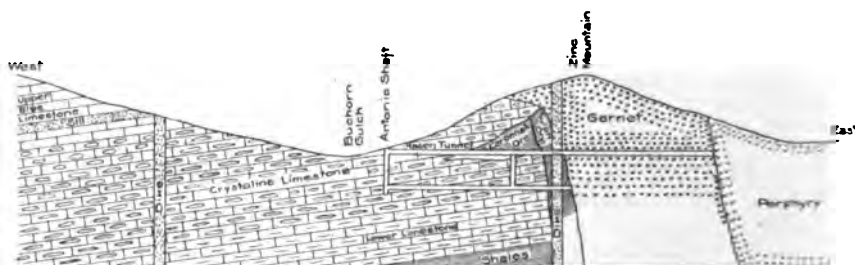
FIG. 3.



Detail of Mineralization on

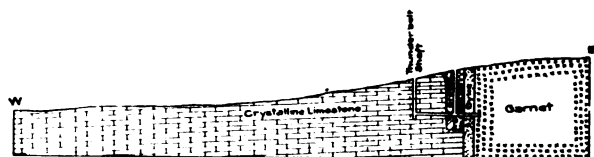
Bedding Planes, especially b-b-b
 In cross fractures, c-c-c
 In crushed zones, z-z-z
 In etched Cavities, e-e
 b-b-b 1" to 4" thick heavy PbS,
 some ZnS secondary gangue
 as above in c-c
 In z-z and e-e gangue as above
 main mineral ZnS; sometimes
 barren, at others, rich.

FIG. 4.



Section through
Nason Tunnel Workings.
The Empire Zinc Co. - Hanover N. M.
Not to scale.

FIG. 6.



Section
Thunderbolt Mine
The Empire Zinc Co. - Hanover, N. Mex.
No scale

FIG. 7.

N-E, S-W Section through Kennedy Mine Wisconsin.
Showing faults and folds in Glass Rock.

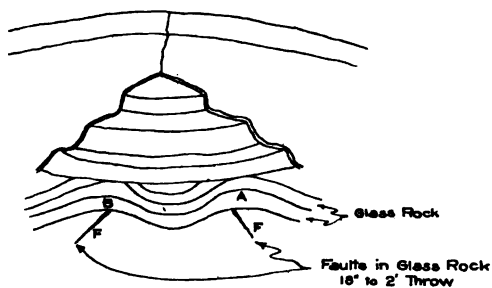


Fig 9.

North West - South East Section
On Floor of Kennedy Mine, near Hazel Green Wis.
Section 50 feet below surface.

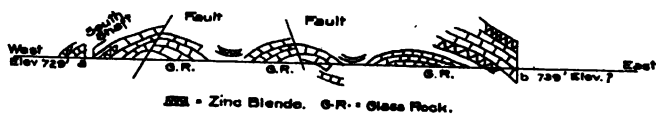


Fig. 10.

composition of the walls, whether similar to that of the veins or different, is of no importance, and that for these veins the fineness of the fibers is determined by the spacing of the pore spaces in the walls. There is no evidence of replacement or recrystallization along the walls, and where one wall contains angles or other irregularities, there are corresponding irregularities in the opposite wall, such that the two surfaces would fit closely together if placed in contact.

Mr. Graham is of the opinion that vein growth "has taken place at the expense of the walls," which "have receded because they were continually destroyed, and not because they were pushed apart by the growing fibers;" but he does not explain why serpentine should replace serpentine of the same composition and of "the same specific gravity." The lower figures given by textbooks for the specific gravity of chrysotile as compared with massive serpentine are possibly due to the difficulty of making accurate determinations in the case of such finely fibrous material.

Dr. Merrill's paper of 1905 in the *Bulletin of the Geological Society of America*² was not overlooked, since at my request he had kindly furnished me with a reprint. I regret that direct reference was not made to this paper, as was done with his other publications. This omission, as previously explained (pp. 1982 to 1983), was owing to the fact that the many theories advanced by geologists to explain the origin of cross-fiber veins had been reviewed recently in three monographs (see footnote 17, p. 1983). Dr. Merrill's theory is outlined in each of these publications and was included in my classification of theories.

Dr. Merrill is still of the opinion that the veins were formed in open crevices "due to shrinkage such as is incidental to the change of a highly hydrated, colloidal substance into a less hydrated and more solid form, and perhaps also to the loss of silica, as suggested by Prof. Kemp."³ No evidence in support of this opinion is given in Dr. Merrill's paper or in his discussion of my paper. He compares the hypothetical cracks in serpentine with "the shrinkage cracks which appear in clay on drying, or, better yet, those which result from the shrinkage of a gelatinous mass of iron carbonate, as in the so-called septarian nodules of clay-iron stone;"⁴ but he cites no evidence tending to prove that the latter were formed in the manner postulated. If the vein spaces in septaria result from evaporation of water and the shrinking of a gelatinous mass, the cracks would presumably begin at the surface and diminish in size as they approached the center. Now the reverse of this is true; the veins are largest near the center and seldom reach the surface (see Plate 34 accompanying Dr. Merrill's paper). This suggests that the growth of the veins may have

²G. P. MERRILL: On the Origin of Veins in Asbestiform Serpentine. *Bulletin of the Geological Society of America* (1905), 16, 131-136.

³*Idem.*, 135-136.

⁴*Idem.*, 136.

accompanied the growth of the concretions as a whole. The view that such veins were not deposited in open cavities is supported by the fact that they sometimes contain detached angular fragments of the wall material.

Concerning the origin of the serpentine, I purposely made no assumption as to whether the source of the water was magmatic or meteoric, for this was not essential to the development of my theory. The equation (p. 1986) was introduced merely as an illustration of the fact that the formation of serpentine might be accompanied by a net decrease in volume if the volume of the water is taken into consideration. If serpentinization is a deep-seated process, as advocated by Dr. Merrill, it is all the more difficult to explain how his cavities could remain open.

Heddle seems to have first suggested the possibility that the fibrous structure in chrysotile might be a result of its growth outward from the interstices between mineral grains (p. 1977).

The increase in the volume of a rock mass when chrysotile veins are formed is due, according to my theory, to the introduction of water and formation of serpentine rather than to the growth of the veins; and the resulting compressive effects have been generally recognized by geologists.

It is not necessary to reply here to the other points raised by Dr. Merrill, since those who are interested may draw their own conclusions after reading the original papers.

The theories advocated in my paper were supported by much evidence, both observational and experimental; they furnish the only plausible interpretation of the established facts; and, therefore, I believe that my conclusions must stand until new facts are adduced tending to disprove them.

The nature of the force that enables growing veins to make room for themselves by pushing apart their inclosing walls was not discussed in detail as I had treated that phase of the subject in a previous publication. Since that time, however, I have obtained additional experimental data, and my present views on the subject are given in a paper on "Pressure Phenomena Accompanying the Growth of Crystals," which will appear in an early number of the *Proceedings of the National Academy of Sciences*.

Notes on Flotation—1916

Discussion of the paper of J. M. CALLOW, presented at the New York Meeting February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 245 to 275

DAVID COLE, El Paso, Tex. (communication to the Secretary*).—I notice that Mr. Callow takes issue with me on certain points I have previously contributed to the *Transactions*, to which I would reply as follows:

Regarding Inspiration: Mr. Callow's refinements of gravity methods made just prior to the advent of flotation in the Southwest were intended to be "down to the minute," and the gravity flow sheet for Inspiration ores which he brought out at that time as the result of his long campaign of experimentation was regarded by many as the last word in wet gravity work. It was not my intention to discredit this good work when I said "he threw small light on slime treatment." "Minus 200 mesh," however, is not synonymous with "real slime." Repeated treatments by gravity methods will result in the separation of a very large percentage of the fine granular sulphides in sizes much finer than 200 mesh, and I gave him due credit for that. The unavoidable losses were made in the "real" slime which I mentioned in the paragraph which Mr. Callow refers to as my "statements on page 1871," and his refinements were not a solution to the real slime problem.

Mr. Callow has called attention to the fact that his best results at Inspiration in gravity work were obtained on *Joe Bush ore* and he presents a table, viz., Tests Nos. 68, 71, and 72, and asks us to consider and compare these results, made by his refined gravity methods, with the results obtained by flotation methods in the new mill "during the second half of 1915 and in August, 1916." Of course, the ore treated in the mill was what happened to come. *It was not Joe Bush ore*, and therefore comparisons of detail figures are of no value.

In regard to Mr. Callow's statements under the caption "Callow Cells Adopted by Arizona Copper Company," I would say that those who have been sufficiently interested to read my paper and his remarks will notice what amounts to contradictory statements on several points which I cannot allow to go unchallenged. I was on the ground constantly and got my information at first hand. The tables published were the official ones. Mr. Callow was not present during the period and has been laboring under that disadvantage.

As stated in my paper, page 1622, the "feeds were not identical," and I would call attention to the fact that the contributing flow sheet depends upon methods of preparing the feed. My request to have the feeds made identical was not heeded. The C-B cell would do anything the Callow

* Received Mar. 29, 1917.

cell would do, plus the one of not being embarrassed by coarse sands. The C-B cell was designed at Morenci especially to fit the conditions in the plant where there is inadequate room for the usual apparatus for preparing the feed. It was made with a view to treating all of the tonnage comfortably in the space available, and at a cost for construction amounting to not more than one-third the amount needed with the use of the Callow cell. Therefore, the question of previous preparation was important, and I proposed that both units be operated simultaneously on exactly the same feed, first on the feed as prepared for Callow treatment, and then on the feed as prepared for C-B treatment. Of course, the C-B cell would handle the Callow feed, but the Callow cell would fail on the proposed C-B feed, and that is the reason the feeds or contributing flow sheets were not made identical.

In my paper, on page 1617, I said: "the blower used with the C-B apparatus was an old one designed for not more than $3\frac{1}{2}$ -lb. pressure; with 6-lb. pressure the shafts were deflected and the machine had to be water-jacketed to keep down the heat (impellers rubbed the sides); it was much larger than necessary and a great excess of air was blown off from open valves. Therefore, no record of the amount of air used or power required could be even approximately obtained." Mr. Callow's statement that the C-B apparatus required 5.9 hp. per ton, cannot, therefore, have a foundation in fact.

The labor used in the testing was identically the same for the C-B as for the Callow apparatus. Much more labor was used on each unit than was necessary.

The statement that the Callow apparatus would recover from 1 to 1.2 lb. of copper per ton more than the C-B recovery would have been called to my attention in detail at the time, if it had been a fact capable of demonstration, and the C-B unit under discussion would have doubtless gone to the scrap heap, which is not the case.

Power, efficiencies, recoveries, and labor requirements were so nearly parallel in the two systems that they were not factors in the Arizona Copper Co.'s decision to use the Callow apparatus in its No. 6 concentrator.

It will interest those who follow this discussion to know that the Arizona Copper Co. operates two separate concentrating plants for sulphide ores. One of these mills is at Morenci. The other one is of later design and construction and is at Clifton. The ores for both are derived from the Morenci mines and are substantially the same in character.

The Clifton mill has a capacity of 500 tons per day and employs the following flow sheet:

In this flow sheet it will be noted that the original C-B machine (moved from the No. 6 mill) occupies a prominent position, handling

the whole tonnage on two of its three cells, making a clean concentrate in very large percentage of the whole, which goes to the filters for dewatering. The Callow cells are used in a second treatment of the pulp,

FLOW SHEET OF NO.4 MILL (500 TON) OF ARIZONA COPPER CO.
CLIFTON, ARIZONA

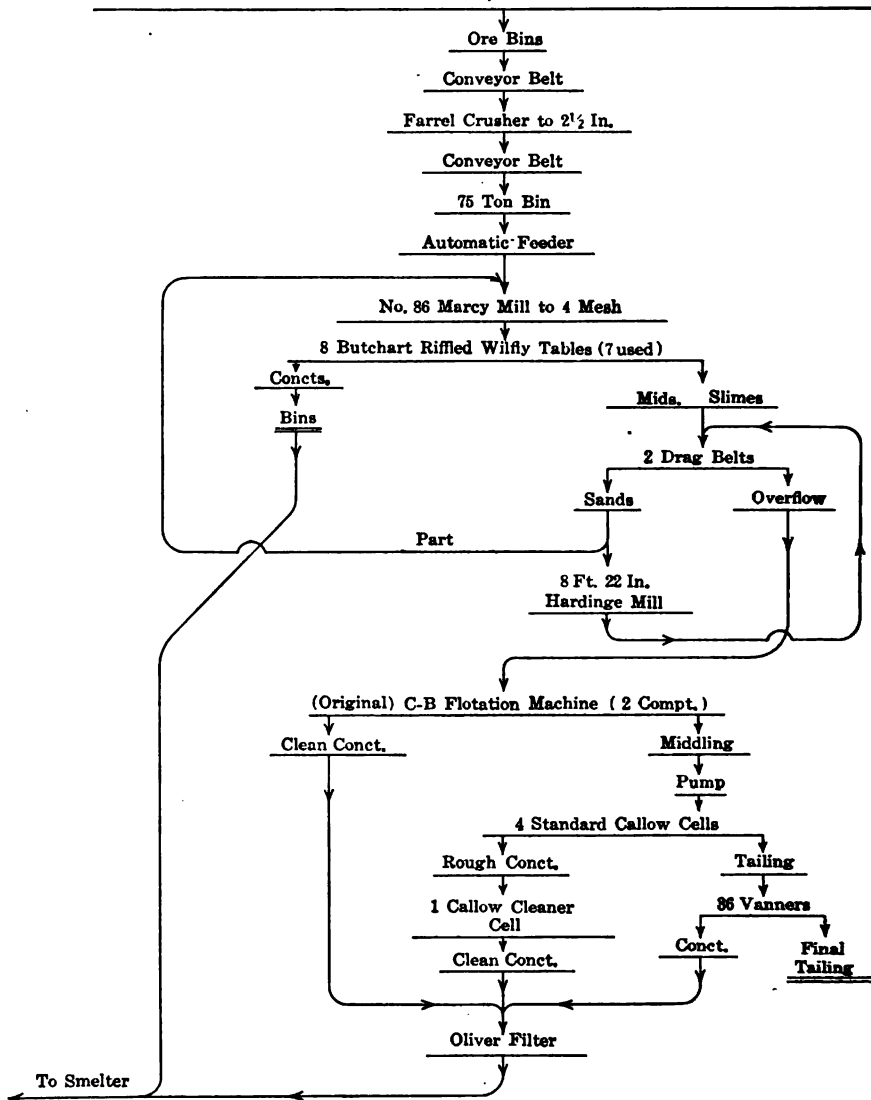


FIG. 1.

and the reject from these cells is finally passed over vanners for final treatment before going to waste. This makes a superior flow sheet for "Morenci" ores and the results obtained compare favorably with the

results obtained in the larger mill at Morenci in which Mr. Callow's cells were "adopted." Whether C-B treatment of the reject of the first C-B treatment would be inferior to the Callow treatment practised is not known, but we have every reason to believe that the results would be parallel.

As throwing additional light upon the power requirements of the rival systems of practising pneumatic flotation, I would say that at Cananea one 3,000-cu. ft. Roots blower, using 98 hp. input to motor, furnishes air for three triplex C-B machines and three duplex cleaner units. The total area of aerating cells is 307 sq. ft. The tonnage handled is 1,200 tons per 24 hr., and one Mexican operator and a helper per shift is found to be sufficient labor to handle the equipment. If the labor received \$4 gold per 8-hr. shift, the cost per ton for this item would be 2 c. The power used in blower, it will be noted, is 2 hp.-hr. per ton treated—this with 5 to 6 lb. pressure of air, including also the centrifugal pumping required in returning rejections from the cleaner.

Potash as a Byproduct from the Blast Furnace

Discussion of the paper of R. J. WYSOR, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 121, January, 1917, pp. 1 to 32.

CHARLES CATLETT, Staunton, Va. (communication to the Secretary*). —I hesitate to discuss in general terms a paper such as Mr. Wysor's, which bears evidence of so much careful and detailed investigation, yet it is the broad possibilities which his paper presents that are really attractive and his cautious conservatism may make us unappreciative of these possibilities.

Mr. Wysor calls attention to the fact that the iron-ore mixture which he uses, carrying 0.28 per cent. of potash, is probably above the average. No doubt this is true, but on the other hand it may be very much less than the mixture at some blast furnaces or very much less than would be the case with special mixtures bearing the question of potash in mind. Unfortunately, iron ores are not ordinarily examined for potash, and with the ordinary analyst there is opportunity for very considerable error in the determination. The 10th Census gives 270 complete analyses of iron ore, on the accuracy of which we may rely. Of these, there are 21, or 8 per cent., that carry more potash than Mr. Wysor's mixture; 14 of these carry approximately twice as much, or more; eight carry approximately three times as much, or more; three carry approximately five times as much; one carries over seven times as much. It may be worth while to take some of the ores that are known to be high in potash and consider the possibilities in the light of Mr. Wysor's investigations.

* Received Mar. 12, 1917.

For instance, there are certain red hematites in eastern Alabama (locally known as "gray ores") which are very high in potash. They are in large quantity, and are low in phosphorus for Southern ores; but have so far not met with general favor because rather siliceous. While shipments have been made of 50 per cent. metallic iron, they would ordinarily run nearer 45 per cent. with the correspondingly high siliceous contents. The average of the analyses of a number of samples, by well-known chemists, gives a K_2O contents of 2.08 per cent., or over seven times the potash contained in Mr. Wysor's mixture. The exact iron contents of his mixture is not disclosed; but if we assume it to be 52 per cent. it would mean that for each unit of iron there are 0.0044 units of potash. For the Alabama gray ores with a basis of 45 per cent. metallic iron there would be for each unit of iron 0.046 units of potash, or nearly nine times as much in proportion.

I am unable to find any typical analyses of potash in Southern cokes. As they commonly carry more ash than the coke used at South Bethlehem, it is quite possible that the potash is higher. Probably 75 per cent. more coke would be used in Alabama per ton of iron. The percentage of potash in the flux used at South Bethlehem is not uncommon. More flux would also be used in Alabama; probably twice as much. This would also serve to increase the potash. But it is very conservative to assume 50 per cent. more coke and 75 per cent. more flux, and if we assume the potash contents in the slag to be the same as at South Bethlehem and the same proportionate loss in the slag, there would be derived, from the coke and flux 11.10 lb. per ton of iron, above what would go out in the slag. Assuming a similar loss from leakage, we would get the following as representing the potash in the gases that would go to the treaters.

	Pounds
Charged into furnace per ton of iron	
From ore.....	87.7
From coke and stone.....	18.97
	106.67
Loss from	
Fumes, etc.....	0.79
Dust-catcher.....	2.82
Top furnace.....	4.76
Slag.....	7.87
	16.24
	90.43

How much can be saved?

I am advised that at the cement plant at Riverside, Cal., where the gases from the kilns are carried through a Cottrell precipitator, as much as 95 per cent. is commonly saved. At the Security Cement & Lime Co.'s plant at Hagerstown, Md., with which I am associated, the waste

gases from the kilns are passed through a Cottrell precipitator. The balance shows that we commonly save 90 per cent. of the material that actually goes into the treater. As the volume and temperature of the blast-furnace gases are much less, it ought to be possible to do better than at a cement plant; but certainly 85 per cent. recovery is conservative. This would mean a saving of 76.86 lb. of K_2O per ton of iron produced.

It may be silly to note that at the present common and current prices for potash this would be worth \$11 or \$12 per ton of iron produced; but we cannot ignore the fact that with the destruction of shipping, and the high freight rates, potash must remain high for a number of years. Even at pre-war prices a furnace working on the gray ores of eastern Alabama, or a similar ore, with a byproduct recovery plant, would have a credit of from \$2 to \$3 on the cost of the iron from the potash saved. This would be in addition to the collateral advantage derived from cleaning the gases.

In addition to this, there are a number of furnaces which also carry zinc and other valuable materials in the escaping gases.

It is difficult to escape the conclusion that in future blast-furnace development consideration will be given to the percentage of potash in the raw material.

The Need and Advantages of a National Bureau of Well-Log Statistics

Discussion of the paper of W. G. MATTESON, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 287 to 290.

ARTHUR KNAPP, Ardmore, Pa. (communication to the Secretary*).
—The author has failed to consider the point which in my mind is the most convincing in advocating some sort of a bureau for the collection of well logs. I venture to say that 90 per cent. of the logs as taken today are absolutely worthless to any one but the one who has taken the log. This is largely due to the careless use of nomenclature. The author says on page 289, "He must ascertain which of the many factors might account for the dry hole in question and often only careful examination of well logs will convey this information." I believe that even a most careful examination of the average log reveals nothing.

I have at hand a book on petroleum, by a well-known author. This book has a number of logs from fields all over the world, from which I shall illustrate. In a well log from Kansas I find "Soft gray shale" and in one from Humble I find "Hard blue shale." As a matter of fact the shale from Humble is softer than the shale from the Kansas well. The difference is that the drillers in Kansas, being used to limestone and

* Received Mar. 12, 1917.

sandstone, call the shale soft because it is soft in comparison to most of the formations that they drill. On the other hand, the shale encountered in the Humble well was about the hardest formation encountered, and was called hard.

Again, I find such things as "mixed rock and sand." I am familiar with the formation in the field from which this log was taken, but I am at a loss to explain what is meant. In one log I find "shell formation" and in the next "4-ft. shell." I know that in the first case the reference is to a formation having shells in it and that in the second case a "shell" is a hard layer. However, one not acquainted with both fields might be misled.

What does "quicksand" mean when found at a depth of 870 ft.? How coarse is a "coarse gravel?" What is the difference between "sand rock" and "sandstone?" What is "pulverized shale?" What is "shale" anyway? I find one log that has in it "laminated shale."

One of the greatest benefits to be derived from the establishment of any bureau would be the necessary standardization of nomenclature.

Also, standardization should be extended to the method of taking samples and determining their properties. The color of a sample is rarely the same after it has been dried as when it was fresh from the well. Even rewetting does not always produce the same color. Many samples disintegrate after a short exposure to the air. Either the laboratory or the field determination must be taken every time. In general, it is impossible to compare a sample from the laboratory with one fresh from the well.

Calculations with Reference to the Use of Carbon in Modern American Blast Furnaces

Discussion of the paper of H. P. HOWLAND, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 111, March, 1916, pp. 627-650.

*Abstract of Discussion by Professor Mathesius, Prepared by
Joseph W. Richards*

Prof. Mathesius analyzes the running of Howland's furnace No. 19, according to the method described by him in *Stahl und Eisen*, 1916, Nos. 30 and 31, and in his work "Die physikalischen und chemischen Grundlagen des Eisenhüttenwesens" (Spamer, Leipzig).

By assuming 500 Cal. lost from the hearth by radiation and cooling water per kilogram of pig iron made, a throat-gas temperature of 600° C., and taking the ratio of CO²/CO as 0.64, he finds that his diagram would show 0.68 kg. of carbon oxidized per kilogram of pig iron, while Howland gives 0.66 kg. He expresses himself as well satisfied with this test of the accuracy of his theory. Making the further assumption that the heat losses from the shaft are 1,000 Cal. per kilogram of pig iron (twice as

great as from the hearth), he calculates that 9.2 per cent. of the iron oxide charged is reduced in the hearth, and 90.8 per cent. in the shaft. Of the latter 15.8 per cent. is also "direct reduction" leaving 75 per cent. indirect reduction, in the shaft, by CO gas, and a total of 25 per cent. direct reduction.

He compares this result with ordinary blast-furnace practice, saying that in foundry iron furnaces three-fourths of the direct reduction ordinarily takes place in the *shaft* and one-fourth in the *hearth*, while in Thomas Basic pig-iron furnaces about 70 per cent. of the direct reduction takes place in the *hearth* and 30 per cent. in the *shaft*. He ascribes this difference to the shaft being much warmer in the foundry iron furnaces.

By rather involved and strained reasoning, Prof. Mathesius then attempts to show that if the blast temperature were raised to 800° C., the furnace could get no advantage from this, using the same ore; the only advantage would be that a more difficultly reducible ore could be used, and worked as efficiently as the more easily reducible ore.

WALTHER MATHESIUS, Charlottenburg, Germany (communication to the Secretary*).—Howland bases his conclusions on the data of Grüner¹ who in his blast-furnace calculations makes the rather idealistic statement that the best results are obtained in a furnace in which the iron ore is reduced solely by the rising gases in the shaft.

Howland's paper contains a table compiled from carefully collected data in which a review is given of the results and conditions of 26 American blast furnaces. These furnaces use mostly Mesaba ore and the results are in most cases very good.

His calculations refer especially to furnace No. 19 in the table, a furnace of the Wisconsin Steel Co. which excels the others in efficiency.

In his investigation he draws the following conclusions: There is no law which governs the relation between coke consumption and the percentage of carbon burned at the tuyères. None of the furnaces in the table work according to the so-called ideal method as stated by Grüner. As far as the author is informed, there is no record of a blast furnace anywhere in which 100 per cent. of the gasified carbon was gasified at the tuyères. Moreover, the author does not consider such a condition desirable.

He reaches these conclusions by comparing the actual working conditions of the furnace with three theoretical working assumptions which he develops in detail in his paper. The furnace, however, does not work according to either theory, as the author tells us, and yet the results are throughout better than they possibly could be according to his theories.

It seemed interesting to me to calculate the working conditions of

* Received Dec. 19, 1916.

¹ M. L. Grüner: *Analytische Studien über den Hochofen* (1875), Wiesbaden, C. W. Kreidels Verlag.

development and construction of such a diagram is explained in the above cited references to my papers on this subject. For those of my readers who are not familiar with them I give the following short explanation:

The diagram assumes the blast at 600° C. and smelting pig iron of the following composition (the same as furnace 1 in Howland's table):

Si,	Mn,	P,	C,	S,	Fe,
Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
1.560	0.750	0.075	4.000	0.035	93.580

A heavy line in the middle of the diagram is taken as abscissa, on the left side of which have been recorded the working conditions that would result if 100 per cent. of the Fe_2O_3 were reduced directly in the hearth. On the right side of the abscissa are recorded the working conditions when the total direct reduction takes place in the shaft, in such a manner that the CO_2 , formed by the reduction of Fe_2O_3 by CO , is again completely reduced by the carbon of the coke to CO .

All possible intermediate steps between these two extremes mentioned are recorded on some intermediate points of the abscissa.

The ordinates of the diagram represent the corresponding consumption of carbon from coke (exclusive of ash and moisture) which is necessary for the working of the furnace at the indicated points of the abscissa line.

The consumption of carbon from the coke is divided into two great parts, one of which makes up for heat losses of the furnace due to water cooling and radiation. The corresponding ordinates are carried downward from the zero line in the center and according to the heat balance of the furnace under discussion are calculated for a heat loss of 500 Cal. per kilogram pig iron in such a manner as to determine the quantity of carbon from the coke which has to be burned with blast at 600° C. in order to furnish the required calories.

Since this condition holds good in all possible proportions of direct reduction between hearth and shaft, the consumption of carbon from the coke must be represented in a line passing the diagram parallel to the axis of the abscissa and at a proportionate distance from it. That line has been marked W. V. (which means heat loss) = 500 Cal.

The second large part of the consumption of carbon from the coke arises from the necessity for covering the heat requirements of the chemical and physical reactions taking place in the furnace. The ordinates corresponding to this heat consumption are carried upward from the abscissa. The upper ends of these ordinates start from a straight line and pass through the diagram from right to left. They are marked 0, 10, 20, 30, 40 and 50 per cent. direct reduction, respectively.

To better characterize the working, some other lines have been plotted on the diagram which were computed according to my theory of blast-furnace practice as mentioned above.

From Howland's article it appears that the furnace gases had a volumetric proportion of $\text{CO}_2/\text{CO} = 0.64$. A dotted line in the diagram shows how this proportion must form itself in the furnace according to the working conditions and especially according to the distribution of the direct reduction between hearth and shaft, taking into account a heat loss of 500 Cal. per kilogram of pig iron. A comparison of this dotted line with the lines indicating the percentage of the direct reduction brings out that in this particular working the total direct reduction is within 20 to 30 per cent.

The working data that we have been using do not, however, enable us to determine how the direct reduction is divided between hearth and shaft. The only data for that purpose concern the temperature of the furnace gases.

Howland does not give any such temperatures, and I had, therefore, to get such data from other American blast-furnace reports. I took these from a paper by Brassert³ which was abstracted in *Stahl und Eisen*. He gives 160° C. as the average temperature of the furnace gases.

The diagram of my theory of blast-furnace practice makes it possible to determine on the basis of this gas temperature how the direct reduction in the blast furnace is distributed between the shaft and the hearth. This is accomplished by putting into the diagram all those gas temperatures that must result at a working yield of 47.2 per cent., a blast temperature of 600° C., and a heat loss of 500 Cal. per kilogram pig iron. The plotting of these lines showing the temperature of the furnace gases is facilitated by a guide diagram at the bottom of the large diagram, the construction of which has been developed in my previously mentioned publications.

Since the lines of the furnace gas temperatures cut the line of the $m - (\text{CO}_2/\text{CO})$ proportion at certain points, it is easy to determine a point corresponding to a temperature of 160° C. by means of interpolation between the points of intersection giving the temperatures of 100° C. and 200° C. respectively. The point so found corresponds exactly with the actual working condition of the furnace under consideration. Accordingly, we can see on the diagram that the total direct reduction is 25 per cent., 36.5 of which takes place in the hearth and 63.5 in the shaft; in other words, 25 per cent. is the total direct reduction, 9.2 per cent. takes place in the hearth and 15.8 per cent. in the shaft. This conclusion gives us a clear conception of a furnace using Mesabab ore, showing that with the prevailing temperatures and composition of gases, and the working speed of this furnace of the Wisconsin Steel Co., only 9.2 per cent. of the Fe_2O_3 charged at the furnace entered the hearth unreduced. It must be remembered, however, that the theoretical calcula-

³ Brassert: Neuzeitliche Entwiebelung des amerikanischen Hochofenbetriebes, *Stahl und Eisen* (1916), 61.

tion could determine only the percentage of Fe_2O_3 charged that entered the hearth unreduced. There is at least a strong probability that those parts of the ore are no longer in the unchanged form of Fe_2O_3 , but have already undergone a gradual decomposition to Fe_3O_4 or FeO . The statement that 9.2 per cent. of the burden enters the hearth unreduced consequently implies that the quantities of oxygen to be directly reduced in the hearth by carbon and carbon monoxide are as great as if 9.2 per cent. of the ferric oxide entered the hearth unchanged. It is not to be doubted that more heat units are necessary for the direct reduction where the oxygen to be combined with the carbon is present in the form of ferrous oxide instead of ferric oxide. But these differences of energy can be neglected, as they are not large enough to cause an appreciable fault in the calculations.

It can be concluded that this percentage would not change appreciably if the percentage of the total direct reduction of the ore was changed, provided, however, the quality of the ore, the blast temperature and the working speed remained the same. This offers the possibility of plotting a new line in the diagram which up to now I had not considered in my theoretical investigations of blast-furnace practice. The new line would indicate under which condition the percentage of direct reduction in the hearth would remain 9.2 per cent. even if the total direct reduction should change within wide limits. This line is marked on the diagram: "Const. 9.2 per cent. dir. reduction in hearth," and is the locus of all points with direct reduction in the hearth of 9.2 per cent. It is to be considered as a standard line for the degree of reducibility of the smelted ore under the given rate of production or working speed.

From the position of the working point of the Wisconsin Steel Co. furnace in the diagram, it becomes possible to read without further computation that the actual consumption of carbon in the operation of this furnace must be 0.68 kg. of carbon per kilogram of pig iron in accordance with the conclusions arrived at in my blast-furnace theory.

Now according to Howland, the consumption of carbon per kilogram of pig iron is really 0.6615 kg.

The concurrence of these two figures can be considered unexpectedly satisfactory, especially so in consideration of the fact that in the framing of my theory several necessary assumptions could not be avoided (as, for instance, the apportioning of radiation losses between hearth and shaft at $\frac{2}{3}$ to $\frac{1}{3}$, etc.), the admissibility of which as before conceded would have to be established by considerable number of carefully computed corroborative furnace operations.

Under detailed investigation of the stated assumption in the single case under consideration, the precise establishment of a hearth balance as given in the appendix shows the presence of a heat loss of 322 Cal.;

completely elucidated by the above data.

In spite of these conditions, the question is still permissible, whether, and in what measure, a saving of coke could yet be effected by an eventual change of procedure.

Since the percentage of direct reduction in the hearth is as low as 9.2 per cent. of charged Fe_2O_3 , much further curtailment could not be expected to bring practical results. The direct reduction could in no case be appreciably lowered and no noteworthy decrease of coke consumption could result from such diminution.

On the contrary, it is worth mentioning that in this particular operation 15.8 per cent. of the direct reduction takes place in the shaft.

The further discussion, therefore, is directed to the task of determining whether, and by what means, in further diminishing this portion of the direct reduction, a saving of coke could be effected.

From the diagrams in my publications on the operation of furnaces for foundry and Thomas iron conducted on equal output and blast temperatures, it becomes evident that in the production of foundry iron generally the apportioning of the direct reduction between hearth and shaft is as follows: About 75 per cent. of it belongs to the shaft and 25 per cent. to the hearth, while in the majority of Thomas iron furnaces the direct reduction taking place in the hearth is 70 per cent. to only about 30 per cent. in the shaft. This noticeable difference is exclusively due to the fact that in the production of foundry iron the temperatures in the hearth must be maintained at a higher degree to obtain a considerable reduction of silicic acid, while in the Thomas iron process, on the contrary, it is necessary to keep the temperature of the hearth low enough to avoid an undesirable reduction of silicic acid. The consequence is that at the present rate of production for foundry iron the shaft is considerably better than for producing Thomas iron. From this results necessarily a considerably increased transformation into CO of the CO_2 formed in the lower portion of the shaft by the ore reduction, therefore greater direct reduction in the shaft. When the diagram established for gray iron and Thomas iron on otherwise identical conditions of operation are contrasted, the corresponding shifting of the lines of gas temperatures in the upper portion bring out these conditions saliently.

Similar shiftings of the lines of gas temperature are seen in the diagrams established for identical operations of gray iron and Thomas iron when the two diagrams are contrasted for the differences in blast temperatures, which shows unmistakably that under otherwise equal conditions a higher blast temperature produces a lower gas temperature and this result corresponds with practical experience.

The lower gas temperature indicates generally a correspondingly lower temperature of the shaft. To diminish the direct reduction in the

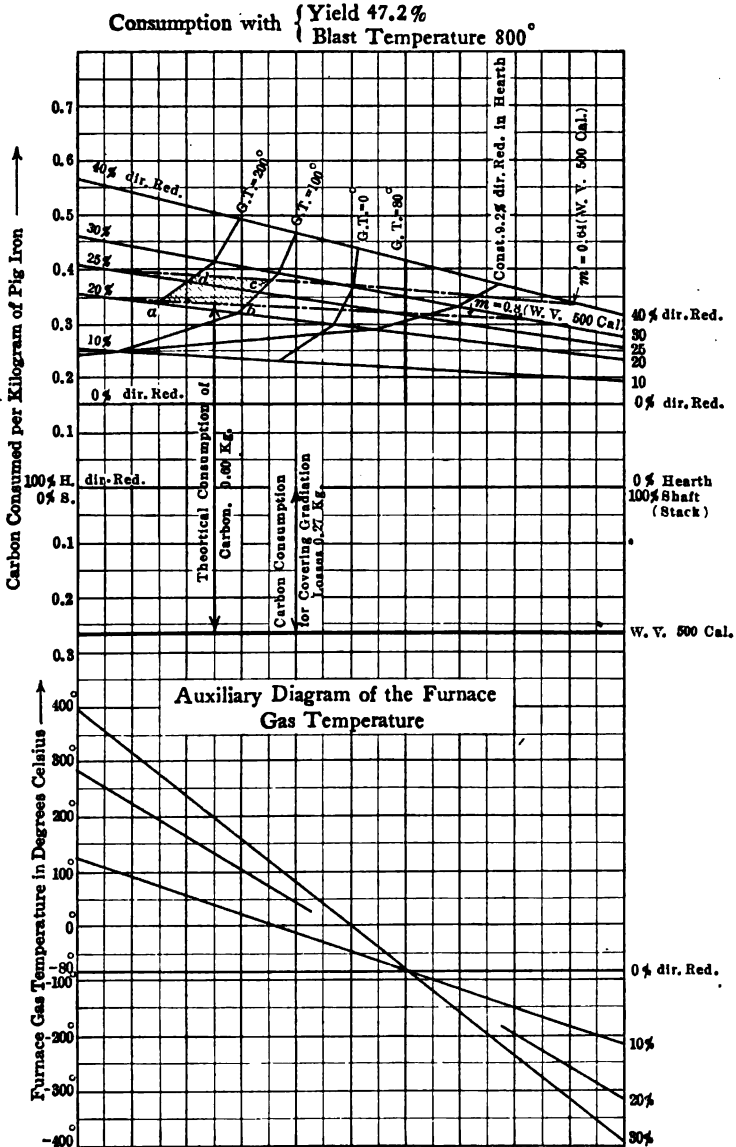


FIG. 2.

shaft of a blast furnace, the most feasible means advisable under otherwise identical conditions is to increase the blast temperature.

In the case of the Wisconsin Steel Co. furnace, the blast temperature

could be carried considerably beyond the present 594° , since even for Thomas iron temperatures up to 800° practically prevail in Germany. A diagram establishing working conditions for this furnace with blast temperatures at 800° would obviously be interesting and is furnished in diagram No. 2.

Detailed computation of data relating to it is not contained in the appendix since it is merely a guide to the calculations and an exhibition of results; it is otherwise based on the constant operating conditions used for diagram 1, and at the same rate of production. Consequently the heat loss here too is 500 Cal. per kilogram pig iron.

The salient fact in this diagram for 800° is that the coke carbon consumption to cover the heat losses, which in the blast furnace operated with 600° blast temperature is 34 kg. per kilogram of pig iron, drops to 27 kg. The considerable saving of coke hereby indicated is obtained simply because more heat units per kilogram of pig iron are carried into the hearth by the higher blast temperature, so that to cover the heat loss, less carbon combustion is required.

But what is the consumption of coke carbon in the upper portion of diagram 2? Two m lines have been introduced in this diagram corresponding to m -proportion of 0.64 and 0.80. The reason for this procedure is based on a note in Howland's publication according to which the 0.80 proportion has actually been observed on a specially propitious day of operation.

If an m -proportion of 0.80 was actually obtainable operating with a blast temperature of 600° , it is reasonable to assume that the same m -proportion can be obtained with higher blast temperature, for a decrease in the coke carbon required should result since the same quantities of oxygen from the ore are transferred to lesser quantities of gas. Whether in reality these more favorable m -proportions are obtainable, remains obviously to be proven by actual observations in practice.

The lines of gas temperature obtained exactly as in diagram 1 were also entered in the upper portion of diagram 2. Practically normal operating conditions can naturally only exist with a minimum gas temperature of about 100° ; and again only practical operation can show whether and in what measure the temperature of the furnace gas exceeds the 100° limit. There is no reason why the temperature should be higher than with a 600° blast; at least we can assume that it cannot exceed 150° .

All possible cases of furnace operations are embraced in diagram 2, in a quadrangle marked $abcd$ which is formed by the intersections of the m lines and the lines of gas temperatures of 100 and 200° .

Presupposing the same rate of production, it is safe to assume that the direct reduction in the hearth, with the same ore, will not be materially altered from that exhibited in diagram 1. In the upper part of diagram

2, however, the line indicating the direct reduction in the hearth (9.2 per cent.) does not intersect the quadrangle *abcd*. Evidently, then, a furnace operated under conditions represented by diagram 2 must obtain a heat surplus in the hearth, supposedly resulting in considerably increased reduction of silicic acid much beyond the aims of operation; in other words, a furnace operated in this manner would produce pig iron containing an undesirable excess of silicates.

In the experience of practical furnace operations, it has been found that the temperature in the hearth is correspondingly lowered by decreasing the quantity of coke. Obviously, then, the temperature could be brought to the point where the desired admixture of silicates in the pig iron results by simply decreasing the coke quantity in the burden.

From the diagram, however, it is further evident that operating along these lines would be impracticable because it would result in gas temperatures below 100° and because an *m*-proportion would be obtained of a magnitude, at least at present, considered unattainable; or that the experiment to stimulate furnace production of pig iron with a normal percentage of silicates by the use of blast temperature of 800° is doomed to disappointment.

This recognition most probably explains why a blast temperature of 800° has practically not been introduced in the furnace under consideration.

The diagram points to the fact that such operation could be carried into effect if, with the introduction into the burden of less easily reducible ingredients, the quality of the charge were altered until the line in the diagram representing the direct reduction in the hearth cut the quadrangle *abcd*.

We learn from this discussion of operating conditions that using somewhat easily reducible ores, such as the Mesaba ores, with blast temperatures of 800° , under ordinary circumstances pig iron with relatively low percentage of silicon is not obtainable unless less easily reducible admixtures are added to the charge in such proportion that the line of constant reduction in the hearth intersects the quadrangle *abcd* in the diagram. It is immaterial whether this addition consists of magnetite or corresponding percentages of slow reducing agglomerates.

Thus we arrive at the interesting observation, that furnaces working with easily reduced ores under conditions as represented can employ higher blast temperatures effectively only if the charge is correspondingly altered to become less easily reducible. This result is directly contrary to those in German operations, where a saving of coke can generally be gained by decreasing the mass of slow-reducing constituents in the burden, as the percentage of direct reduction in the hearth in most of these cases is above the requisite. And so the question arises whether saving of coke is actually possible in the manner here described. Diagram 2

furnishes the answer, showing that the consumption of coke carbon can reach the amount of 0.60 kg. only if in the new operation an m -proportion of 0.80 per cent. and a gas temperature of 150° is present. This would result in a saving of 0.08 kg., or about 12 per cent., of coke when compared with coke carbon consumption in diagram 1.

It should, however, be observed that increasing the blast temperature is not the only means available in furnaces to obtain a decrease of direct reduction in the hearth of the furnace under consideration, since altering the quality of coke should produce similar results. The use of denser coke, for instance, would decrease the reductive action on the CO_2 rising from the lower portions of the furnace. The lower limit of operation thus attainable would be indicated in the diagram and would be at the intersection of the line of constant reduction with the line of the m -proportion = 0.80. According to experience at present, a better m -proportion cannot be assumed. The working condition thus attained should correspond with operation point *B* in diagram 1, which would point to a coke consumption of 0.65 kg. and a corresponding saving of coke of 4.6 per cent. The diagram, however, makes it also evident that in such a case an increase of the blast temperature could not effect a further economy of coke because then an m -proportion of unattainable magnitude would have to be present.

The working out in practice alone can show whether a denser quality of coke could be used without interfering with the present rate of production, or whether the use of such a denser and consequently less easily combustible coke would not materially retard the course of reduction in the hearth, lower the production and correspondingly increase the heat loss per kilogram of iron, and therefore necessitate a higher coke consumption.

A third means to decrease coke consumption would be the acceleration of the rate of production by which the proportionate heat loss per kilogram of iron would be lowered together with its attendant coke consumption.

In German operations, with charges of smaller yield than here employed, a not inconsiderably larger production per cubic meter of furnace capacity has actually been achieved. In regard to this there are two limits, the one being the combustibility or speed of reaction of the coke in the hearth, and the other the duration of the time that the molten mass remains in the hearth being so shortened that it is insufficient to allow a completely adequate separation of the smelted iron drops from the slag. In such a case the slag drawn from the hearth contains such a considerable quantity of fine suspended particles of iron that important losses of metal result. This limit must admittedly be variable according to the viscosity of the molten slag.

Recapitulation

The foregoing review based on detailed observation concerning the operation of a blast furnace furnished by Howland and with the aid of the theory of the reduction process developed by the author, demonstrates that a correct idea can be obtained of the process of operations taking place in the furnace under discussion.

The coke consumption arrived at theoretically coincides sufficiently with results established in practice.

It also demonstrates that even in an operation as perfect as the present one a saving of coke could safely be expected by an increase of the blast temperature from 600° to 800° and some further measures.

It may safely be anticipated that the systematic elucidation of conditions in other furnaces might in a majority of cases lead to the discovery of ways and means for further considerable saving of coke.

ABSTRACTS FROM THE CALCULATIONS

Concerning the working of furnace No. 19—Table 1, p. 628—the following data are given for the month of February, 1915:

American Measures		German Measures
1. Production	580 tons of iron	589 tons of iron
2. Consumption of coke	1,673 lb. containing 88.6 per cent. C	747 kg. per ton of iron
3. Consumption of limestone	780 lb. per ton of iron (100 per cent.)	348 kg. per ton of iron
4. Addition of scrap	75 lb. per ton of iron (85 per cent. Fe)	33.5 kg. per ton of iron

Analysis of produced pig iron:

	Per Cent.
Si	= 1.560
Mn	= 0.750
P	= 0.075
C	= 4.000
S	= 0.035
Fe	= 93.580

Analysis of furnace gas:

CO ₂	= 15.1 vol. per cent.; therefore $m' = 0.64$.
CO	= 23.6 vol. per cent.
N ₂	= 58.6 vol. per cent.
H ₂	= 2.8 vol. per cent.

Moisture in coke = 1.85 per cent.

Blast temp. = 594°.

Furnace gas temp. not given.

The consumption of pure carbon to produce 1 kg. of iron is therefore $0.747 \times \frac{88.6}{100} = 0.6615$ kg. That is assuming the average constituents of the Mesaba ore, according to literature, to be as follows:

1. Amount of iron in the ore 53.8 per cent.
2. Amount of H_2O in the ore 11.9 per cent.
3. Percentage of ash in coke 7.5 per cent.

Calculation of Production

One ton of iron contains 935.8 kg. Fe, of which $0.85 \times 33.5 = 28.5$ kg. are furnished by the scrap and the remainder of $935.8 - 28.5 = 907.3$ kg. by the iron ore. The ore contains 53.8 per cent. Fe = 76.9 per cent. Fe_2O_3 . Since 907.3 kg. Fe are equivalent to 1,295 kg. Fe_2O_3 , the amount of iron needed for 1 ton of iron is: $1,295 \times \frac{100}{76.9} = 1,684$ kg.

The constituents of the burden are:

	Kg.
1. Ore.....	1,684.0
2. Scrap.....	33.5
3. Limestone.....	348.0
4. Ash in coke.....	56.0
	2,121.5

The production is therefore: $\frac{1,000}{2,121.5} \times 100 = 47.2$ per cent.

Calculation of Data for Diagram

Calculation of slag. The burden per 1 kg. of iron is composed of:

- 1.295 kg. Fe_2O_3 .
- 0.200 kg. moisture in ore.
- 0.153 kg. CO_2 in the form of $CaCO_3$.
- 0.030 kg. Fe in scrap.
- 0.046 kg. oxide from Mn, Si, P, according to analysis of pig iron (Mn assumed as Mn_2O_4).
- y kg. slag

$$\begin{aligned} \text{Total burden} &= y + 1.724 \\ \frac{\text{burden}}{\text{pig iron}} &= \frac{100}{47.2} & 1.724 + y &= \frac{100}{47.2} \\ y &= 0.3975 \text{ kg. of slag per 1 kg. of pig iron.} \end{aligned}$$

In the calculations which follow these abbreviations have been used:

- C = the quantity of carbon in kilograms to produce 1 kg. of pig iron.
- C_{Fe} = the quantity of carbon in kilograms contained in 1 kg. of pig iron.
- C_x = quantity of carbon in kilograms consumed in the course of incidental reactions (splitting of CO_2 in shaft; reduction of Fe_2O_3 in the hearth).
- $C_{Mn, Si, P}$ = the quantity of carbon in kilograms consumed in the reduction of the oxides of Mn, Si, P.

From the total heat balance obtained in the usual way from the foregoing data, it follows that the actual loss of heat by radiation and cooling water amounts to 509 Cal. per 1 kg. of pig iron. In the calculations which follow, this total loss of heat has been assumed to be 500 Cal.

The theoretical values of the consumption of carbon and the respective temperatures of the furnace gas had been calculated assuming that the direct reduction takes place solely in the hearth, and that it amounts to 0, 10, 20, etc., per cent. and further assuming that the direct reduction of 10, 20, 30, etc., per cent. takes place solely in the shaft.

The values calculated in using the actual existing blast temperature of 594° are shown in Chart No. 1.

The values obtained in using the assumed blast temperature of 800° are shown in Chart No. 2.

The calculation of the quantity of carbon required and the temperatures of the blast-furnace gas have been made by separate heat balances, one for the hearth heat, the other for the shaft heat.

The following assumptions have been made for these heat balances:

1. That two-thirds of the loss of heat through cooling water and radiation takes place in the hearth and one-third in the shaft.

2. That the materials to be smelted in the hearth enter the hearth from the shaft preheated to 1,400°; and that the gases leaving the hearth have a temperature of 1,650°.

The hearth balance gives us the consumption of the coke carbon per 1 kg. of pig iron, and the shaft balance enables us to determine the corresponding temperatures of the furnace gas.

The separate hearth and shaft balances for the case: 0 per cent. "direct" reduction, heat loss = 0; X = coke consumption; Z = furnace-gas temperature, are:

I. Hearth Balance

Heat received and developed:

1. Combustion of C to CO = 2,140x - 96.6.

The iron absorbs per 1 kg., 0.04 kg. C.

Combustion to CO = $0.886x - 0.04$.

$(0.886x - 0.04) 2,416 = 2,140x - 96.6$.

2. Heat carried by charge entering hearth at a temperature of 1,400°:

	Quantity	Spec. Heat	Temp.	
(a) C.....	0.886x	0.35	1,400 =	434x
(b) Pig iron.....	1.0	0.17	1,400 =	+ 238
(c) Slag.....	0.3975	0.264	1,400	+ 147
(d) Oxygen of the oxides of Mn, Si, P, calculated from 0.046 kg. total quantity of oxide.....	0.0218	0.24	1,400	7.3

3. Blast temperature of the total carbon burned by the O_2 of the blast:

$0.886x - 0.04 - 0.01629 = 0.886x - 0.0563$.

C C_{Fe} $C_{Mn, Si, P}$

Quantity of blast is: $(0.886x - 0.0563) \frac{16}{12} \times \frac{100}{23}$.

Factor $\frac{16}{12}$ corresponds to the commutation of carbon in oxygen.

Factor $\frac{100}{23}$ corresponds to the commutation of oxygen in air.

The factors $\frac{10}{12} \times \frac{100}{23}$ give 5.8.

0.237 = spec. heat of the atmospheric air.

594° = blast temperature.

$$= 3,297x + 249.8$$

Heat absorbed and disbursed:

$$1. \text{ Heat carried by tapped pig iron; } 265 = 265.0$$

$$2. \text{ Heat carried by tapped slag } 0.3975 \times 450 = 178.9$$

Heat contained per kilogram of pig iron and slag
= 265 and 450 Cal. according to Grüner.

$$3. \text{ Direct reduction of iron: according to assumption } = 0$$

$$4. \text{ Direct reduction of Mn, Si, and P } = 120.9$$

$$(a) \text{ Mn } = 0.0075 \times 1,990 = 14.9$$

$$(b) \text{ Si } = 0.0156 \times 6,496 = 101.5$$

$$(c) \text{ P } = 0.00075 \times 6,014 = 4.5$$

$$120.9$$

Heats of reaction according to Le Chatelier, Berthelot and Thomsen:

$$5. \text{ Heat carried by the gas leaving the hearth with a temperature of } 1,650^{\circ}:$$

$$(a) \text{ Nitrogen: } (0.886x - 0.0563) \frac{16}{12} \times \frac{77}{23} \times 0.244 \times 1,650 = -101.2 + 1,592x$$

$$(b) \text{ CO: } (0.886x - 0.04) \frac{28}{12} \times 0.245 \times 1,650 = -37.8 + 836x$$

$$425.8 + 2,428x$$

Equating the heat received and developed with the heat absorbed and disbursed:

$$3,297x + 249.8 = 2,428x + 425.8.$$

$$869x = 176.0.$$

$$x = 0.2025 = \text{consumption of coke.}$$

$$C = 0.1794 = \text{carbon from coke.}$$

II. Shaft Balance

Heat received and developed:

$$1. \text{ Cooling of gas from } 1,650^{\circ} \text{ to } Z \text{ (furnace-gas temperature):}$$

$$(a) \text{ Nitrogen } (0.886x - 0.0563) \frac{16}{12} \times \frac{77}{23} \times 0.244 (1,650 - Z)$$

$$(0.886x - 0.0563) (1,797 - 1.089z) = 1,592x - 0.965xz - 101.2 + 0.0614z.$$

$$(b) \text{ Carbon monoxide.}$$

CO produced in the hearth.

Cools unchanged down to 1,000° C.

In temperatures above 1,000°, CO cannot be consumed through indirect reaction, as every molecule of CO₂, produced at a temperature above 1,000°, is through the action of free carbon (C), immediately reconverted

$$\text{into CO } (0.886x - 0.04) \frac{28}{12} \times 0.245$$

$$(1,650 - 1,000)$$

$$= 329x$$

$$-14.9$$

(c) From 1,000° on CO deoxidizes Fe_2O_3 in accord with our assumption: direct reduction = 0 is here the total amount of Fe_2O_3 in the charge. 1.295 kg. Fe_2O_3 give 1.07 kg. CO_2 . These give up heat in the process of cooling from 1,000 to z : $1.07 \times 0.2145 (1,000 - z) =$ + 229.7 - 0.2297z

(d) Cooling from 1,000 to z of the remaining CO. From the total amount of CO, 886x go to:

1. The carbon transferred to the iron = 0.040 =

2. The quantity corresponding to the CO_2 produced by indirect reduction = 0.292

0.332

The remainder of CO: $(0.886x - 0.332)$ goes as CO into the furnace gas.

Heat evolved: $(0.886x - 0.332) \frac{28}{12} \times 0.245$

$(10,008x) = 506x - 0.5060xz - 189.6 + 0.1896z$

(e) Cooling of the limestone CO_2 from 1,000° to z , $0.153 \times 0.2145 (1,000 - z)$.

The indirect reduction of the Fe_2O_3 through

CO generates in heat $1.295 \times \frac{2,280}{159.7} =$ 18.5

The equation $\text{Fe}_2\text{O}_3 + 3\text{CO} = 2\text{Fe} + \text{CO}_2$ generates per gram mol.: 2.228 Cal.

159.7 = mol. weight of Fe_2O_3 ,

Total: 2,427x - 1.471xz - 24.7 - 0.0015z

Heat absorbed and disbursed:

1. The heating to a temperature of 1,400° of the burden entering the hearth:

(a) Carbon (see hearth balance heat received, sub 2) = 434x

(b) Pig iron (see hearth balance heat received, sub 2) + 238

(c) Slag (see hearth balance heat received, sub 2) + 147

(d) Oxygen contained in the ore heat + 7.3

2. The preheating of the limestone CO_2 $0.153 \times 0.2145 \times 1,000$ + 32.8

3. Decomposition of the limestone 0.348×450 + 156.5

4. Preheating of the total quantity of the oxygen of the Fe_2O_3 to 1,000°, $0.3880 \times 0.24 \times 1,000$ + 93.2

5. Evaporation and superheating of the ore and coke moisture:

(a) Ore moisture 0.2 $(607 + 0.292z)$ + 121.4 + 0.0584z

(b) Coke moisture $0.02z(607 + 0.292z)$ 12.1z + 0.0058xz

Heat change of the superheated water vapors according to Renault.

446.1x + 796.2 + 0.058xz + 0.0584z

$$\begin{aligned}
 1,980.9x - 1.4772xz - 0.0599z &= 820.9 \\
 x &= 0.2025, \text{ substituted} \\
 399.2 - 0.2993z - 0.0599z &= 820.9 \\
 -0.3592z &= 421.7 \\
 z &= -1,173^\circ.
 \end{aligned}$$

The calculation therefore gives the blast-furnace gas the theoretical temperature of $-1,173^\circ$.

It is understood that this figure has only a theoretical value with the aid of which value and the aid of a corresponding number of figures obtained by the same means, the curves or lines representing the blast-furnace gas temperatures can be entered in the chart. Only those parts of the chart can be of practical value in which the recorded temperatures of the furnace gas are about 100° and higher.

The determination of x and z fixes those points in the chart which correspond to the assumptions on which the calculations were based, i.e., heat loss = 0, direct reduction = 0.

In the calculations which follow, it will be shown in shorter form, how the necessary ordinates for the other points in the chart are determined.

If the total loss of heat of the furnace through cooling water and radiations is 500 Cal. per kilogram of iron—of which 500 Cal., $\frac{2}{3}$ = 333 Cal. belong to the hearth and $\frac{1}{3}$ = 167 belong to the shaft—the x equation, previously computed under the assumption: heat loss = 0, will now be:

$$\begin{aligned}
 869x &= 176 + 333 = 509.0 \\
 x &= 0.586 \text{ kg. coke per kilogram of pig iron} \\
 C &= 0.519 \text{ kg. C per kilogram of pig iron.}
 \end{aligned}$$

The shaft balance equation with heat loss = 500 Cal. will be:

$$\begin{aligned}
 1,980.9x - 1.4772xz - 0.0599z &= 987.9 \\
 1,161 - 0.866z - 0.0599z &= 987.9 \\
 0.9259z &= 173.1 \\
 z &= 187^\circ
 \end{aligned}$$

With heat loss = 1,000 Cal. the x equation is:

$$\begin{aligned}
 869x &= 843 \\
 x &= 0.970 \\
 C &= 0.859 \\
 \text{The } z \text{ equation } 1,980.9x - 1.4772xz - 0.0599z &= 1,153.9 \\
 1,921 - 1.434z - 0.0599z &= 1,153.9 \\
 767 &= 1.494.z \\
 z &= 514^\circ
 \end{aligned}$$

If the direct reduction is 10 per cent. and takes place only in the hearth, it means that 10 per cent. of the total quantity of Fe_2O_3 is re-

duced by solid carbon. The quantity of Fe_2O_3 is 1.295 kg., 10 per cent. = 0.1295 kg. require for their reduction of solid carbon:

$$0.1295 \times \frac{36}{159.7} = 0.0292 \text{ kg. C.}$$

The fraction $\frac{36}{159.7}$ corresponds to the ratio of the molecular weights 3C to Fe_2O_3 .

The following values in the hearth balance are thereby changed:

(a) Under heat received and developed, subdivision 2 (heat carried by the burden into the hearth) will be augmented by the heat quantity which the oxygen of the 10 per cent. of Fe_2O_3 carries with it into the hearth. This heat quantity is 16.6 Cal.

(b) Under heat received and developed, the algebraic expression, corresponding to the blast heat, sub 3 will be smaller, because the oxygen of the blast burns a quantity of carbon smaller by 0.292 kg. than is burned in the case 1.

The blast heat is therefore:

$$(0.886x - 0.04 - 0.0163 - 0.0292) 5.8 \times 0.237 \times 594 = 723x - 69.8.$$

It must be remarked, so as to avoid mistakes, that the absolute quantity of the blast heat depends on the values of x , as determined by the final equation.

(c) Under heat absorbed and disbursed, the heat quantity necessary for the reduction of 10 per cent. Fe_2O_3 through C has to be substituted. The oxygen bound to 10 per cent. Fe_2O_3 is 0.0388; the needed quantity of heat is therefore $0.0388 \times 4,200 = 163.0$ because, for 1 kg. of oxygen, dissociated from the Fe_2O_3 through the reduction process, a heat quantity of 4,200 Cal. is needed.

(d) Corresponding to the variation in quantity of the blast, a variation in the quantity of nitrogen takes place, and therefore a variation of the heat taken from the hearth by the nitrogen. This heat quantity is:

$$(0.886x - 0.04 - 0.0163 - 0.0292) 1,797 = 1,592x - 153.6$$

Therefore, the balancing of hearth heat received and heat disbursed gives:

$$869x = 301.2$$

$$x = 0.346$$

$$C = 0.312$$

Shaft Balance

Heat received and developed:

The changes which appear in the shaft balance are somewhat greater, and they are therefore again related here in detail:

1. Cooling of gas from 1,650 to z :

$$(a) N_2 (0.886x - 0.0855) (1,797 - 1.089z) = 1,592x - 0.965xz + 0.0932z - 153.6$$

$$(b) \text{Cooling of CO to } 1,000^\circ \text{ remains unchanged} = 329x \quad - 14.8$$

$$(c) \text{Cooling of CO}_2 \text{ produced by reduction of } 90 \text{ per cent. of Fe}_2\text{O}_3, 0.963 \times 0.2145 (1,000 - z) \quad - 0.2065z + 206.5$$

$$(d) \text{Cooling of the remaining CO from } 1,000^\circ - z \text{ from total amount of } - C \text{ are subtracted}$$

$$1. C_{Fe} = 0.040$$

$$2. C$$

The CO_2 obtained in the indirect reduction

$$\text{equals } \frac{0.2626}{0.3026}$$

The remainder passes as CO into the blast-

$$\text{furnace gas } (0.886x - 0.3026) \frac{28}{12} \times 0.245$$

$$(1,000 - z) = 506x - 0.5060xz + 0.1728z - 172.8$$

$$(e) \text{Cooling of the limestone CO}_2 \text{ unchanged} \quad - 0.0328x + 32.8$$

$$2. \text{Reduction of 90 per cent. Fe}_2\text{O}_3 \text{ by CO} \quad + 16.6$$

$$2,427x - 1.471xz + 0.0267z - 85.3$$

Heat absorbed and disbursed:

For the establishment of heat balance, the only point to be considered is, that only 90 per cent. of the oxygen of the Fe_2O_3 is heated to $1,000^\circ$, and that 10 per cent. has to be heated to $1,400^\circ$. As no other change takes place, the total value of the heat disbursed varies little from that obtained at 0 per cent. direct reduction:

$$446.1x + 796.1 + 0.0584z + 0.0058xz.$$

The balancing of heat received with heat disbursed and the insertion of x gives for heat loss = 0: $z = -358$.

$$\text{For heat loss} = 500x = 0.730 \quad \text{For heat loss} = 1,000^\circ x = 1.114$$

$$C = 0.6465$$

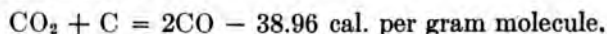
$$C = 0.987$$

$$z = 361^\circ$$

$$z = 592^\circ$$

The balances are correspondingly computed for 20, 30, etc., per cent. direct reduction in the hearth.

If the direct reduction is transferred into the shaft, this means that a certain amount of the CO_2 , derived from the direct reduction of Fe_2O_3 , is dissociated by the carbon. Ten per cent. direct reduction means that in the indirect reduction of 1.295 kg. Fe_2O_3 , 1.07 kg. CO_2 are produced, of which amount 10 per cent. = 0.107 are dissociated by the carbon according to the equation:



The requirement of C is 0.0292 kg., that is the same quantity, which at

"10 per cent. direct reduction in the hearth" there directly reduced the Fe_2O_3 .

Compilation of the Values Entered in the Charts 1 and 2

CHART 1

A. Direct Reductions in the Hearth Only

Direct reduction, per cent.	W.V.*	C	Σ
0	0	0.175	-1,173°
	500	0.519	187°
10	0	0.312	-358°
	500	0.647	361°
20	0	0.430	52°
	500	0.770	491°
30	0	0.557	134°
	500	0.897	585°

B. Direct Reductions in the Shaft Only

Direct reduction, per cent.	W.V.	C	Σ
10	0	0.204	-1,355°
	500	0.550	104°
20	0	0.233	-1,620°
	500	0.574	20°

CHART 2

A. Direct Reductions in the Hearth Only

Direct reduction, per cent.	W.V.	C	Σ
0	0	0.152	-1,537°
	500	0.415	80°
10	0	0.258	-662°
	500	0.521	129°
20	0	0.363	-219°
	500	0.627	280°
30	0	0.468	58°
	500	0.732	396°

B. Direct Reductions in the Shaft Only

Direct reduction, per cent.	W.V.	C	Σ
10	0	0.181	-1,730°
	500	0.445	199°
20	0	0.210	-1,950°
	500	0.474	318°

*W.V. means heat loss.

ment of the balance it must be considered, that the hearth receives a carbon quantity smaller by 0.0292 kg. and that in consequence, the heat developed in the combustion process of C to CO, as well as the heat carried by C from the shaft into the hearth will be correspondingly smaller. Further in the establishment of the shaft balance, under heat received, we must take into consideration that it has to be assumed first of all that the total Fe_2O_3 is reduced by the CO; and that only a corresponding amount of the so generated quantity of CO_2 is dissociated by the C. All other items of the balance remain unchanged.

In changing the heat blast to 800° , no change takes place in the balance with the exception of the heat values carried by the heated blast.

Portable Miners' Lamps

Discussion of the paper by EDWIN M. CHANCE, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 235 to 243.

E. C. LEE, Pittsburgh, Pa. (communication to the Secretary*).— I would like to add a word to the arguments presented by Mr. Chance and others regarding the superiority of carbide lamps over open-flame oil lamps.

The chief argument by advocates of oil lamps, is their sensibility to the presence of black damp. This is entirely correct. The oil lamp is sensitive to the presence of black damp, so sensitive, in fact, that it goes out in the presence of a low percentage of black damp. The question in my mind is whether the oil lamp, by going out in the presence of a low percentage of black damp, is more conducive to the safety of the miner than a carbide lamp, which indicates the presence of black damp to those who have been instructed as to its behavior in the presence of black damp; but which does not go out until the proportion of black damp present is considerably higher than that required to extinguish an oil lamp, while still being lower than the percentage which is dangerous to men. Personally, I would much prefer having a lamp that I know would go out before the atmosphere became too contaminated to support life, rather than have my light go out in the presence of the relatively small amount of black damp necessary to extinguish an oil lamp. I believe that any man who has had the experience of groping his way along the rail after his lamp has been extinguished by black damp will agree with me in this statement.

* Received April 7, 1917.

The Significance of Manganese in American Steel Metallurgy

Discussion of the paper of F. H. WILLCOX, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 199 to 207.

D. F. HEWETT, Washington, D. C. (communication to the Secretary*).—I am not prepared to discuss the metallurgical use of manganese in the form of alloys. In connection with other work for the U. S. Geological Survey, however, I have been investigating incidentally, for about 5 years, the American as well as other sources of manganese. There are some conclusions that come out of this investigation that may interest those whose greatest interest lies in the use of the alloys.

From a geological point of view, most manganese ore deposits may be regarded as concentrated masses of manganese oxides, as there are few masses of the other manganese minerals that are rich enough in manganese and low enough in silica to warrant smelting. The deposits of the Huelva district, in Spain, which are largely manganese carbonate and silicate, are exceptions to this statement. Some deposits of manganese oxides, such as those of Russia and Cuba, are bedded with ancient sediments, and although they are clearly concentrated under unique conditions, the source of the manganese cannot be determined. Probably 60 per cent. of the world's production of manganese ore, or 95 per cent., if that of Russia be excluded, is derived from the superficial parts of deposits that have undergone weathering and enrichment near the surface. In other words, the situation with the manganese ore supply is such as it would be with regard to copper, if instead of being derived largely from deep zones of deposits of deep-seated origin, such as veins, it were derived solely from the zone of secondary enrichment.

Manganese oxide occurs in all rocks but is most abundant in igneous rocks, of which, according to Clarke, it forms an average of 0.10 per cent. It also forms 0.05 per cent. of the average limestone, but only traces are present in shales and sandstones. It is interesting that although many commercially important manganese deposits occur in rocks that are but slightly richer in manganese than the average, the most important deposits occur in rocks of unusual character, that are much richer in manganese than the average. Thus, the most important deposits of India and Brazil overlie or occur near masses of rocks that contain excessive amounts of manganese garnet and pyroxene.

The chemical processes which aid in the formation of large masses of manganese oxide near the surface are obscure and probably complex. From available knowledge, it may be stated that many manganese de-

* Received April 3, 1917. Published with the permission of the Director of the U. S. Geological Survey.

...of and climates, which favor rock disintegration rather than decay.
Even if allowance be made for the possible discovery of deposits not now known, it is apparent to any person who has given a little thought to the subject, that the United States is woefully deficient in large manganese deposits. This may be based upon two conditions. Either, on the one hand, there are few if any large masses of manganese-bearing silicates that could yield by weathering important quantities of manganese oxides, or, on the other hand, the climate and types of weathering now prevailing in the United States are not such as are capable of producing large accumulations of manganese oxides from the sedimentary rocks which so largely underlie the United States.

I realize that generalized prophecies not based upon the thorough consideration of much data are little more than guesses, and where applied to ore deposits, are hazardous, but I think we are warranted in hoping that, within the next decade perhaps, there will be developed in Central and the northern part of South America, where the climatic conditions favor deep rock decay, deposits of manganese oxide capable of contributing largely to our needs. Thus far, only a few manganese deposits which are clearly formed by processes of rock decay are known in this region, but recently one was discovered in Costa Rica that appears to offer prospect for large production. With the shortage of manganese ores and the consequent high prices, the search for deposits becomes more active. It seems to me that those interested in the manganese ore supply would do well to investigate any statements which indicate that new deposits have been found in Central America or northern South America.

Geology of the Iron-Ore Deposits of the Firmeza District, Oriente Province, Cuba

Author's reply to discussion* of the paper of MAX ROESLER, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 118, October, 1916, pp. 1789 to 1839.

MAX ROESLER, Firmeza, Oriente, Cuba (communication to the Secretary†).—It is substantially admitted by all who have recently been in touch with these deposits that the orebodies lie in the fine-grained dark-green rocks, called diorite by most writers and diabasic by myself. The three points of disagreement concern the agency of included limestone blocks as causes of deposition, the agency of the

* *Bulletin* No. 124 (April, 1917), 439

† Received April 5, 1917.

granite as a source of the mineralization, and the primary or secondary nature of the hematite.

In regard to the first question there is a complete sequence of opinions. Professor Lindgren believes that the orebodies are localized in the limestone and does not admit replacement of the diorite. Professors Singewald and Miller admit the replacement of diorite (diabase), but only in the vicinity of limestone. Professor Kemp and I are in entire accord in considering the mineralization of the diorite (diabase) as entirely independent of the limestone.

To the evidence already submitted in regard to the mineralization of the diabase, I have very little to add. Duplication, of garnet and epidote rims in diabase next to aplitic dikes, and of cases of retained porphyritic structure in thoroughly mineralized diabase porphyry, has made me feel very sure of the replacement of the diabase. My work in the Daiquiri district has added to the number of occurrences in which, to my knowledge, no limestone is in evidence near the ore. It is, of course, not possible to prove that where there is now no evidence of limestone, there never was any. But since the prospect work based on the theory that the ore was localized along a system of fractures has in several cases succeeded in cutting extensions of the ore zone, I have faith in the latter theory. Furthermore, in the recognizable contact bodies in limestone, the mineralization is preponderantly in the form of specular hematite. In the cases where a coarse diabase porphyry has been replaced, and the porphyritic structure maintained, the mineralization is preponderantly magnetite. In the larger orebodies in which no limestone is in evidence, the mineralization is largely massive magnetite with an intergrowth of a minor and varying amount of hematite. It seems fair to believe that the geochemical processes that produced the larger bodies are more closely allied to those that produced the replacement of the diabase porphyry than to those that acted in the case of the limestone.

The question of the relation between ore and granite rests on two points: first, the relation of the granitic apophyses to the granite massif; second, the relation of the granitic apophyses to the ore.

The first point is a question of fact. As I have already said in the article under discussion (page 1802):

"The granitic apophyses, as now exposed, are rarely more than 150 ft. above the top of the granite massif, and their downward extension where exposed by erosion can be seen to merge into the granite."

Further study convinces me that in several places the granitic apophyses have reached a higher limit away from the granite, for the rest I stand by that statement.

The second point is a question of interpretation. Any interpretation that seeks to dissociate these apophyses from the ore must offer adequate explanation for the following facts.

these associations between granitic rocks and ore. In fact, a study of the Juraguá Iron Co's tonnage sheet shows that over 84 per cent. of the ore mined since 1908 comes from mines in which there is such an association. In deriving those figures, the Ocaña mine was considered as away from granite. Considering that mine as directly connected, the total figure would become over 97 per cent.

In all the orebodies that are being mined or prospected near Daiquiri, there is such an association. That is, if one kept away from granitic rocks in that area the mines would be shut down.

The granitic apophyses contain inclusions of magnetite. They are accompanied by garnet rims. They can be seen to change into epidote and quartz veins.

I can see no interpretation that will explain all the facts, except that these apophyses were a feature of the mineralization, and were contemporaneous with it as well as extending beyond it in duration. Dr. Berkey has already answered the suggestion that if they were associated with the ore they would have to be coextensive with it.

It seems to me, so far as the theory of ore genesis is concerned, that the evolution of highly siliceous emissions from a basic magma would be more than likely to be accompanied by the production of an acidic magma by differentiation. In this case the acidic magma has reached near to the point of deposition of the emissions and has been exposed by erosion. But the difference in viscosity between such emissions and aplitic material must be so great that the emissions might travel very far from their accompanying magma. So, even if there were no exposure of granite, the amount of siliceous material accompanying the ore would, in my opinion, justify the postulating of a magma of a more siliceous nature than the diabasic, as a hidden and immediate source for the mineralization.

The Ocaña mine has been cited as an example of an orebody whose genesis is different from the genesis that I worked out for the other mines in the Firmeza district. As a matter of fact, it is the poorest of all the active mines for study, because the ore lies along a hillside, close to the surface, and both it and the wall rocks have been subjected to extreme superficial alteration. However, I have found some further evidence in regard to the mineralization of this mine, in the course of my work during the last 7 months.

There is undoubtedly replacement of limestone by specular hematite, as evidenced by typical contact phenomena. There is equally definite replacement of the coarse diabasic porphyry, as evidenced by a friable mass of magnetite containing recognizable, kaolinized feldspars. These two types of mineralization are in close proximity in this mine. On

the other hand, we have in most of the mines only one type, or at least one type vastly predominant. Do not these circumstances point to a dominating cause outside the nature of the rocks rather than to an interdependence? I feel that they do, and that the dominating cause was physical; i.e., temperature and pressure.

In some of the lower and newer workings I have found an aplitic dike, and the opening up of some new ore farther down the hillside brings the bottom of known ore to within 75 ft. vertically of the outcrop of the contact of the diabase and a mass of quartz porphyry. The superficial alteration is too great in this mine to make it possible to prove the relationship. On the other hand, there is quartz andesite all through the mine, and some aplite. The independence of the mineralization and acidic rocks is certainly not shown.

Mr. Graton's suggestion as to the need of more work on the question of hematitization is in accord with my own ideas. I hope at some future date to have more to say on this subject.

Professor Miller's criticism of my statement in regard to the formation of the lower coral limestone terrace by sea movement is entirely justified. The statement, as quoted by him, is far more definite than I had any right to make it. It was meant to be more in the nature of a suggestion.

An Investigation on Rock Crushing Made at McGill University

Discussion of the paper of JOHN W. BELL, presented at the New York Meeting, February, 1917, and printed in *Bulletin* No. 122, February, 1917, pp. 171 to 176.

A. O. GATES, Salt Lake City, Utah (communication to the Secretary*).—The writer is delighted by the results shown in Mr. Bell's paper, which prove in an experimental way different from that followed by the writer¹ that "the work done in crushing is proportional to the surface exposed by the operation," per Rittinger's Law.

Mr. Bell's work was all done in commercial types of crushing machines, while my work was done in a testing machine; his work represents fully one hundred times the expenditure and time that my work represented; a great number of tests were made by his organization to determine single points, while I was obliged to depend on the general trend of a curve for results; we agree absolutely that "Kick's law," for which Mr. Stadler was sponsor, does not apply to rock crushing. While my work indicated the reliability of Rittinger's theory, Mr. Bell's work clearly establishes the reliability and makes it Rittinger's law.

* Received April 4, 1917.

¹ *Trans.* (1915), 52, 875.

While Mr. Bell clearly recognizes the variation in the value of the minus 200-mesh product, I hope his paper will not set a precedent in the adoption of 780 as the average reciprocal of diameter of the minus 200-mesh. I indicate, on page 900 of my paper, a method of approximating this value, but I hesitate to set any value to it. If the curve of "per cent.-through" plotted against "reciprocals of diameters" follows the law of the hyperbola there will be as much, or more, surface in the finest 1 per cent. as there is in the next finest 9 per cent., and as much in the finest 0.1 per cent. as in the rest of the finest 1 per cent., and so on.

Probably the safest way to make comparisons is to neglect something like the finest 1 per cent. from all calculation, and determine the surface in the minus 200-mesh and plus the finest 1 per cent. by assuming that the 200-mesh points lie on the straight line of the logarithmic plot shown on page 900 of my paper.

With regard to choke crushing: Mr. Bell in his 1913 tests, as shown in Table 2, obtains much higher results by small reductions and by feeding singly to avoid choke crushing, while in section 31 of my paper I indicate the same thing. But I wonder if these are not *apparent* higher results, for the reason that in "choke crushing" there is probably an excess of unmeasured surface in the minus 200-mesh and minus 2,000-mesh to account for the power consumed. Probably there is no economic value in the minus 2,000-mesh except in Portland cement; the practical man may neglect it, but theorists must take it into consideration.

Mr. Bell is not convinced that the work could be carried on to better advantage in a testing machine such as I used, than in the commercial machines that he used. Of course, the best way for him to learn that it could, would be to try it. My first "testing machine" was a 22-lb. stamp. The idea of the 22 lb. was that when dropped 1 ft. 1,500 times, 1 hp. was expended. I used this in 1909 long enough to prove that the "efficiency" of certain rolls was about 25 per cent. How much of the horsepower was expended in vibrating the kitchen floor was not recorded.

I did consider a small set of rolls for the work I wished to do in determining the crushing constant of rocks, but when I went to Purdue shortly thereafter, I cast about for a suitable machine in the University laboratories. The ordinary power-driven Riehle and Olsen machines I rejected because of the trouble in getting power and in controlling it when I did get it. We had a drop testing machine with a very heavy weight, with a drum on which the rebound could be measured so that the actual energy spent could be readily calculated, but before adopting this device I came upon the Amsler-Laffon machine which so admirably served my purposes.

Such testing machines must be in the laboratories of many of our universities and cement plants. Washington University has one at St. Louis and the Portland Cement Co. of Utah has one, both being of

American make. There was no difficulty in measuring the energy absorbed; in fact, my machine had an automatic recorder. My machine was just about 100 per cent. efficient, for I had actual measurement of the pressures exerted between the dies and the actual distance the dies were moving measured right at the point of crushing. I believe my method eliminated all friction losses, and therein lay its scientific advantage.

I would like to see some of our Schools of Mines put into their mining laboratory courses such tests as I made. These tests could be made and worked up in two afternoons. They would perhaps make the theory of crushing plainer to the student and would stimulate more research along this rather important line of milling. Outside of the machine, the expense would be small, and with the right supervision, valuable data could be accumulated.

Now that we have what I believe is satisfactory proof to most engineers of the manner in which energy is distributed in crushing, the next step is to make further practical use of it. Constants should be determined for the various rocks that are crushed. This should be done by the U. S. Bureau of Mines. Institute Committees should consider the standardization of the measurement of crushing—whether we shall speak of it in terms of “surface units” or “mesh-tons,” and if the latter, whether we shall use the Tyler mesh number, or the corresponding reciprocal-of-diameter.

I congratulate Professors Porter and Bell and their associates on the success they have achieved in this investigation, and the milling fraternity upon having one important point absolutely settled.

Stoping in the Calumet and Arizona Mines, Bisbee, Ariz.

Discussion of the paper of PHILIP D. WILSON, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1099 to 1117.

CLARENCE M. HAIGHT, Franklin, N. J. (communication to the Secretary*).—In that part of Mr. Wilson's paper describing the Gilman cut-and-fill system, a few features do not appear to be fully explained.

In what way is the waste used for filling obtained? The text mentions that if waste is encountered when driving the filling raises, these raises are stopped until the fill is needed; that seems to suggest mining the capping for fill, which might make the backs of the stopes bad as the stopes approach the top of the ore. If, on the other hand, the fill is brought to the raises in cars, are the crosscuts that are necessary in ore or waste? Is the cost of this fill included in the stoping costs given?

* Received Mar. 9, 1917.

will have to be worked on center lines which are offset half the stope width, 20 ft., from the centers of the stopes above, in order to take advantage of the "rafter" effect of the stringers so named. How is the fill to be introduced into the lower stopes? There are no crosscuts mentioned to provide for this; or is it intended to tap the filled stopes above for the fill? In the latter case the method of control of this fill will be of interest.

Ore that will admit of a stoping width of 40 ft. with the lengths mentioned in the description is stronger than any with which many mining

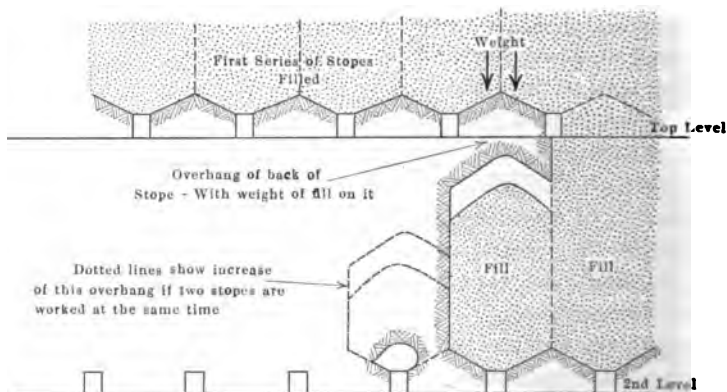


FIG. 1.

men are familiar. Has the practice of starting to mine a second stope before the preceding one is fully completed been tried to any extent? If so, has it worked as well as was anticipated? The stoping width in such practice becomes greatly increased in parts of the stope, and where a stope has already been finished, the top or back of the stope that is being worked becomes an overhang instead of an arch.

Mr. Wilson says that no stopes of the second series have as yet been brought up under the completed ones. Conditions under which the second series of stopes will operate when nearing the filled and completed ones above, will be greatly different from the original ones, especially after one stope has reached completion; the accompanying sketch will show this better than a description.

The dead weight of the fill in the old stopes will be much heavier than any encountered from an undisturbed capping. At Franklin, in the top slices, where the fill above the working places consists of rocks of all sizes packed only by their own weight, a heavy crushing effect due to weight is felt when only 20 ft. of length (about 3 sets wide) of a working

place is exposed, and it is rare that the timbers will hold up for more than one set width more, about 27 ft. When this second stage has been reached and tried, doubtless many men who are actively interested in mining will be glad to learn how the system works out and how the ground behaves under the weight.

Is there a special reason for the use of the long stringers, *a*, rather than sets? As the chutes make it necessary for at least one post, *c* (and two are used when necessary) under each stringer, sets would require less timber and, being shorter, could be more easily handled. Two legs of 10 by 10-in. timber 8 ft. long and a cap of 12 by 12-in. timber 7 ft. long would give the same strength over the track and the same clearance. The legs would be on the solid, so no sills would be needed. All the other arrangements could remain the same; the 18-ft. rafter stringers could be braced against the legs and still reach to the side of the stope. Assuming both posts, *c*, as necessary under each stringer, the sets would require 108 ft. (b.m.) less of timber for every 5 ft. in the length of the stope; with only one post under each stringer, there would still be a slight saving in favor of the sets.

If the cutting-out stope mentioned should be carried a few feet higher, the sets could be put in place and blocked temporarily, until the stope had reached its width and the rafter stringers placed. In this way the temporary timbering with "Cousin Jack" stringers could be eliminated. Unless the extra cost of mucking the additional ore broken to the floor by carrying the cutting-out stope a few feet higher exceeds the cost of erecting and dismantling the temporary timber work, the sets would appear to be worth trying, unless, of course, there is a special reason for the non-use of sets. Does it not happen in actual practice that the cutting-out stope is carried higher than is actually necessary for the temporary timbering, so that in reality the "sets" plan would not mean any more mucking than at present?

PHILIP D. WILSON, Warren, Ariz. (communication to the Secretary*).—In replying to Mr. Haight's discussion, I will endeavor to take up the points to which he calls attention approximately in the order in which he mentions them.

The waste used for filling is obtained almost entirely from exploration or development work in barren ground. It is often possible to regulate the amount of such work by the need of filling in the stopes. The capping of a cut and fill stope is never mined for fill. Whether or not the crosscuts through which the fill is brought to the raises are in ore or waste depends entirely upon the shape and contour of the orebody on that level. The cost of handling and banking filling in the stope, usually insignificant except as the top of the orebody is approached, is in-

* Received Apr. 5, 1917.

cost of tramping to the shaft, hoisting and dumping on the surface.

The second, or lower, series of stopes is worked on center lines directly beneath the centers of the stopes above. As yet no stopes of the lower series have been carried up to the sill of the stopes above. The intention is, however, to erect square sets on the consolidated fill, coming under the long horizontal stringers three sets or more wide. Timbering will be started when within 16 ft. of the level. When horizontal stringer is once caught up it will be a simple matter to come under the rafter stringers with stulls, or with square sets if necessary.

The main purpose of the long horizontal stringer is to provide a simple and safe means for catching up the filled stope above when the lower stope comes up beneath it. While this stage has not yet been reached in this type of stope, very similar conditions have been encountered many times in mining beneath large filled open stopes. It has been found that the filling, added in comparatively thin layers and subjected each time to the pressure of the broken ore, becomes by the time the stope is finished thoroughly consolidated and compact, in effect almost as strong as virgin ground, requiring blasting to loosen it. This would not, of course, be true of the thoroughly loosened matte above the Franklin top slices, instanced by Mr. Haight, where the dead weight of the fill is tremendous.

Filling is introduced into the lower series of stopes through raises which are holed into the original crosscuts on the level above. It is probable that the final 16-ft. square-setted section directly beneath the level will be filled by tapping the filled stopes above, but this has not yet been done.

The practice of starting a second stope while the preceding one is still being mined has been quite successful. It has been found advisable to fill this second stope from a raise near the section line of the original stope with the peak of the cone consequently close to the gob lagging, and not to attempt to maintain an arch in the back of the stope.

The length of the horizontal stringers, *aa*, should have read 18 ft., that of the rafter stringers, *bb*, 14 ft. 2 in., in the original paper, page 1114. Thus the apparent saving of timber by Mr. Haight's suggestion that sets be used instead of the horizontal stringer and posts is eliminated.

Furthermore, as stated above, the use of the latter makes it a far simpler matter to catch up the filled stope when coming up with a new stope from below. The use of the temporary "Cousin Jack" stringers has been a decided success. Their erection is very simple and rapid and any such means of substantially reducing the labor item is particularly valuable in a district where wages are so high. Cars can be loaded just as rapidly through the floor laid on the "Cousin Jack" stringers as from the permanent stope chutes.

Diesel Engines Versus Steam Turbines for Mine Power Plants

Author's reply to discussion* of the paper of H. HAAS, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1171 to 1183.

H. HAAS, San Francisco, Cal. (communication to the Secretary†). —Fig. 1 plainly shows that the comparison of the steam-turbine and Diesel-engine plants was made on a basis of 6,000 kw. continuous operating load. The tabulation of costs for an 8,000-kw. load is merely done to show the relation of increased output on the unit power costs. Both plants would sacrifice their reserve capacity, which is to insure continuous operation, if operated to generate 8,000 kw. continuously; unless Mr. Hawkins advocates the operation of one of the turbines at 33.33 per cent. overload, retaining the other as a standby, which, besides overtaxing the turbine, results in a great loss of efficiency.

It would be well for Mr. Hawkins to qualify certain of his statements. Thus, under the caption "Diesel Engines, Selection of Units," he says that "two of the largest Diesel-engine builders take the position that, in the present state of the art, they would not attempt to put out units over 1,000 kw.;" he should qualify it by saying that "two of the largest *American Diesel-engine builders* take the position that, in the present state of the art in the United States, they would not attempt to put out units over 1,000 kw."

The reason that American manufacturers restrict themselves at present to units of that size is to be found not so much in a lack of experience in building large units as in a lack of a sufficient demand for them in the United States to justify the heavy capital outlay connected with the development of such engines. There are relatively few localities in the United States where fuel prices are as high as they are in the Southwest, and regions of high fuel prices are sparsely settled and have a very limited demand for power. Where *low* fuel prices obtain, steam turbines are active competitors of Diesel engines for *large* power plants, particularly when the station load factor is *low*. As most of the American manufacturers of Diesel engines work under license agreements with European manufacturers, which limit their market to the United States and its possessions and Canada, a further limitation is imposed upon them, leaving to European firms practically the entire world as a market.

The greater number of power plants range in size from 300 to 1,000 hp.; the average size of the central station, on a basis of all central stations in the United States, not greatly exceeding 500 kw.; in these sizes it is

* *Bulletin* No. 120 (December, 1916), 2213.

† Received Mar. 19, 1917.

hard for steam engines to compete with Diesel engines, unless fuel oil prices are excessively high, and coal prices very low, or the exhaust steam is used for heating purposes.

When Mr. Hawkins makes the statement: "While it is true that in Europe units in excess of 2,000 hp. have been built in the two-cycle type, they are not considered, even by the manufacturers, to be out of the experimental stage," he cannot be familiar with the development of large Diesel engines abroad, where a number of manufacturers build engines in units of 3,000 hp. and 4,000 hp. and engines of that size have been in successful operation for a number of years.

As is plainly stated in Fig. 1 and in Table 1, the units selected for the Diesel-engine power plant are four 3,000 hp. (2,000 kw.) Sulzer vertical single-acting two-stroke cycle engines. They are engines with six cylinders each and are built by Sulzer Bros. of Winterthur, Switzerland. Engines of this type are past the experimental stage and are in successful operation in different parts of the world.

Another item needing correction should be stated: the two 1,250-hp. Diesel engines installed at Tyrone, N. M., were built by Carels Bros. of Ghent, Belgium, and only the third unit was built by the Nordberg Mfg. Co., the licensee of Carels Bros.

Cost of Plant

As for Mr. Hawkins' "revised" figures of "Cost of Plant," I will say that, having been connected as engineer with one of the foremost European Diesel engine builders and also with the leading manufacturer of Diesel engines in the United States, I have cost data that Mr. Hawkins may possibly not possess. The costs as stated were conservative at the time the estimates were made, namely, late in 1915. They would not hold good at present as all materials have since then greatly risen in price.

Mr. Hawkins quotes my article in the *Engineering and Mining Journal* of Apr. 26, 1913, incorrectly. The plant there considered is of 1,200-kw. capacity, not of 1,400-kw. It comprises three 400-kw., not three 450-kw. units. These are single-acting, four-stroke cycle, four-cylinder engines and are not comparable in size or character of engine to a plant of four 3,000-hp. units of two-stroke cycle engines. Moreover, the higher cost of the small plant was partly due to special flywheel type of alternator, whereas the cyclic regularity of multicylinder two-stroke cycle engines is so good that standard type of generator can be coupled direct to the engine.

Fixed Charges

Mr. Hawkins' arguments, that the life of Diesel engines is an unknown factor, the present engines having been developed within com-

paratively recent years, might with equal force be made as regards large steam turbines using high steam pressures and highly superheated steam, developed but recently. There are Diesel engines running today that have been in continuous operation for more than 15 years with practically undiminished economy. The amount of interest and amortization to be charged in any well-equipped power plant is greatly a matter of financial policy and not so much a question of the actual life of the plant. In our age of technical and industrial progress, plants lose their usefulness through obsolescence rather than actual deterioration, and the management with foresight favors high amortization charges, *i.e.*, short life, to provide a sinking fund for the replacement of obsolete with new efficient machinery.

With this principle in mind, the Diesel plant should be the one more favored of the two, as with the constant rise in fuel prices, the economic advantage is entirely with the Diesel engine plant for the conditions considered.

Fuel

Mr. Hawkins argues that a fuel consumption of 0.64 lb. per kilowatt-hour cannot be obtained with engines of the two-stroke cycle type of that size, and cites my article in the *Engineering and Mining Journal* of Apr. 26, 1913, in which I said that the fuel consumption of two-stroke cycle engines is 10 per cent. higher than that of four-stroke cycle engines. My figures in my later article do not in the least contradict statements made in the earlier one. All I can say is that well-built, four-stroke cycle Diesel engines, even of *moderate size*, will deliver one brake-horsepower-hour with a fuel consumption at or near full load not exceeding 0.40 lb., and two-stroke cycle engines with a fuel consumption of 0.44 lb., which is in accordance with statements made in the article referred to by Mr. Hawkins.

These are not fuel consumptions merely obtained during shop tests, but are secured in actual, continuous operation of Diesel engines. The following is the commercial performance of a Diesel-engine plant in the United States, secured with four-cylinder four-stroke cycle Diesel engines directly connected with alternating-current generators.

Fuel oil consumption per kilowatt-hour delivered to the switchboard:

At full	load, 0.590 to 0.610 lb., average 0.600 lb.
At three-fourths	load, 0.590 to 0.614 lb., average 0.602 lb.
At one-half	load, 0.686 to 0.703 lb., average 0.692 lb.
At one-fourth	load, 0.880 to 0.966 lb., average 0.932 lb.

The fuel consumption of 0.64 lb. per kilowatt-hour for the 3,000-hp. two-stroke cycle Diesel units is conservative, and is based on operating results with such units. This high economy, apart from being due to

the large size of units, is brought about by a number of constructional features which greatly increase the mechanical efficiency of these engines.

Mr. Hawkins then goes on to say: "The above applies to full-load operation. At fractional load there will be a falling off in economy of the Diesel engine in the same way that there is for steam turbine."

This should be qualified by saying that there is a falling off in "the same way," *but not in the same amount*, as the overall efficiency of the steam turbine decreases at a greater rate at fractional loads than that of the Diesel engine.

One of the principal characteristics of the Diesel engine, to which it owes its attractiveness as a prime mover, is its "flat" fuel consumption curve at fractional loads. Thus between full and three-quarters load there is no appreciable difference in its fuel consumption. In modern high-grade engines this increase at three-fourths load rarely exceeds 1 to 2 per cent. and at half load is but 10 per cent. greater than at full load.

This phenomenal efficiency is brought about by an increase in the indicated thermal efficiency of the engine at fractional loads, which counteracts to a certain extent the loss in mechanical efficiency due to an increase in the internal work of the engine and the air compressor at fractional loads.

The economy of the steam turbine is only one link in the chain of the different factors affecting the overall steam-plant economy. The boiler-plant efficiency, the maintenance of a high vacuum, greatly dependent on an ample supply of cold circulating water for condensing purposes, and the efficient operation of the auxiliary and condenser equipment all affect the steam-plant economy. It is, therefore, a well-established fact that the continuous operating economy of a steam plant differs materially from records secured during performance tests, and that the percentage increase in fuel consumption at fractional loads is greater than that of Diesel engines. As the economy of the Diesel engine is two and a half to three times as good as that of the most economical steam plant at full load, a greater percentage increase in fuel consumption at fractional loads is, of course, more serious than the mere numerical figures indicate.

The conditions leading to the selection of the type of plant considered are fully enumerated on page 1175 of my article and need not be repeated here. The engines will deliver 1 kw.-hr. with a fuel consumption of 0.64 at full and three-fourths load. At an overload of 10 per cent. there is also no material increase in fuel consumption, to change this figure. The Diesel-engine plant could therefore have swings of 1,500 kw. to 2,100 kw. from three-fourths load to 10 per cent. overload and the fuel consumption stated in my article would still hold good. Such engines can be started at a moment's notice, which is not feasible with steam turbines, unless they have previously been heated, which has to be done gradually, particularly where highly superheated steam is used.

Such load variations, however, occur but seldom in a plant supplying current to a mine, mill, and metallurgical works; the load is fairly uniform, with a high load factor. If peaks resulting from hoisting operations are high, it will prove economical and increase the life of the power equipment, to absorb such peaks, either by the use of a flywheel generating set, by using compressed-air storage and air-operated hoists, or by following the plan outlined in paragraph 6, page 1177 of my paper.

Gravity is no criterion of the availability of a fuel for Diesel-engine operation. A light fuel oil may be very undesirable, whereas a heavy fuel oil, even 12° Bé., may prove to be highly successful. The composition of an oil will be the deciding factor determining the usefulness of a fuel oil for Diesel engines, and gravity sheds little light on that.

Mr. Hawkins argues that "there is of necessity always a falling down in actual operation from the test results," and that the fuel consumption is therefore higher than that obtained on acceptance tests. The contrary is the case. After Diesel engines have operated for some time, their fuel consumption is lowered, due to an improvement of their mechanical efficiency with the greater smoothness of motion in all parts. Nor would "any slight wear of cylinder reduce the economy very materially due to the enormous leakage possible," as such a state of affairs would bring the engine automatically to a stop. "Enormous leakage" would fail to secure the high compression required to heat the air sufficiently for the ignition of the fuel, and would have to be corrected at once by the engine operatives.

Maintenance and Lubrication

From Table 1 it will be noted that two machinists with yearly wages of \$3,360 form part of the regular operating force of the Diesel plant, to maintain engines at highest efficiency. This is directly chargeable to maintenance. Besides, it is usual for the engineers on shift to assist in any engine cleaning or repairs. As the 1 per cent. allowed for maintenance in Table 1 is based on the cost of the plant in America, this will take care of the higher price level here. Operating data in my possession, of plants in America and Europe, make the above allowance a fair average. It is obviously impossible to allow for all contingencies. An element of good or bad luck is a factor in the repair cost of any power plant. This element of risk is possibly best taken care of by insurance underwriters.

Steam Plant

The reasons for selecting 6,000-kw. units are plainly stated in paragraphs 1 to 3, page 1175 of my paper. Paragraph 2, page 1175, reads:

"The selection of two 6,000-kw. rather than three 3,000-kw. steam turbines is made on account of the greater efficiency of a 6,000-kw. over two 3,000-kw. units.

near its full-load capacity, a number of small units is justified; such diversified load conditions, however, apply mainly to central stations in cities, rather than mine, mill, and smelter power plants, which have invariably a high load factor during the entire day."

The above should suffice in answer to Mr. Hawkins' criticism regarding the selection of units.

Fig. 3, in which the factors influencing the selection of a Diesel-engine or steam-turbine power plant for a given plant cost, fuel price, and load factor are graphically shown, has no relation whatever to Fig. 1 and Tables 1 and 2.

The latter apply to small and medium-sized power plants up to 6,000-kw. capacity, are based on actual operating economy and not on performance tests. Overall efficiencies of 12 to 16 per cent. for steam plants of that size are the exception rather than the rule.

In conclusion, it must be said that neither maintenance nor lubrication of the modern high-grade Diesel plant are "excessively high," nor did I say that the lubrication expense may vary "from \$2 to \$4 per kilowatt-hour." I said that lubrication may cost from \$2 to \$4 per kilowatt-year (8,760 kw.-hr.). Better results than the lower figure are secured in high-grade engines that use forced feed lubrication, and filter and cool the oil before reusing it. While the lubricating-oil consumption used to be very high in Diesel-engine plants, improved construction and oiling systems have lowered that very materially, so that less than 1 gal. per 2,400 hp.-hr. suffices in good engines with proper attendance.

Maintenance may vary from \$1.25 to \$5 per kilowatt-year, depending on a variety of conditions, such as size of plant, load factor, make of engine, and very small plants may even have a higher maintenance charge. The deciding factors in the selection of either type of prime mover will be the plant cost, fuel price and load factor. Water conditions may also influence the decision.

Mr. Freyn's statement quoted by Mr. Hawkins reflected the status of the Diesel engine in the United States more than 5 yr. ago, when the building in America of the high-grade European Diesel engine had not even commenced. Since then conditions have very materially changed and a number of American manufacturers under license from European builders build high-grade engines. These are being introduced in increasing numbers at present, as their economy is well established and a matter of record for many years.

The Diastrophic Theory

Author's reply to discussion* of the paper of MARCEL R. DALY, presented at the Arizona Meeting, September, 1916, and printed in *Bulletin* No. 115, July, 1916, pp. 1137 to 1157.

MARCEL R. DALY (communication to the Secretary†).—The fundamental disagreement between Eugene Coste and the writer proceeds essentially from the contradictory views they hold on the origin of petroleum. This difference cannot be bridged, nor can inorganic and organic theories be discussed here. The writer has suggested that the proper thing in such a case would be to agree to disagree.

Coming to the detailed objections further presented by Mr. Coste, the writer feels that, in the main, they take their source in a wrong interpretation of the theory, and the writer is sometimes too generously invested with views that are not his own.

1. The only requirement for the theory is that the oil proceeding from organic remains be distributed in a water-laden sediment, and that this sediment be submitted to the increasing compression due to the continuous accumulation of superimposed strata, which would tend to squeeze out the liquid contents (p. 1152). This squeezing out would be further aided by the distortion of the strata (p. 1148, a). Nothing here makes impossible the slow formation of the oil, if this is considered necessary, as the piling up of the strata in the syncline would require by itself a protracted length of time, and, further, the squeezing-out process is a slow one. Possibly the formation of the oil may even continue during part of the process of distortion. But the mode of formation of the oil and the time and temperature required for it are left to the chemist to decide. The diastrophic theory, which is essentially a theory of motion and accumulation dependent on the laws of mechanics, is not directly concerned with them, provided the general points heretofore outlined be agreed upon. Perhaps it would be wise not to claim too many "eons" for the formation of the hydrocarbons, if the statement made by Sir Boverton Redwood is correct that "the coral reefs of the Red Sea are charged in places with recent petroleum, formed in that torrid climate from the swarming organisms occupying the shallow pools."¹

2. There is nothing mysterious in the tendency of the fluids, squeezed out from their original and heavily compressed layers, to congregate in some more porous or less compressed rock. This is a simple mechanical effect, from which there is no escape. A mechanical system

* *Bulletin* No. 120 (December, 1916), 2204.

† Received Feb. 21, 1917.

¹ *Petroleum* (1913), 1, 133-134.

reaches a state of equilibrium only when its potential energy reaches a minimum. It is for this same reason that water runs down hill, collects in bottoms, and does not run up hill again when so collected. Further, the squeezing out of the oil from the clayey layers becomes possible only by the deformation of the oil globules when compressed between the very fine particles of the clay; whereas in sandy layers this deformation would not take place to the same extent, on account of the greater volume of the grains and the corresponding enlargement of pore spaces. Here, the oil globules would return to their spherical form, by reason of surface tension, and would become trapped between the grains. A certain amount of the water contained in the sand layers may be replaced by the liquid containing the hydrocarbons. This portion would escape either through the overlying strata or laterally; water, on account of its lower viscosity and higher capillarity, being able to filter through where oil could not.

The theory has never claimed that every sand would contain oil, nor that every mud is the parent matter from which oil proceeds.

3. I think that most of the objections contained in this paragraph would have been omitted if the critic had followed more closely the description of the process as stated by the writer. For instance, when it comes to oil migration, why does Mr. Coste hold the writer responsible for a transfer of the oil through "miles and miles" to "far-away oil and gas pools" under conditions admittedly adverse to motion, when the writer has specifically declared that "the extent of the movement would be variable in the extreme according to local conditions of the strata" (p. 1153). In the same way, the writer has never claimed any state of incompressibility for the strata under consideration, and, above all, any general and absolute incompressibility for any stratum. Compressibility is essentially relative, and the whole theory is based on the production of zones of unequal compression. The writer has simply said that a layer of sand may become incompressible when it still contains a great percentage of holes and may thus act as a reservoir even under considerable pressure, which is a fact (p. 1145). Further, Mr. Coste seems to have lost sight in his discussion of a well-known mechanical principle, which may be expressed as follows: When a material body is distorted by the application of two external forces, different in magnitude and direction, if one of the forces enters first alone into action and then both forces act together, the strains resulting from the application of the first force will be modified by the intervention of the second. This may be termed a superposition of effects. In such a case, an area which is compressed under the action of the first force may become decompressed and even stretched when the second enters into play. An illustration of this is a vertical column, first subjected to a load which compresses it, and later to a horizontal thrust which would bend and break it. In the same way, strata may be compressed vertically at a first stage, and later

submitted to a horizontal thrust which would throw them into folds, changing entirely the distribution and the nature of the strains, and even opening fissures at the summit of the arches, if the superincumbent load is not large enough. If there is any fallacy in this case, it seems to belong to the commentator and not to the author of the theory.

The last sentence of paragraph 3 in Mr. Coste's discussion raises an interesting question. As the same question is asked by Mr. Park, both will be answered together.

The writer concedes readily the difference between the original anticlinal theory and the structural theory propounded by F. G. Clapp, in which "the anticlinal theory is only one of the factors in the accumulation of oil and gas pools," which may become even "unimportant;" and he wishes it to be understood that his objections to the "anticlinal or structural theory" are exclusively limited to the "anticlinal" part of it.

The interpretation of the anticlinal theory itself given by the writer is not a personal conception, but an abstract of the interpretation of this theory given by W. T. Griswold,² in *U. S. Geological Survey Bulletin* 318, where the movement of oil in a porous stratum, completely saturated with water, is explicitly ascribed to buoyancy.³ This conception has been severely criticised since by Malcolm J. Munn,⁴ who seems to imply that there is yet something to disagree about in this direction. Although the writer is aware that this explanation of the motion of oil is losing ground every day, it is no less a fact that its influence is still felt in many textbooks and papers of recent date. Expressions like these are frequently to be encountered: "If the rocks are thoroughly saturated, there will be a general migration of the globules of oil upward through the strata."⁵ Or: "Petroleum . . . persistently climbs the slope of inclined water-containing strata."⁶ Or again: "It is to be expected that in saturated rocks, petroleum, a liquid lighter than water, will rise to the highest parts

² W. T. Griswold and M. J. Munn: *Geology of Oil and Gas Fields in Steubenville, Burgettstown and Claysville Quadrangles, Ohio, West Virginia and Pennsylvania. U. S. Geological Survey Bulletin* 318 (1907).

³ W. T. Griswold writes as follows: "Oil and gas entering a porous rock that is completely saturated with water, will be forced up to the top of the porous stratum by the difference of the specific gravity of the hydrocarbons and the water. Here the oil and gas will remain if the porous stratum be perfectly level; but if it has a dip sufficient to overcome the friction, the particles of oil and gas will gradually move up this slope, the gas with its lower specific gravity occupying the higher places" (p. 14).

⁴ M. J. Munn: *Anticlinal and Hydraulic Theories of Oil and Gas Accumulation. Economic Geology* (1909), 4, 509; and more recently *U. S. Geological Survey Bulletin* 547, (1914), Reconnaissance of the Grandfield District, Oklahoma.

⁵ C. T. Lupton: *Oil and Gas in the Western Part of the Olympic Peninsula, Washington. U. S. Geological Survey Bulletin* 581-B (1914), 79.

⁶ K. C. Heald: *Oil and Gas Geology of the Foraker Quadrangle, Osage County, Oklahoma. U. S. Geological Survey Bulletin* 641 (1916), 41-42.

They evidently imply a motion of the oil due to buoyancy, so that the old interpretation seems to remain in force at least in certain quarters.

It is the writer's opinion that the buoyancy of the oil cannot be interpreted as a direct source of motion of the oil on any reduced scale more than it can be on any large one. *Buoyancy simply does not move the oil.* It does not move the oil more than the weight of a stone moves the stone when transported by a current of water. It may *deflect* the motion imparted to the oil by hydraulic action, and, in this respect, its importance is enormous; but it stops there.

This deflective action would be adequate to explain mechanically such phenomena as the segregation of the oil globules during the process of migration, and the accumulation of the oil in places where the velocity of the current would be reduced, such as in convex arches along the dip or the strike of the sand (see latter paragraph), etc. But buoyancy without a current would not bring the result.

The writer is also of the opinion that, barring exceptional conditions, the velocity of ground waters would not be sufficient to overcome the resistances offered to the motion of the oil (p. 1141). So that the motive force would have to be looked for elsewhere, and diastrophism seems to remain the only possible source to which we may apply.

Capillary action may promote the concentration of oil and gas in the sands, as explained by C. W. Washburne⁷; but the writer is not prepared to admit that it is the sole agent of such a concentration, and still less that it is the controlling factor of migration and accumulation of the hydrocarbons in pools. Its effects are related to the lithological character of the rocks and not to their structure; whereas everything points to an intimate relationship between oil or gas accumulation and structure. Further, capillary action would not account for rock pressure.

Coming back to the structural theory, the practical value of which the writer fully recognizes, we may ask: Is it a *theory* in the proper sense of the word, or is it not rather a statement of facts and conditions and a method for working out these statements in a practical way? A theory ought to be the philosophical explanation of the ascertained phenomena, thereby linking these phenomena together by some general principle. Where is the general principle to which the motion of oil and gas through the strata, their concentration and final accumulation in pools may be ascribed? It is precisely this "missing link" that the writer has endeavored to define. Whether his hypothesis, which is essentially a mechanical one, may be accepted, rejected, modified or perfected, remains an open question, and the criticism of men of the experience and knowledge of Mr. Clapp may do much to bring a final conclusion. In

⁷ *Geological Survey of Canada, Memoir 88* (1916), 169.

⁸ *Trans.* (1914), 50, 829.

the mind of the writer, the merit of the diastrophic theory—if any—does not consist so much in the theory itself, but in the direction of research it opens to future investigators. The writer feels that the importance of mechanics in the interpretation of geologic phenomena cannot be overestimated. After all, everything in Nature obeys the laws of mechanics, and most geologic phenomena are nothing else than the expression of these laws.

The practical usefulness of any general theory that would definitely establish the laws that govern the motion and accumulation of the hydrocarbons is obvious. Today, using Mr. Clapp's expression, "successful oil location depends preëminently on inference." But an inference is a hypothetical induction built on the comparison of ascertained facts in every specific instance. Undoubtedly, an ascertained law would be a time and money saver in the field; when the true causes of a phenomenon are fully understood, it becomes less difficult to anticipate the conditions under which it may be repeated and to determine on the ground the places where such conditions are to be found.

Coming to the questions asked by Mr. Clapp, and numbered 1 to 5:

Questions 1, 2 and 3 may be answered together, as the facts stated would be derived, under the diastrophic theory, from the same cause. Without going into details, the writer wishes to say that he has been brought to foresee the following general law: "Whenever an oil-bearing region is folded by a simple dominant thrust or repeated thrusts in the same direction (Appalachians, Carpathians), the fluids inclosed in the strata, and which are put in motion, will have a tendency to collect ahead of the line of application of the thrust, and to segregate in successive deposits of increasing density, or decreasing buoyancy, with the increase of distance from this line." This implies that gas would tend to accumulate nearest to the line of thrust; oil, farther from it; and that water may reach still farther away.

The first part of this law would result from the application of a principle which has been called upon several times in this paper; *viz.*, when a mechanical system is in equilibrium, its potential energy is a minimum. So that fluids will have a tendency to migrate from the zones of higher compression—or of higher potential—toward the zones of lower compression—or of lower potential—*viz.*, away from the source of pressure.

The second part of the law would be a consequence of the same principles which regulate the deposition of bodies heavier than water in a current of decreasing velocity.

Let us consider the simple case of a horizontal current of water, in an inclosed channel, provided with a steady motion and a progressively decreasing velocity from *A* to *B* (Fig. 1); the decrease of velocity being due to a progressive increase of the width of the channel. Let us imagine that several bodies heavier than water, of varying densities but individu-

ally of the same shape and volume, are at first collected in *A*. The motion of the water will tend to carry them downstream, toward *B*. The transporting power of the water will depend on its kinetic energy ($\frac{1}{2}mv^2$), or its capacity to do work; and as same volumes and shapes are considered, and as the mass of the water per unit volume remains constant, this energy will vary only with the velocity of the current.⁹ If the velocity increases, so will the transporting power; and, conversely, a reduction of velocity will mean a decrease of the power of transportation. Now, the specific gravity of the materials has a marked effect upon the mean velocity necessary to move them. Bodies heavier than water, like stones, would move along the bottom of the channel by rolling, sliding, or more generally by a succession of leaps; *viz.*, rising from the bottom and returning to it after having described a curve in the water. It is essentially gravity which causes the resistance through which the



FIG. 1.

kinetic energy imparted to the materials by the water is thus spent in friction or collisions. So the greater the gravity, the greater the requisite mean velocity of the current.¹⁰ The result is that to each material there corresponds a minimum velocity under which transportation is no more possible, and that each material will be dropped at the point where the corresponding minimum is reached. A mechanical segregation will

⁹ Du Buat has shown that when a body is at rest in a moving fluid, if the motion is steady and the flow takes place in a pipe, the pressure *R* exerted by the water against the body may be expressed by $R = M \frac{mv^2}{2} S$, in which *v* is the undisturbed velocity of the stream, *m* the mass of the water per unit volume, *S* the projected area of the body on a plane normal to the direction of flow, and *M* a coefficient depending on the shape of the body and its relative proportions with the transverse section of the pipe (M. Bresse, *Cours de Mécanique Appliquée* 2e P.-Hydraulique, p. 429). According to Lord Rayleigh's theory (*Philosophical Magazine*, December, 1876), if a plate with parallel edges be held in a stream normally to the direction of flow, the mean unbalanced pressure on the body is $= 0.88 \frac{mv^2}{2} S$, where the letters have the same signification as above. Both formulæ show that the power of transportation of a current of water is, at the same time, a function of the kinetic energy of the water ($\frac{1}{2}mv^2$) and of the shape and volume of the body transported (*S*). If these two last factors are supposed to be the same for the different bodies considered, as *m* is a constant, the intensity of the force of motion will simply vary, in each case, with the square of the velocity of the current.

¹⁰ Chailly gives the following formula for finding the velocity required to move rounded stones or shingle: $v = 5.67\sqrt{a \cdot g}$ where *v* is the velocity of water in feet per second; *a* the average diameter in feet of the body to be moved; and *g* its specific gravity. The velocities required to move such bodies of the same shape and volume, but of varying densities, would thus vary as the root of their specific gravities.

ensue, whereby the respective distances of transportation, from the point of origin to the point of deposition, will increase as the specific gravity of the body decreases.

Let us repeat the same process with bodies of varying shape and volume, as well as of varying gravity. For each series of bodies of same gravity, the increase of volume will cause a corresponding increase of resistance to motion;¹¹ and the mean velocity required for the motion will increase with the volume. The bodies will thus be deposited at distances increasing as their volume decreases. There will be, not a *point* of deposition for the bodies of like gravity, but a more or less extended *zone*; and the zones of deposition of the series of different gravities may encroach on each other.

Experience agrees with this interpretation. For instance, the principle is used in ordinary gold washing, or for the assortment of sands of like size but varying specific gravity, etc. Worded as it is, this interpretation applies only to bodies heavier than water; but it may be shown that, with a simple modification of terms, it may be extended to bodies lighter than water; or, in other words, that the same law regulates the order of deposition of materials of any gravity in a steady current of decreasing velocity flowing through an inclosed channel.

We may remark that the action of gravity which creates a resistance against motion is not really represented by the specific gravity of the material (ratio of its weight to that of an equal volume of water), but by the difference between this specific gravity and the specific gravity of the water; for a body immersed in a fluid suffers a loss of weight which is equal to the weight of fluid it has displaced. If δ be the specific gravity of the body, and if the specific gravity of water is taken as a unit, the resistance due to gravity is a function of $(\delta - 1)$ and not of δ . In the C.G.S. system $(\delta - 1)$ would express a density. For lack of a better expression we may term it the "density in water" of the body. Now, this particular density may have a positive or a negative value, depending upon whether δ is greater or smaller than unity. If positive, the action of gravity will be directed down; this is the case of bodies heavier than water. If negative, the direction will be up, and the "density in water" becomes the buoyancy of the body per unit volume. In either case, the resistance to motion may be explained along the same lines. But the

¹¹ Du Buat's formula shows that the pressure exerted by the moving water against a body at rest and immersed in the current, is proportional to the projected area of the body on a plane normal to the direction of flow. For a spherical body, the projected area would vary as the square of the radius of the sphere and the weight as its third power. Weight, hence resistance to motion, in the case of bodies heavier than water, would increase more rapidly than the projected area, hence than the motive force. As solid bodies transported by a current are more or less rounded after a short time, the same relation is approximately true for them, if we consider their mean radius.

may be remarked further that the absolute value of $(\delta - 1)$, when $\delta < 1$, decreases when δ increases; or, in other terms, that the numerical value of buoyancy decreases when the density of the body, that is lighter than water, increases; and the buoyancy of a body having the same density as water would be equal to zero. A general law may then be expressed, which would be applicable to all materials, whatever their gravity: "When a flow of water takes place in an inclosed horizontal channel, with a steady motion and a decreasing velocity from one end to the other, material bodies of varying specific gravity, shape and volume, will have a tendency to segregate and to be deposited at distances from the point of origin which will increase as their respective 'densities in water' ($\delta - 1$) and their respective volumes decrease."

Applying this law to oil and gas transported in a current of water under the conditions stated above, we find that: (a) oil would have a tendency to segregate from the gas; (b) it would have a tendency to be transported and deposited farther than the gas from the point of origin.

But there are several factors that have not been taken into account in the preceding simplified theoretical instance. The chief ones may be enumerated as follows: (1) The motion will not take place in an inclosed channel, but through many irregular and capillary channels, and the resistances to motion will vary, not only according to density, shape and volume, but also according to the individual nature of the bodies transported (viscosity, surface tension, etc.) and the local texture and porosity of the "sand." (2) The strata through which the flow takes place will not be horizontal, but flexured. (3) The materials would not proceed, as a whole, from the same point of origin, but would be first in a state of irregular dispersion over a more or less wide area, from which they would have to be collected by the current. (4) Some chemical or physical changes may occur: gas may evolve from the oil during the process of migration through the sand, as well as in the final area of deposition (reservoir), whether it be dissolved gas released by the reduction of pressure, or gas produced by chemical action (?).

It is not possible to discuss here in detail the effects that such a set of conditions would entail; and the lack of experimental data as well as the complexity of the problem does not allow a definitive conclusion. But a careful investigation of the nature and probable influence of the intervening factors has led the writer to believe that the tendency toward segregation of the materials (oil and gas), as well as to their relative order of deposition, would still obtain. Only, for each material the zone of accumulation instead of being unique, may be repeated several times in succession; for the velocity of the transporting water would pass through a series of maxima and minima, and the minimum

velocity required for the deposition of this material may be encountered more than once. Further, these zones of accumulation, instead of being limited to reduced areas, would become zones of more or less great extent, and the oil zones and gas zones would sometimes encroach on each other. The water zone—the water being in large excess—would extend all through, with or without local interruptions.

In Pennsylvania, at least for oil and gas, the rule holds good (question 1). As Mr. Clapp remarks, "natural gas is found east of Pittsburgh in abundance, while oil is generally absent." The thrust would have proceeded here from the southeast, and the first pools encountered, when we start from the line of application of the thrust and follow in the direction of the movement of deformation toward the northwest, are gas pools. The oil pools are mostly farther west and northwest; Pittsburgh happening to be nearly on the dividing line. This distribution is exactly what should be expected from the law. It is further interesting to note that the gas and oil regions (zones of deposition) encroach sometimes on each other, as may be seen by the inspection of the map, and this too would agree with the conclusions of the writer.¹²

In the fields of central Europe (Galicia, etc.) (question 2) where at least two consecutive thrusts in the same direction have taken place, the results are more obscure. But, here again, it seems as if oil would be ahead of gas. Transylvania occupies the central portion of the sector of which the Carpathians constitute the arc, and gas is found in this section at many points of the basin drained by the Maros River and its tributaries east to the Hargitta range, and by the Szamos river northward, while oil is mostly found farther north, east or southeast, in the outlying region. The general disposition seems to follow here closely the disposition of the fields of Pennsylvania, and probably for the same reasons.

As far as salt water is concerned (question 3) the writer has shown that the water should reach farther away than the transported materials (oil and gas). This is evident from the fact that these materials would be dropped from the current when the minimum velocity which would allow their motion, was reached. But water itself would continue to move until its own speed was reduced to nil. So that it becomes possible to imagine the formation of "pools of water," ahead of the region of oil and gas, and farther from the line of application of the thrust. Originally, the pressure of the water in such pools would have been due to diastrophic pressure, less the loss of head encountered. But the "keeping up" of such a pressure on water alone would be very questionable on mechanical grounds. By reason of its reduced amount of compressi-

¹² M. L. Fuller: The Gaines Oil Field of Northern Pennsylvania, 22nd *Annual Report of the U. S. Geological Survey* (1902), Plate XXXVI, Map of oil and gas fields of Pennsylvania, opposite p. 579.

bility,¹³ water is a bad accumulator of pressure, unless it is held in an absolutely tight container, like the cylinder of a hydraulic press. But terranes are not so tight, and such a pressure would soon disappear. If then, pressure is ever found to exist in a "water pool," ahead of the oil and gas regions, and if such a pressure can not be traced to static pressure due to artesian action, it must indicate the presence of some oil or gas reservoir with which the water would be in communication, and that would act as an accumulator for the pressure. It would not be necessary, in such a case, for the accumulator to be located in the immediate neighborhood of the water pool, as pressure may be transmitted by water

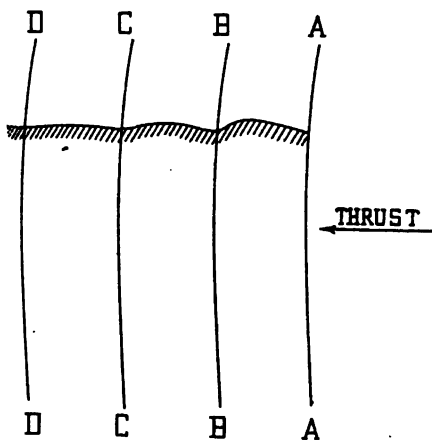


FIG. 2.

itself through underground channels, the same as it would be through a pipe from an ordinary compressor.

Questions 4 and 5.—In order to answer these, it will be useful to enter into some more details of the mechanical process of deformation and of the migration of the fluids as interpreted by the writer. This part has only been sketched in the preceding paper.

Let us imagine a prism composed of a series of horizontal strata, submitted to a lateral horizontal thrust, acting in the direction of the arrow (Fig. 2). For the sake of simplicity, we will suppose, at first, that the strata are continuous, individually homogeneous and of equal thickness all through, and that the superincumbent load is distributed in a uniform way; further, we will consider only this part of the prism that is situated above its theoretical neutral plane. We have seen (*Bulletin* No. 115 (July, 1916), 1147) that, under such conditions, the prism will be

¹³ Regnault, Grasset and Cailletet give about 50 millionth for the compressibility of water at 0° C., for one atmosphere of pressure.

deformed, if the thrust is of sufficient magnitude; that the deformation will begin first along the line *AA* of application of the force, and, by and by, will reach farther away from it, giving rise to successive waves or longitudinal bends (anticlines and synclines) normal to the force, more or less parallel to *AA*, and decreasing in importance the farther we reach in the direction of the arrow; and, finally, that the rate of the compression to which the different points of the prism are submitted, will follow a similar law, decreasing in a general way as their distance from *AA* increases, but presenting a wavy succession of maxima and minima, the maxima corresponding to the synclines and the minima to the anticlines. The whole prism may be divided into zones, individually elongated in a transversal direction to the prism and parallel to *AA*, in which the amount of compression and of deformation obtained are greater the nearer the zone stands to the line *AA*.

The first consequence is that the fluids inclosed in the strata would be put in motion *as soon* as the deformation itself begins, and that they would move from the areas of higher pressure toward the areas of lower pressure, *i.e.*, away from the thrust. But a time may arrive at which the elasticity of the compressed material of the strata in the first zone *AABB* will be overcome; the material, unable to satisfy any longer the accumulated stresses by simple bending, will give way, and ruptures or crushings will occur. These disruptions will not take place at the same time at every point, as the active forces and the resistances would never be equally distributed everywhere; but the material will give way in succession at points irregularly distributed along this zone, and these points will become as many *centers of impact*, transmitting a sudden pressure to the inclosed liquids. This will be renewed again and again, until compression has reached its maximum effect, for the time being, in the zone *AABB*, *viz.*, until the accumulated stresses have been everywhere satisfied by deformation in this zone, and a new temporary equilibrium obtained. From this moment, the principal effect of the deformation will be transferred to the next zone *BBCC*, where the same succession of phenomena may take place; then to *CCDD*, etc. *So that while AA remains, throughout, the line of application of the thrust, the zone of active deformation, or the zone of the centers of pressure, will progressively move away from it in the direction of the thrust, and this displacement may continue as long and as far as the thrust itself continues to operate. It may thus reach considerable distances, with an ever decreasing effect, until it dies out.*

The same process will be repeated many times. Stresses will constantly accumulate in the zone *AABB* and new disruptions will follow, that will spread out, from place to place, or from zone to zone, away from the thrust, until they reach the limit of the deformed area. The region will thus be gone over repeatedly by *waves of deformation* that will

follow each other in succession and gradually decrease with the distance, as ripples do in a still water where stones are dropped at intervals from the same point.¹⁴

It may be stated here immediately that this process would not imply, as a necessary consequence, a migration of equal importance for the fluids transported (oil and gas). On the contrary, we will see that migration may sometimes be very limited in extent. But the compressed region as a whole would be, under the simple conditions admitted, progressively combed of its transportable material, the combing process following the direction of the movement.

Let us now return to the moment of our description where the compressed rock material is giving way along the zone *AABB*. We have seen

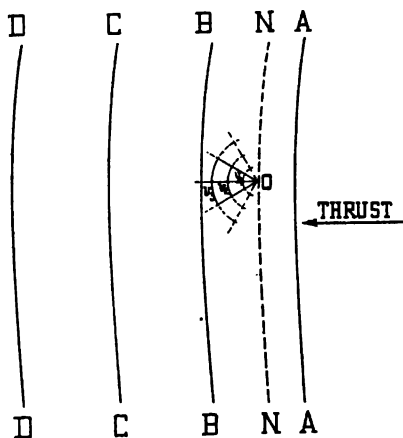


FIG. 3.

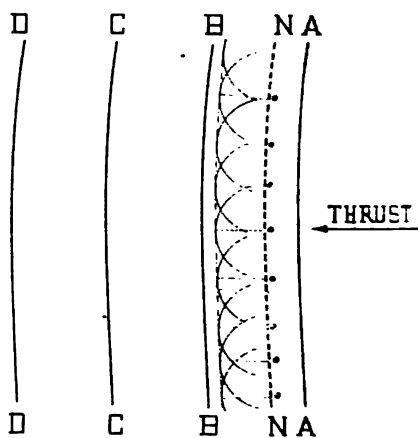


FIG. 4.

that disruption would take place on different points of the zone in irregular succession, creating successive centers of impact. Let us consider one of these centers, *O*, which may be situated on any line *NN* parallel to the line *AA* of application of the thrust (Fig. 3). The fluids inclosed in a given stratum, compressed at the point *O*, will be pushed forward in the direction of the arrow. But this movement will not take place simply in a straight line, normal to *NN*; it will be fan-shaped, and the surface covered by the liquids in motion will take the form of a sector, whose center will be in *O*. The cross-section of the stream will thus increase progressively in the direction of flow, and if the resistance to motion is uniform, as in our simplified case, the velocity of the liquid will progress-

¹⁴ The resulting motion imparted to the liquids would thus become essentially discontinuous and the transportation of the material from one point to another would be composed of a succession of independent stages. But during each of these stages, the motion of the current may be considered as steady or as constituted of a succession of steady motions, with a sufficient degree of approximation.

ively decrease away from the thrust. The points of equal mean velocity $v_1, v_2, v_3, \dots$ written in a decreasing order, will be distributed along concentric arcs of circles, of successively increasing radii. If now the same process is followed for every point of the line NN , and the points of equal mean velocity $v_1, v_2, v_3, \dots$ are marked on the normals to this line, these points will be found to be located on the envelopes of the corresponding circles, *i.e.*, on lines parallel to AA (Fig. 4). And the mean velocity of the sheet of liquid displaced, when considered in its entirety, will be found to decrease progressively along the direction of flow, *although the motion may not take place everywhere at the same time*. This will be true whatever the position of the line NN in the zone. So that every particle of the liquid contained in the zone and which is put in motion, will successively reach the velocities $v_1, v_2, v_3, \dots$ along more or less narrow strips parallel to AA . If then, among these velocities should be found the minimum velocity beyond which the transportation of a given material ceases to be possible, this material would be deposited along the corresponding strip. In other words, the general trend of the deposits would be parallel to the line of application of the thrust.

Precisely there is a factor which determines such a reduction of velocity and fixes the location of the strips of deposition; this is the fac-

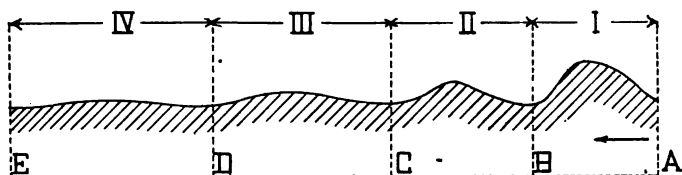


FIG. 5.

tor of structural deformation. If, instead of considering the single zone $AABB$, we would consider a series of such zones in succession, they would present in section the appearance of Fig. 5. The synclines being compressed more than the anticlines, each syncline individually becomes a center of pressure, whose importance decreases away from the thrust; and these centers will act as relays in the movement of the liquid. Between two relays, the velocity of the liquid would decrease when the stream passes over the arch; for the reason that the raising of the liquid from a lower level to a higher one and the sudden increase of the section or of the number of the channels of flow, when entering from a less open to a more open portion of the stratum, would cause a loss of head and an increase of the total section of flow, hence a reduction of speed. When the arch had been passed, the velocity would again tend to increase when the liquid enters the following syncline; but this new rate of velocity would be inferior to the one obtained in the preceding syncline, by reason of the loss of head encountered and the progressive reduction of the

compression away from the thrust. So that the velocity of the current would diminish away from the thrust, but would at the same time present a succession of ever-decreasing maxima and minima, the maxima corresponding to the synclines and the minima to the anticlines or arches.

Now, with the progressive increase of compression and consequent increase of deformation, an anticline would progressively increase in height, and the difference of porosity between the summit of this anticline and the bottom of the preceding syncline would increase also.¹⁵ So that the loss of head itself would tend to increase and the velocity of flow to decrease. Thus, the velocity of the flow passing over the arches would become lowered in the course of time, during the process of deformation, and it may reach the velocity-limit corresponding to deposition before this limit is reached on any other point of the line. The summits of the arches become in this way the natural loci of deposition.

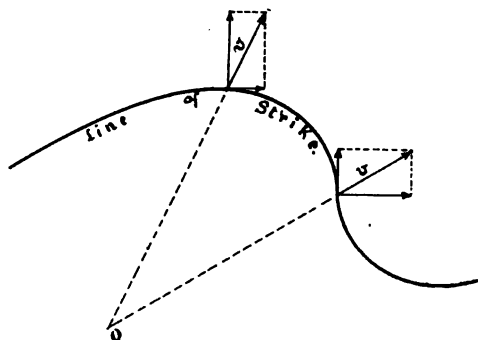


FIG. 6.

As the arches themselves are parallel to the line *AA* of application of the thrust, so will be the final deposits or pools.

The same result would obtain with monoclines and the rims of terraces. These forms mostly represent a stage of incipient flexure farther away from the thrust, especially when the strata are slightly bent downward, as in a basin of deposition. And again, the same process would explain why changes in the direction of the strata along their strike may cause local concentration of the material. We have seen that the flow issuing from a local center of pressure would be diverging in direction (fan-shaped). When a line of strike is encountered, any velocity which is not normal to it may be decomposed into two components: one normal to the strike, and the other tangent to it. The first would move the liquid along the dip, in a vertical plane; the second would move it along the

¹⁵ The anticlines considered here are the anticlines of the Appalachian type. Forms exaggerated by compression would give a different result. But it must be remembered that the case under treatment is the simplest possible one.

strike, in a horizontal one (Fig. 6). And the horizontal incurvation of the strike may bring about local reductions of velocity (conflicting currents, etc.), and consequent areas of deposition.

It is evident that the extent of the migration of any transported material (oil or gas) under the conditions described, would depend primarily on the distances at which two successive zones, presenting the velocity-limit necessary for the deposition of this particular material, may be encountered. Between two such zones, the territory would be combed of the material—if any; but such material need not go farther than the second zone. *Transportation and concentration, or migration, becomes, under this conception, an essentially local phenomenon* in the affected area, which may take place independently on different points of the field, either at the same time or at another. This, in contradistinction with compression and deformation, which are phenomena of a general order, that would follow the laws of continuity throughout the field.

In developing the preceding theory, the writer has had in view the simplest possible conditions, and it may be objected that such conditions are nowhere to be encountered at the same time on any extended area. This is readily conceded. The number, direction, relative importance, and age of the successive deformative movements to which a region has been submitted, may vary widely, inducing consequent irregularities in the distribution, extent, nature and physical conditions of the layers. Lateral variation will be the rule in any stratum, and unconformities may occur between the series. Simple flexion may be complicated by the sliding of the beds; structural forms exaggerated by compression; faults and other tectonic accidents or even igneous intrusions may occur. Finally, changes may take place subsequent to deformation and accumulation, such as a change in the local level of the waters, etc. But whatever the conditions may be, the whole trend of the observed phenomena will have to be underlain by the same general principles of which the writer has tried to make an application; for these principles are not relative, but partake of the independent reality of the laws of mechanics. Only, the result will be that of a superposition of effects and may be complicated accordingly. Thus, when the geologist would come to locate an oil region, or simply a pool, under the diastrophic theory, he would not be exempted from considering every particle of evidence any more than he is today under Mr. Clapp's structural theory; but he would at least be provided with a few comprehensive laws, which would allow him to anticipate the general trend of things, to seriate the intervening factors according to their probable rank of influence, and finally to reach a conclusion based not only on inference, but on a logical process of reasoning, where deduction would come to the help of induction.

It now becomes possible to answer the further questions proposed by Mr. Clapp.

(Question 4). If the writer interprets Mr. Clapp¹⁶ correctly there are many cases in which oil is found on both sides of a fault crossing a productive field, and the question is: how is oil to be found *on both sides* of the fault, as, for instance, in Oklahoma, where such faults are assumed to be thrust faults.

To this, the following answer may be made. According to the views of the writer, heretofore presented, the movement towards accumulation keeps pace with deformation and accumulation begins as soon as in the future reservoir the velocity of the transporting waters reaches the minimum under which transportation is no more possible. In a simple flexure (anticline or terrace) accumulation may begin and may even be completed long before the limit of deformation has been attained. On the other hand, a fault, especially a thrust fault, is generally one of the last expressions of deformation. When a stratum is compressed tangentially its tendency is to bend before breaking. Thus, compression may first accumulate the hydrocarbons in a reservoir and only later break through this same reservoir with a fault, after the accumulation has been completed. According to the relative position of the fault and of the reservoir, the hydrocarbons would be found either in the front, in the rear, or on both sides of the fault. The last case, where the fault cuts through the reservoir, would naturally be the most frequent, as the location of both the reservoir and the area of maximum local stresses would be nearly coincident. This would appear the simplest and most probable explanation of the facts. In the case under consideration, the oil deposit would antecede the faulting, in the same way as the coal seam antecedes the fault which has displaced it.

(Question 5). We have seen that a distinction must be made between the line of application of the thrust and the zone of active deformation or the zone of the centers of pressure. The former's relative position in the deformed area remains constant during the process of deformation, whereas the second would move away from it progressively, in the direction of the force. Concentration and accumulation may thus take place as far as deformation remains effective, provided the other conditions (existence of oil and gas in a disseminated state, presence of water, porous layers, etc.) are favorable; and the distance attained is limited only by the extent of the area of deformation itself. On the farthest borders of such an area, where the effects of compression would be reduced, the velocity of the transporting waters would be reduced in proportion, and the velocity-limit necessary for deposition may be reached with dips smaller than elsewhere. But it must be remembered that the remoteness of the "mountain belt" or the "major axis" from the point of accumulation, would have no bearing whatever upon the distance covered by the migration of the hydrocarbons. As previously stated, migration is essentially a local phenomenon.

¹⁶ From correspondence with Mr. Clapp.

Answering Mr. Park, it is not the intention of the writer to assume that the oil existed in the original strata where the parent matter was at first deposited (in lagoons, marshes, deltas or at the bottom of the sea) from the moment of the deposition of such parent matter or even soon afterward; and the writer readily admits that the transformation may have occurred not only during the time of accumulation of the sediments in the syncline, but even during part of the movement of deformation itself. But, as already stated, this point is immaterial for the theory, which is interested in the genesis of the oil only in so far as the mode of formation concedes a first state of dissemination of the hydrocarbons in a water-laden sediment, submitted to increasing vertical pressure and subsequent lateral thrust or thrusts.

It frequently happens that, from the start, it is not possible to attack a problem in its more complex form; and this is especially true when we try to submit natural phenomena to mathematical treatment. The number of the factors that would affect the results, their intricacy, and, let us say, our ignorance of their real nature and true mode of action, is such as to put any solution out of reach, unless we consent to reduce the problem to its simpler lines and to consider, at first, some well-defined and better-known specific instance. The writer is aware of the inconveniences and of the danger inherent in such a procedure, but, unfortunately, it is the only one at our disposition. If the results are not always entirely satisfactory, it is no less true that the solution of a specific case may open our eyes to the general direction of things, exactly as the knowledge of a curve between narrow limits may give us a glimpse of the class to which it belongs.

Among the known oil and gas fields, the Appalachian belt seems to the writer to present one of ideal qualities. Here, the structure is relatively simple and has been well worked out; the principal folding has taken place at one defined period and under the influence of a single dominant thrust; and later disturbances have not obscured the results. In this region, the thickness of the strata involved justifies us in considering the oil sands as located above the theoretical neutral plane, which simplifies the treatment, and the gentle anticlines offer a simple feature of distortion. These conditions render an easy analysis of facts and application of mechanical principles. Mechanical laws are absolute, and if they are true in one field, they are equally true in any other. They simply need to be applied with proper data and proper judgment. We concede that sometimes conditions may be confusing, as in the South Coast Range of California, where the folds are often shallow, as Mr. Park remarks, and where, further, two different sets of thrusts, acting from two different directions, may have to be considered. But liquids will always tend to move from zones of higher to zones of lower compression; for every material transported in a current of water, there will be always a velocity-limit under which transportation is no more possible and deposition ensues; distorted

strata will always obey the general laws of distortion; and the mechanical reasons which have brought the writer to consider diastrophism as the principal motive power and water under hydraulic action as the transporting agent, will remain the same. As in an equation, you may have to change the coefficients; the fundamental value of the equation remains.

Let us state here, incidentally, that forms of distortion, like an anticline, may present zones of porosity at one time and not at another. An anticline may be said to begin with the smallest incurvation of a stratum and it may end in a flattened isoclinal fold, overturned or not, faulted or thrust or not. Between the extremes, all kinds of intermediate forms exist, in which the internal structure of the material would not be the same, and the conditions of porosity would vary accordingly. These would even vary for any given anticline during the folding process. What the writer has said about anticlines must be understood to apply to the anticlinal ridges of the Appalachian type, which he had in view, but not to forms exaggerated by compression.

It is true that anticlines of the aforesaid type are important, in so far as the accumulation of oil is concerned, as zones of greater porosity; a container is always needed to hold the contents. But this is not the sole function of an anticline, nor is the anticline the sole possible container. This has been pointed out by the writer.¹⁷ Convex edges or arches along the dip or the strike of the strata, may be good places for accumulation in general, as these would result in a reduction of the velocity; and buoyancy, whose real function seems to appear here, would act as gravity does when a slackening current of water drops its load: only the action would be reversed. Buoyancy would deflect the direction of motion upward and the hydrocarbons would become trapped under the concave or vault-like portion of the strata.

The lithological character of the strata may have much to do in the way of easing or impeding the circulation of the liquids; but an extensive layer with the same lithological character, the same thickness and the same dip throughout, probably would not be a favorable place for accumulation, as it would offer poor chances for an abrupt reduction of motion, which seems to be an essential factor of deposition or accumulation. It is precisely where "folds dominate the structure" that the best conditions of deposition ought to be found.

There is no mechanical reason why rock pressure, as the writer understands it, should be fairly constant over considerable areas in an oil or gas pool; but there are reasons why it should *not* be so. According to the writer's views, rock pressure in oil or gas pools is essentially due to the original pressure of deformation, less the loss of head encountered on the way, plus a certain possible amount of added pressure due to progres-

¹⁷ *Bulletin* No. 115 (July, 1916), 1143, 1153.

sive gas increase.¹⁸ Now, an oil pool or a gas pool cannot be compared to a gas holder or to a reservoir for water under constant pressure. In a gas holder, the gas is at the same pressure everywhere, and in a reservoir under constant pressure, water is at the same pressure on the same horizontal level. For the reason that, in the first case, gas molecules can move with an equal freedom in any direction; and, in the second case, water molecules have an equal freedom of movement in all directions of the same horizontal plane. But a gas pool or an oil pool is formed by the juxtaposition of an indefinite number of cells, intercommunicating in the most irregular way, and equal freedom of movement is not permitted in any direction. Further, the resistance to motion between cells is variable, and the resistances accumulate with the distance between points. So that, even barring the possible additional pressure due to irregular gas increase, the original pressure due to deformation would not have to be the same at all points, as the loss of head would be at variance; and no balance could be reestablished between points, in the course of time, as the resistance to motion between cells would remain the same. The conclusion is that rock pressure, under the diastrophic theory, must vary, as a rule, from one point to another in an oil or gas field, and the theory agrees once more with the observed facts.

Mr. Park, in common with Mr. Coste, remarks that the very factors that the writer calls upon to seal up an oil pool and to maintain a given rock pressure, might equally well be called upon to prevent migration originally. To this it may be answered that conditions in both cases are entirely different. In an oil and gas pool, the rock pressure is essentially static; whereas, during the process of migration, the motive pressure would be dynamic, the transporting agent being water acting under a ram effect. As an illustration in the case, you may consider the following test: try to drive a nail in a board simply by the weight of the hammer bearing on the head of the nail, and then drive the nail by a blow. The difference is apparent at once. The first case is a case of static pressure; the second, that of dynamic action. There are other differences, all in favor of the process of migration: Rock pressure is only a remainder of the pressure to which migration is due; again, the "tubes" of the capping rock which holds the gas would be of a subcapillary size, instead of being capillary as the channels of the sand strata through which the movement toward accumulation takes place, and the resistance offered by the Jamin tubes is in the inverse ratio of their diameter; then again, original oil would be in a state of disseminated particles in a matrix of water, instead of being in the state of a more or less continuous mass, as in an oil pool, etc.

Oil which has moved through the rock pores and has collected under pressure at a given place, would not move back over the path it originally traveled any more than the nail which has been driven in the wood

¹⁸ *Bulletin*, No. 115 (July, 1916), 1143, 1153.

would be thrust out of the board. Static pressure would be unable to undo here what repeated dynamic action had caused to be.

The fact that "in the California fields, where water sands and oil sands are interstratified, no pressure comparable with the pressure in the oil sands is recorded for the water sands which lie between the oil sands, unless a distinct flow of petroleum gas is noted with the water," is easily explained, in accordance with physical and mechanical laws, as a consequence of a common original diastrophic pressure. Let us note first that Mr. Park's remark implies that whenever there is a difference in the pressure between an oil sand and a water sand, there is an intermedial layer between both sands which is impenetrable to this pressure, or else the balance of pressure would be soon reestablished. The conclusion is that the pressure in both sands is independent. Now, let us suppose that, at a certain moment, the same amount of pressure should be imparted to both sands, as would be the case if these layers were submitted together to a common diastrophic pressure; and, for the sake of simplicity, let us consider the temperature as constant. Liquids are compressible only in a very small degree; which means that a very small reduction in volume would correspond to an enormous increase of pressure, and, conversely, that a very small expansion of volume, at constant temperature, would allow a very great reduction of pressure. Whereas gases are essentially compressible, and a great increase of pressure, at constant temperature, would only be reached by a corresponding large decrease of volume; or, conversely, the volume would have to be largely increased to decrease the pressure to any serious extent. Gases are thus good accumulators of pressure, where liquids are not. As oil would always be in relation with some gas, an oil sand would become a good accumulator of pressure also; but a water sand would not, unless accidentally put in direct relation with the gas. On the other hand, strata, especially Tertiary sediments, are never absolutely tight, and a very small expansion of the water body would rapidly reduce the pressure to the normal; whereas oil strata, containing gas, would be in a very much more favorable condition to retain a large part of it.

The fact pointed out by the writer in reference to the increase of rock pressure with the depth—or age—of the strata in the Eastern fields (Appalachian belt), is not contradicted by this other fact that the situation seems to be reversed in some Californian fields, where greater pressures are noted in younger beds resting unconformably on older rocks; as the tectonic conditions would not be the same in both instances. In the first case, the whole pile of sediments has been involved in the same deformative movement, due to a single dominant thrust, and, therefore, the result is simple and follows the law of continuity; whereas, in California, deformation has been due to several thrusts, acting at different periods, in different directions, and it has affected unequally the uncon-

formable series. The mechanical law of the superposition of effects, to which the writer has previously referred, may have largely interfered in this case, and its influence would have to be defined.

It is the writer's view that "fossil pressures," to use the picturesque expression of Mr. Park, are to be considered as the rule in Nature and not the exception. The earth's body is in a constant state of readjustment, as well shown by seismic and kindred phenomena; and these phenomena draw their source from the past much more than from the present, as they point to the relief of slowly accumulated strains. When the lower sediments of a syncline are compressed by superincumbent strata, the first compression dates from the time of the first deposition of these strata; the "heaving" in mines, the deformation of rocks in quarries during operations, the production of fissures on the summit of arches denuded by erosion, etc., show as many traces of fossil strains. Would it be wise to deny the possibility of a fossil pressure where conditions for its formation and preservation seem to have been the best? If there has ever been such a thing as deformation through diastrophism, strains have been the result; and the gases inclosed in the strata, as well as the liquids in relation with them, having been submitted to these strains, must have acted as accumulators where favorable conditions obtained. They must show traces of these strains today, unless these traces have been dissipated since. But if they had been dissipated, would not the pressure due to gas formation have been dissipated in the same way? Further, the building up of rock pressure within the reservoir by the change of some oil into gas, seems to meet with a serious objection. Washburne has pointed out that crude oil is exothermic, and therefore has no tendency to split or decompose under its temperature of dissociation¹⁹ ("Cracking point," from 250° to 400° C. and above); and the possibility of such a temperature could not be considered, as a rule. Thus, if oil migration is due primarily to rock deformation, it becomes difficult to escape the conclusion that rock pressure in an oil or gas pool is, at least partially, due to deformation itself.

The writer thinks that he has answered the different objections raised by his critics and shown that the items criticized agree with the diastrophic theory instead of being in opposition to it. But he does not intend to say that the theory cannot be amended or perfected. If the principles on which the theory is founded should be recognized as true, some definite and more complete form may be reached as a result of "team work," in which critics and criticisms may play their part.

¹⁹ *Trans.* (1915), 51, 607.

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the St. Louis meeting, October, 1917, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Dec. 1, 1917. Any discussion offered thereafter should preferably be in the form of a new paper.

The History and Legal Phases of the Smoke Problem*

BY LIGON JOHNSON, NEW YORK, N. Y.

(St. Louis Meeting, October, 1917)

ONLY the acute phase of the smelter fume problem is new. The problem itself is older than the Christian era.

While both lead and copper were mined and crudely smelted some 3,000 years ago, it was not until the Roman occupation of the Iberian Peninsula and the British Isles, which occurred but shortly before the beginning of the Christian era, that there was any evidence of smelting operations on a scale sufficiently large to permit a fume problem.

Around Huelva, Spain, are found more than 30 million tons of slag from lead smelting conducted there by the Romans. Pliny tells us that more than 20,000 slaves were employed in the Iberian mines. Extensive mining and smelting by the Romans in England and Wales were coincident with the Iberian proceedings.

These metallurgical operations were upon a sufficiently large scale to produce marked results both upon the surrounding country and the smelter workers, but, as they were those of a conqueror upon conquered soil, conducted by slaves, imperial Rome failed to recognize that such a thing as a smoke problem did or could arise, and whatever fume question there may have been at that time remained a question only to those who had no chance to answer any phase of it.

The smelting operations of the Romans extended over about 400 years, and little is recorded of lead and copper smelting from that time until the 16th century. From the revival beginning at this period up to the present century, the growth of smelting has been comparatively gradual. In Great Britain, smelting was conducted almost wholly in localities where metallurgical operations were of paramount importance, and the communities that grew up in smelter localities were due to and dependent upon the mines and smelters. This fact had much to do with the comparative freedom of British smelting from burdensome fume litigation and legislation. To a lesser degree these conditions applied to the German operations which, when coupled with the further fact that most of the early German operations were to a greater or less degree fiscal workings of Prussia, Saxony and Brunswick-Hanover, accounts for

*Originally presented at a meeting of the New York section, Jan. 26, 1917.

superior to the German, and ultimately a strong legislative barrier was erected around German operations, particularly in the case of new smelters and under new locations of old plants.

Nowadays, German smelting is, or at least was, prior to war conditions, under supervision of special officials in the nature of mining police and courts. Before construction of a smelter could be begun, application for a permit had to be made, accompanied by general data as to location, character, capacity, height of stack, character of ore to be treated and the like. Where a permit was once granted, it was not revocable, but notwithstanding the permit, if damage was done, the smelter could be required to change its methods and to install such appliances as would, so far as possible, prevent injury. Where a permit was refused, no smelter could be built. The refusal of a permit by the Bergpolizei, or mining police, was not final, but an appeal could be had to the Berg Gerict, or Mines Court, and finally to the Ober Berg Gerict, or Mines Court of Appeals, which was the tribunal of last resort. Incidentally, the Bergpolizei had inquisitorial or supervisory powers not only as to damage to vegetation but also as to the safety of the works and the life and health of the employees.

It is supposed by many that the British Alkali Act was aimed at the smelting industry. This is far from correct. The original acts were directed chiefly at the manufacture of soda, acids and ammonia. In 1861, the House of Lords provided for an inquiry "into the injuries resulting from noxious vapors evolved from certain manufacturing processes, and into the state of the law relating thereto." While the investigation covered smelting as well as the other enterprises, the resulting act (the Alkali Act of 1863) covered only so-called alkali works and provided for the elimination of not less than 95 per cent. of the hydrochloric acid produced in alkali processes. By an amendment of 1874, it was enacted that not more than $\frac{1}{2}$ grain of acid should be contained in each cubic foot of gas escaping from the works.

The original act was made more comprehensive by amendments and revisions of 1874, 1881 and 1892 but, other than operations under the wet copper process and certain zinc processes, it was not until 1906 that smelting generally was included; and, even then, no actual regulation was provided, as the act so far as smelting was concerned merely declared that the inspectors under the act "may inquire whether in any works in which sulphide ores are calcined or smelted, means can be adopted at a reasonable expense for preventing the discharge from the furnaces or chimneys of such works into the atmosphere of any noxious or offensive gas evolved in such works, or for rendering such gas when discharged harmless or inoffensive" (Act 1905, Sec. 8 (1)).

No limitation was placed on smelting operations beyond inquiring as to the installation of such remedial methods or appliances as might be installed at a reasonable expense. There was no prohibition as to tonnage, and the first report of the Inspectors touching on smelting was in the 44th annual report of the department for the year 1907, 45 years after the passage of the original act, and this embodies no smelter regulations.

Fume controversies and the attendant litigation and legislation are of comparatively recent origin in the United States. Probably the first thing in the way of legislation in this connection was the ordinance of Oakland, Cal., enacted in 1872, prohibiting the erection or maintenance of any smelter within the corporate limits of that city. Subsequent California legislation was that of Contra Costa County limiting fume strength and by the State providing for a smelter commission to investigate the effect of fumes liberated in smelter operations. Efforts were also made to prohibit by legislative enactment the elimination of stack gases containing more than 0.2 per cent. sulphurous content. This proposed legislation, however, was defeated. There has been little legislation, either by the States or municipalities of the United States. Not until heap roasting of ores rich in sulphur was practised on an extensive scale, was there any litigation of consequence in this country, and to the condition resulting from heap roasting may be charged no small amount of the prejudice by the farmers against smelters and belief by them of the complete destruction of vegetation, animals and soil vitality through smelter operations.

The basis of this prejudice and belief is easily apparent, even to this day, in a visit to Butte, Mont., Shasta County, California, Ducktown, Tenn., and places where heap roasting was extensively practised.

Under heap roasting, strong fumes, in dense volume, were liberated directly on the surface of the earth. These fumes were heavier than air and cold, being practically of atmospheric temperature. They did not float off in the atmosphere but hugged the ground, rolling along in front of the wind in constant volume until absorbed by the soil and vegetation. There was little diffusion and the radius of fume influence gradually widened with the destruction of each successive barrier of vegetation.

The topography of the country surrounding the roast heaps was, almost without exception, sharply declivitous, as the roasting operations were conducted in hilly and mountainous regions. The soils of the regions surrounding the smelter sites were coarse and grainy, requiring the binder of the roots of vegetation and humus to hold the soil particles together.

The destruction of plant life adjacent to heap-roasting operations resulted not only in the loss of vegetal life above the soil but of the roots below, and, with every rain, the top soil was eroded until large areas were

tion could find a foothold, even when the fumes were no longer present. It was from this erosion that the belief of the killing of the soil first arose.

From what they could already see, the people around the smelters believed the smelter smoke to be the veritable breath of the Upas tree which destroyed all within its reach, and that this deadly agency was daily reaching farther and farther from the smelter and in and upon the homes and farms surrounding the smelter sites. Litigation was then not only threatened but instituted. First, the farmers sued the Tennessee and Ducktown companies, and upon the failure of these suits the State of Georgia took up the cudgel. The farmers in Shasta County, California, brought suit against the plants at Kennett and Coram; the Benecia residents and adjacent farmers sued the Selby smelter; the United States, by reason of the ownership of a forest reserve adjoining the Keswick plant, filed its bill against the Mountain Copper Co. Not long after this, the Deer Lodge Valley farmers filed their suits against the Washoe Co. at Anaconda, and Salt Lake valley farmers began proceedings against the smelter operations in that valley. The epidemic of smelter litigation was under way.

With these suits, developed the early so-called smoke experts. Most of them had been in the heap-roasting communities where the destruction of vegetation and soil erosion had followed the heap roasting. No definite knowledge of fume action was available, and the experts for both plaintiffs and defendants, in the absence of knowledge, began to form pet theories only to be met by pet theories of others directly at variance.

In the good old days of theory, an expert could make a casual inspection, or examine a few cross-sections of leaves under a microscope, or make a few comparative tests for the sulphur contents of healthy and unhealthy plants, and deliver an epic on what the fumes were and were not doing. The appearance of the country around the old roast heaps and early plants was too big a handicap for the smelters to overcome, and, with the notable exception of the Washoe litigation, in practically every case an injunction followed.

About the time of the institution of these suits, the smelters began the abandonment of heap roasting and the development of present day operations. In this development, prior theories expounded by the then experts or testified in the smelter suits had considerable bearing. In one locality everyone was assured that the injury came wholly from the dust particles; in another, from the dust particles as the nucleus for minute drops of acid; in another, from sulphuric acid vapor, which could be seen as a white fume cloud, and practically everything was blamed more than the chief, if not sole offender, the SO_2 . In many cases the courts followed these theories in their decrees.

The first development went to long settling flues, smoke houses and dust chambers, cooling and slowing down gases, and taking out so far as possible the solids and acid vapors. At the Tennessee plants, acid making was attempted.

In most of the litigation, under readjustments to conform with decrees entered or modifications of these decrees—which modifications, as a rule, were secured under very substantial cash payments to the farmers—the smelters secured a new lease on life. Several, however, such as the Balakalala plant in California and the Highland Boy at Salt Lake, were completely closed, and passed into history as smelters. Most of the new installations, methods and readjustments provided by the decrees were in conformity with the theories of the experts in that particular case.

It is interesting to note that not only were the findings of fact in the different cases widely at variance, but the conclusions of law in many instances directly in conflict, even in cases in different circuits of the United States courts. For instance, in California and Montana, the United States court held that the court could consider the balancing of conveniences, that is that the court, in determining whether or not an injunction would be granted, could weigh the damage that would be done under closing a smelter against the benefits the farmers would receive under an injunction. In the federal court in Utah a directly contrary holding was made. In the findings of neither law nor facts did the courts coincide.

Some little time was required to make plant changes to conform with the theories of the experts adopted by the court or the farmers, and still further time was required to demonstrate the effectiveness of the changes. Most of the theories failed to pan out, and gradually a new crop of complaints, and claims that the recent installation did not prevent the damage, began to grow. Some claimants were insistent. Plant managers, remembering the results of old litigation, sought to temporize, instead of ascertaining the real facts and meeting them squarely. Some tried to buy peace.

Let me say just here that nothing in the Selby report is truer than the statement that (p. 14) "The policy that 'buys off' trouble, as the most expedient commercial method for abating it, has been responsible for much of the smelter litigation of this country and the intense ill feeling that unfortunately exists toward smelters in many smelter communities." It did not take long under this practice for the price of peace to rise to prohibitive figures and a new epidemic of litigation was threatened.

Going back a little beyond this second period of threatened litigation and before the storm clouds of it were yet clearly over the horizon, I think that I may say that the smoke problem was then considered by most plant managers as the least of their troubles. Little or no thought

or study was given the question, and when it was mentioned it was waved aside. The worst most of them feared was having to judiciously distribute a little peace money.

A number of the smelters, such as the Washoe, Mammoth, Bully Hill, the old Mountain Copper, the proposed Engles smelter and others, were either in or adjacent to National Forest Reserves and numerous reports of damage, or threatened damage, began to come into the Chief of the Forest Service and the Land Office. These reports were in turn referred to the Department of Justice. I was at that time Special Assistant to the Attorney General and these reports were finally referred to me with instructions to prepare and file bills for injunction unless some satisfactory solution could be reached.

While it was probably unknown to the smelters generally at that time, it was never the intention of the Government to close any plant. What it sought was the immediate and serious consideration of the fume problem and active effort to correct harmful conditions where such existed. Only one suit was filed, and that was against the Anaconda plant. In each of the other cases stipulations were entered providing for research work and the installation of the highest types of methods and appliances known to smelting science or else the abatement of the operations producing fumes in harmful quantities until such appliances had been installed and proven elsewhere, upon which they were to be installed at the smelter entering into the stipulation. In the Anaconda case the first scientific commission was agreed upon. We proposed that the smelter operate under the best type of methods and appliances known to science and commercially feasible at the plant and that a commission of John Hays Hammond, Dr. L. D. Ricketts and Dr. Joseph A. Holmes, Director of the Bureau of Mines, be designated as a commission to investigate and prescribe what changes or installations should be made under the agreement.

It was not long after this that the threat of further litigation by the farmers became ominous, and, as I had completed my work with the Department of Justice, so far as smelting matters were concerned, I resigned to become what might be termed smoke or field counsel for the American Smelting & Refining Co.

The first urgent smoke matter in this connection was the Selby litigation between Benecia residents and the Selby plant, which had been pending a dozen or more years. A flat injunction against the operation of the Selby plant had been granted, which injunction had been fought through the courts and finally approved by the Supreme Court. The case had been through the court of last resort and the injunction was to become effective in a month or 6 weeks' time.

On examining the record, I found this suit to be somewhat different from the average smelter suit. Very little claim of injury to vegetation was made. More stress was placed on injury to animals, but the chief

ground of complaint was excessive discomfort and alleged nausea and illness produced by odors which the witnesses uniformly declared "to smell like rotten eggs" or the sulphur spring of an adjoining county which gave off quantities of sulphureted hydrogen.

An examination of the Selby plant disclosed that, regardless of what lead loss there might have been at other times, under changes made or under way no lead loss would occur and therefore no possible source of injury to animals could exist from the plant's operations and that there was no appreciable amount of sulphureted hydrogen given off by or generated in the smelter plant.

In the San Francisco region, in the spring and summer months constant trade winds blow, and these winds follow a direct line from the Selby smelter to Benecia where most of the complaining witnesses lived. Beyond the Selby plant, but also directly in line of the trade winds over the Selby plant, was a large oil refinery and asphalt works. The odors complained of by the witnesses all came in the early morning hours, and investigation disclosed that at the times involved in the testimony, the oil and asphalt stills were not capped and were poured at three or four o'clock in the morning. Here, then, was a source of odors which traveled on the same winds that carried the Selby smoke. The smelter was guilty of none of the things upon which the injunction was based. Capping the oil and asphalt stills practically disposed of the unpleasant smells.

We notified the County Attorney and Commissioners of Solano County of these facts and said that we proposed to decline to observe the injunction prohibiting the operation of the plant. This would have meant additional long drawn out and expensive litigation. To obviate such a situation, I proposed that the whole matter be referred to a commission of scientists of the highest type, the members to be agreed upon jointly, and the finding to be entered in the Selby case as the findings and decree in that case. Every access to the plant and fullest facility for examination was to be given the commission. After many conferences, an agreement along these lines was arrived at and the Selby Commission came into being. The members of the Commission were Dr. J. A. Holmes, Director of the Bureau of Mines, Dr. E. C. Franklin, then Director of the Chemical Laboratories of the U. S. Public Health Service at Washington, and Ralph A. Gould, a chemical engineer of San Francisco. With the Selby Commission came the first scientific research in smelter fumes along the lines of normal field conditions.

The examination of the Selby Commission extended over a period of about a year and a half and in the end clearly demonstrated that the smelter was doing none of the things found against it in the original decree; and the injunction was vacated.

Shortly after the time the Selby Commission began on its work, the murmuring of discontented farmers became much louder. Crops had

failed in many localities and wherever this crop shortage was in a smelter region the smelter was blamed. This was particularly true where prior payments had been made to farmers. Peace was quoted at war prices.

As I before stated, this was very shortly after the Selby Commission began its work. Other than the research just beginning under the Selby Commission, there had not been in the United States or elsewhere any experiments or research work with smelter fumes or SO_2 under natural field conditions. It is true that Haselhoff and Lindau, Von Schroeder and Reuss, E. Schroeter, Wieler and Hartlieb, R. Hartig, Wislicenus, Freitag, Sorauer, Ramann, Gerlach, Schmitz-Dumont, Sabachnikoff, Raubner, and Stockhardt in Europe, and Haywood and Pierce and, to to a limited extent, Ebaugh and Talmage, in the United States, had carried on some experiments, but in none of these were normal field conditions approximated. Some fumigations were carried on with leaves or twigs or small parts of plants in bell jars. Others were in hermetically sealed cabinets or "smoke houses" so constructed that only abnormal conditions could result. The plants used were grown in pots or flats and were grown and kept under conditions that were not normal. The fumes introduced were from burning sulphur or sulphur and alcohol, or else concentrated SO_2 let in from a tank or gas burette. The concentrations were unknown, absorption was not considered nor were chemical analyses of the air made. Environmental factors of temperature, humidity, light values, barometric pressure and time element of exposure were not considered. The plants used in the experiments were not grown to maturity, nor were their food or crop values ascertained. As a matter of fact, at the time these experiments were conducted, no quick and accurate method of air determination was even known to the experimenters. Even had these investigations been conducted with normal plants, under natural field conditions, the lack of knowledge or records of actual fume strengths reaching the plants and of the environmental factors involved would have rendered the experiments practically valueless.

Some investigators of smelter regions placed much reliance on the sulphur content of plants collected in smelter neighborhoods, notwithstanding the fact that investigations in the Department of Agriculture have shown that the sulphur content of the same plant may vary as much as 3,000 per cent. at different times during the season and at the same period the content of the same species of plant will vary markedly, particularly under different percentages of soluble sulphur in the soils, which frequently occurs at short distances.

In other words, from farmers and their experts the smelters were coming in for full blame for all crop failures and the managements actually did not know whether or not they were doing the damage. From my past experience, I knew that there were numerous plant injuries attributed to smelter smoke which even the most competent expert could not differ-

entiate by mere observations or even microscopic study. Numberless diseases and injuries which were not caused by smelter smoke were pointed out as evidence of smoke damage.

No plant manager knew how much, if any, of the damage attributed to the smelter was caused by it, and if there was damage, what operation of the smelter or element of waste had produced the damage. And, too, with the payment of damages communities of smoke farmers grew up. Their idea of farming was to let their places grow up in weeds and collect from the smelter the value of the maximum crops ever produced. Many smelters had paid damages for conditions which, if later developed, were not remotely attributable to smelter operations; but the farmers once having been paid insisted on continued payments. The price of peace was rising above the possible profits of smelter operation.

The time for theorizing had passed. It was necessary that the smelter managements should know just what damage was being done, and if there was damage, how it was done and what was necessary to prevent it. Under my urgent recommendation, the first research department to ascertain all this was installed and, in conjunction with the Selby Commission, it blazed the way for the smoke engineering.

This research work involved chemistry, plant pathology, plant physiology, entomology, agronomy, dairy husbandry and veterinary toxicology and meteorology.

You may wonder why all this was necessary just to find out whether or not the smelter was doing any of the things claimed, and why one or two experts familiar with the appearance of vegetation in smelter regions would not have sufficed.

We had to know first what smelter fume or parts of fume would do damage; how the injury could be caused; what conditions were or could be confused with smoke injury, and finally, where damage was or could be done, what steps were necessary to obviate it.

This meant extensive research work under normal field conditions to ascertain the effect of smelter eliminations upon vegetation and animal life. The time at hand is too short to go into minute description of this research work. It will probably suffice to say that we found that dust and acid vapors were practically negligible quantities in the fume problem so far as vegetation was concerned. Where damage was done, this could be traced almost if not wholly to sulphur dioxide. Treating plants with the amount of solids that would be deposited on vegetation under general smelting operations produced no result. The average vegetation can stand fifty times the strength of acid vapor that it can resist where the sulphur is administered in the form of SO_2 . In the ordinary smelter fume the dust and acid vapor are but a small fraction of the sulphur content, the latter being chiefly confined to sulphur dioxide.

It then became necessary to ascertain the relative resistance of the various plants to sulphur dioxide and the conditions under which injury occurred when the SO_2 was present in sufficient quantities to do damage.

To develop this fact it became necessary to know exactly how smelter fume acted upon plant life, how it obtained ingress to the plant structure and its effect after its absorption in the plant.

It may not be amiss just here to describe in a few words how this ingress comes about and the results which follow. In the beginning we found that only during the period that the plant was in leaf could injury occur. Subjecting a plant to SO_2 or other fume while not in leaf produced no result. In most instances, and practically without exception in alkaline soils, we found that treating the soil with sulphur or sulphuric acid—that is, dusting or spreading crude sulphur upon the soil or spraying the soil with acid—resulted in increased crop yield. Sulphur administered in this way was a benefit and not an injury. This conclusion was verified by independent experiments conducted at the University of California in Berkeley, at an experiment station in Oregon and at the experimental farm of the Anaconda Copper Co.

The question then narrowed down to the effect of smelter fume through leaves and on leaf structure and to present this clearly it is necessary to describe the ordinary leaf and its functions.

All leaves under normal field conditions, in climates such as we have to deal with here, have an epidermis covering both under and upper leaf surface. This epidermis is impervious to moisture and for all practical purposes, except where stomata, or breathing pores, occur, is also impervious to gases. These breathing pores are so small that considerable magnification under a microscope is necessary to see them at all. The stomatal openings are entirely too small to admit the minutest suspended solids, water globules or particles of mist or vapor. The stomata at the leaf surface are faced with two guard cells, that are also covered by practically impervious epidermis. These guard cells, under certain leaf functions open or close the stomatic chamber. With the guard cells closed, the leaf under ordinary conditions presents impervious surfaces to the elements. The inner content of the leaf, or the mesophyle, is composed of palisade cells, collecting cells and sponge parenchyma, in which, under the action of light, starch and sugar are formed and plant food manufactured and supplied to the plant.

We next found that under conditions of absolute closure of the stomata the plant was a number of times more resistant to SO_2 than when the stomata were open. Under all ordinary conditions the stomata form the points of ingress of gas. It followed, therefore, that where an external condition brought about the closure of the stomata it brought about increased resistance, and we finally worked out four factors in chief in fume injury, where sulphur dioxide is present in strengths sufficient at any

time to bleach plant life, these factors being light, humidity, temperature, and constant wind direction. The bearing of the latter comes from the fact that injury, except from abnormally strong fumes, does not come from mere fume puffs but from steady application of SO_2 for several hours. The other three factors bear upon stomatal opening.

After we had learned the strength of SO_2 necessary to produce markings upon plant life under field conditions, it became necessary to know whether or not SO_2 was present in such quantities in the field. It was also necessary to know accurately the wind constancy and direction, the limits of the smoke stream, temperature, humidity, and general weather data. Complete installation of standard meteorological instruments, under the supervision of the local U. S. Inspectors was made; and portable laboratories, that is, laboratories set up in small automobiles which could follow the smoke stream as well as make general tests, were secured. This was in addition to the fixed air stations maintained. Under the method of air analyses devised by J. R. Marston and later refined by A. E. Wells, chief chemist for the Selby Commission, quick and accurate determination of air samples was possible, and, more important still, these samples could be taken in a few seconds time, and, in this way, the constancy or inconstancy of the SO_2 determined. For the smelter plants, we cross-sectioned the several flues at the base of the outlet stacks, and the stacks, to afford analysis of their contents, and installed self-recording thermometers at appropriate points in the flues and stacks for accurate and complete data as to gas temperatures.

With this knowledge, the next step was to ascertain with scientific exactness the cause of crop failure or injury where the appearance of external conditions, and often even microscopic cross-sections, were seemingly identical with the results of smoke injury, but where our other investigations had developed that smoke injury could not have occurred. There are many diseases, pathological conditions and insect injuries which, both externally and under the microscope, present an appearance practically identical with that of smoke injury. With the completion of this investigation, a survey of the entire so-called fume zone followed.

It is interesting to note that, in determining the injury of plants by disease, we can establish our diagnosis with scientific exactness not possible in the diagnosis of any disease of human beings or animals. We can establish our diagnosis beyond the possibility of doubt or question. This is done by what is known as the pure culture method. The unhealthy plant is pricked by a needle-pointed instrument which is then dipped into a special and pure plant jelly, and a culture of the disease organism is grown. The disease organism is isolated and reproduced. A healthy plant is then inoculated with this culture and, after the disease develops, comparison of appearances is made; a second culture is made from the inoculated plant and a second inoculation of a healthy plant follows. In

Where culture media are impractical, resort to direct inoculation is not only possible but convincing in its results. For insect causes, we find and collect the causative insect. As some insect colonies, although numbering thousands or even millions, are so small that the individuals are only observable under very high-powered microscopes, and others work wholly out of sight and beneath the soil, it requires a competent entomologist to handle this phase.

It will suffice to say here that we proceeded through the whole list of plant pathology, physiology, entomology, agronomy and dairy husbandry and veterinary toxicology with equal care and certainty. The time is too short to go into detailed explanation of the plan pursued under each. Great credit is due to P. J. O'Gara, chief expert in charge, and E. P. Fleming, chief chemist, for the perfection of means for and methods of the scientific research done as well as for the results accomplished.

One other agency in smoke determination has been established with relation to plant life, and that is the evidence of so-called guide plants. There are certain plants, such as barley, which are especially susceptible to fume injury and will show bleaching long before the possibility of injury to other plants. Where the guide plants are unmarked, it is self-evident that the trouble with adjoining hardier plants is not smelter smoke.

We also carried crops to maturity after producing severe burns on some and repeated burns on others, with astonishing results as to crop yield. We found the real damage from the average burn more imaginary than actual. We ascertained the protein content of bleached plants, finding it as high or higher than in the unbleached. We proved food values by actual feeding. In fact, we covered every phase we could find.

In the work I have mentioned, we not only secured information valuable to the smelters but we accomplished actual and beneficial conservation so far as the agricultural resources of the country were concerned.

Scientific farming is not yet widely practised. Farming methods are often handed down from father to son. Year after year the same crops are planted, often on the same soil. This applies to farming generally and not merely to smelter communities.

In many places not only do the soils become impregnated with disease-producing fungus and germs, but the result becomes so widespread that no healthy seed can be secured in the community. It often happens that disease conditions may largely be eradicated by proper treatment and planting of the seed and crop cultivation, but these facts, as well as that the seed is diseased, are unknown to the farmer.

I recall a case where enormous claims for loss of potato crops were

filed against a smelter. Upon investigation we found both the potatoes and the soil of the region highly infected with two of the most disastrous of potato troubles, rhizoctonia and fusarium. A subsequent search of the seed stores in the community failed to disclose a single healthy sample of seed. Naturally the potato crops were failures. The sad part of it was that there was no need of this loss. Healthy seed procured elsewhere and planted in soil where potatoes had not been grown for several years, would have given the old-time record crops, and proper treatment of the local seed would have produced infinitely larger returns.

Most of the diseases and insect conditions of agricultural communities are largely preventable, and for those that are not steps may be taken to greatly reduce the harmful effect. Much of the trouble can be directly pointed out to the farmer, and the means of eradicating or lessening these troubles shown to him. This, of course, involves the demonstration that the unhealthy crop conditions are not attributable to the smelter.

The farmer who follows the suggestions for eradicating unhealthy conditions soon finds his crops exceeding those of his neighbors who have failed to observe the same precautions. In this way, gradually, it is true, the influence of the smelter's experts is felt, a friendlier feeling is engendered, agricultural conditions are improved and actual conservation accomplished.

This brings us down to one final theory which in some places is still firmly entrenched and threatens possible trouble. I refer to visual clearance and the belief that anything seen coming from a smelter stack should be labeled with a skull and cross bones. Our investigations showed that the visible fume was little to be feared as an actual instrument of damage to vegetation. Under former operations, where a good part of the lead content of the ore was blown up the stack and where heavy arsenic fumes were given off, this metallic loss had some connection with certain animal troubles, but had little effect on vegetation. Where this character of loss was not involved, the visible fume was and is chiefly dangerous from the psychological point of view. By this I mean that most farmers (and many will be honest in their belief) will assert, so long as there are visible stack eliminations, that every crop failure is due to the smelter fumes. Therefore, to meet this, it will be advantageous to obtain the greatest possible clearance up to the point—and I want to stress this especially—where the efforts to secure visible clearance renders the smelter operations more liable to do actual damage. Beyond this point it is better to try to educate the farmer and get rid of the belief. Just because a farmer sees dense volumes of soft coal smoke pouring from the stacks of manufacturing plants, business houses, engines and the like, he feels no uneasiness about his crop. There is little, if any, more danger to crops in the visible smelter smoke. And, too, even where the belief of damage is deep seated, mere belief can be met by facts, but

in the face of actual damage.

Get clearance first for actual recoveries, and to prevent any excessive loss of toxic solids, and next, so far as possible, to meet any psychological conditions, but do not place visual clearance above the likelihood of actual damage. As I have previously stated, the visible part of smelter fume is not the harmful agent.

Our investigation and the temperature curves we have worked out have shown us that where there is fume injury the answer to the problem is in hot gases, gas dilution, and high stacks. Anything that cools or slows down the gases, or results in their liberation at low altitudes, invites, to say the least, the probability of actual damage. Promising a community complete visual clearance, or encouraging the belief that visual clearance means impossibility of damage, is borrowing trouble for the future, for such a clearance, unless I am mistaken, means low gas temperatures.

These smoke problems are bringing to the front a new specialist, the smoke engineer. The far-sighted smelter management, under existing conditions, for all new constructions, and even alterations and additions to old plants, will find this new expert a necessity rather than a luxury. The best way to take care of trouble is to prevent its starting. Many of the smelters realize this and welcome the coming of the smoke engineer and specialist.

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